A mining research contract report
for the Abandoned Mine Lands Program
JANUARY, 1991

BLASTING FOR ABANDONED MINE LAND RECLAMATION

(Closure of individual subsidence features and erratic, undocumented underground coal mine workings)

Contract J0289004
Calder & Workman, Inc.

BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

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16. Abstract (Limit: 200 words)

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At the White site in Montana erratic underground rooms with no available documentation were collapsed. An adit leading into the mine was also blasted. Both individual room blasting and area pattern blasting were studied. A total of eight blasts were fired on the one acre area. Exploration requirements and costs were found to be extensive.

Field procedures and methods are reported. High speed camera films and seismograph readings were taken for most blasts. Results of the test programs are analyzed. Conclusions are provided concerning the technical and cost feasibility of blasting for AML reclamation.

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b. Identifiers/Open-Ended Terms

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c. COSATI Field/Group

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FOREWORD

Calder & Workman, Inc. of 206 8th Street, Washburn, North Dakota has prepared this report in fulfillment of United States Department of Mines contract J0289004. Mr. Larry Fletcher of the Twin Cities Research Center was the Technical Project Officer for the Bureau of Mines. Mr. Larry Rock was the Contract Specialist.

The report contains the results and conclusions concerning all work completed under this contract. The work was performed between May 1988 and January 1991 and the report was submitted in January 1991.

Sincere thanks are extended to the North Dakota Game and Fish Department for permission to use the site near Beulah and to the State of Montana for permission to perform research at the White Site near Scobey, Montana. We also thank the North Dakota Public Service Commission, AML Division, for their interest and support of the project. Without the cooperation of these states the research could not have been performed.

Thanks are due to Larry Fletcher of the Bureau of Mines for his interest in the project and for the helpful guidance he has provided throughout. We also wish to thank the subcontractors and consultants who worked with us during the research. Their dedication and hard work were instrumental to the success of the project.

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CHAPTER 1.0 INTRODUCTION

1.1 GENERAL DISCUSSION

In May of 1988 Calder & Workman, Inc. was contracted by the United States Bureau of Mines to perform a research contract pertaining to the study of blasting techniques applied to the reclamation of Abandoned Mine Lands (AML). The research included field testing at two abandoned mine sites; one in North Dakota and one in North Eastern Montana. These locations provided good opportunity to study the use of blasting to close individual subsidence features and to cave in very irregular underground workings.

The field work in North Dakota commenced in August 1988 and the field work in Montana was performed during September 1988.

A variety of methods are used, throughout the United States to mitigate the problems associated with abandoned mine lands. When reclamation involves filling individual sinkholes the technique frequently used is to haul fill from a suitable source and dump this material into the hole. The cost is quite reasonable provided there is a source of fill material near the reclamation site. If the dirt used is far from the site the cost of haulage becomes increasingly significant.

Stabilizing old underground workings can be performed by one of several methods. Common among these are hydraulic and pneumatic backfill. Provided the location of the old workings can be accurately established these methods work well, but are often costly.

Another approach is to daylight the old rooms and crosscuts using excavating equipment such as small draglines. The cost of this technique will depend on the depth to the main entries.

In general, blasting has not been used for reclaiming abandoned mines. Reports of such work in the literature are limited. (1, 2, 3, 4). There are several possible reasons for this.

1. Lack of familiarity with blasting techniques and procedures by those involved in AML work.

2. Concern that blasting may cause damage to buildings in the vicinity either from ground vibrations or airblast.

3. Concern that residents in the area where blasting is conducted will object to ground vibration and airblast.

4. Concern that flyrock from blasting operations will pose a hazard.

5. Uncertainty whether a blasting program would provide a successful mitigation of AML hazards and, especially, doubt as to how the results can be evaluated.

The authors have thought that blasting techniques can play an important role in AML reclamation. Basic technology, from which to develop a blasting program, exists. Often, emergency situations can be responded to quickly by utilizing blasting methods. The costs were expected to be competitive.

Therefore, Calder & Workman, Inc. (Bauer, Calder & Workman, Inc. at that time) performed previous research including field testing for the Abandoned Mine Land Section of the Office of Surface Mining (4). This research was done in North Dakota on land owned by the North Dakota Game and Fish Department and adjacent to the area where part of the current test work has been performed. The work concentrated on the complete collapse of abandoned underground workings that were regular in nature and for which good documentation in the form of mine maps was available

The study confirmed that blasting techniques could successfully collapse the workings while, at the same time, suitably addressing other concerns associated with blasting. Also, costs proved to be competitive.

The current study has expanded upon the prior research. In this case individual sinkholes have been closed. In addition, a small underground operation, consisting of an adit and approximately one acre of very irregular workings, for which there were no accurate mine maps, has been tested. Both test sites have allowed available knowledge to be expanded to include the use of blasting for common but potentially difficult projects.

In general, blasting will not prove suitable when AML problems occur in very close proximity to residential and business structures. For example, if a subsidence feature opens up in a homeowner's back yard blasting is unlikely to be a suitable mitigation approach. However, when reclamation needs are five hundred feet or more from structures the method can be considered. Of course, in remote areas there is little concern about the effects of blasting on buildings.

Occasionally it is desired to perform reclamation in places where it is difficult to access the property. Mountainous areas would be an example. Then blasting could be a very good technique since the method may be able to be utilized with limited equipment needs and transport by helicopter, for example, may be feasible.

Therefore, for the majority of AML projects, blasting should be considered as an option. The purpose of the research reported here is to determine the likely success and cost of reclaiming specific types of AML features and to provide information from which AML managers and contractors can develop such programs.

1.2 THE SITES

Two sites were chosen for field test work. One was in central North Dakota near the town of Beulah. The location is shown on the map in figure 1.1. The second site was in North Eastern Montana, about 14 miles north of Scobey in Daniels County and quite close to the Canadian border. The site location can be seen in figure 1.2.

For purposes of identification the test site in North Dakota has been designated the Beulah Site. The location in Montana was called the White Site.

1.2.1 The Beulah Site

At Beulah the research performed was solely for the purpose of determining methods for closing individual subsidence features. The property is owned by the Game and Fish Department and is adjacent to Game and Fish land upon which previous studies had been conducted.

The Beulah Site is only a short distance east of Beulah, North Dakota. It is located directly off highway 200A. Therefore the site was easily accessed and serviced.

The Beulah Site was selected for a number of important reasons. These are listed below.

- There were a substantial number of individual subsidence features of varying size and shape that could be blasted. Therefore, a good test of the ability to close individual sinkholes was assured.

- The area is underlain by many unreclaimed workings. Therefore, the site afforded the opportunity to study the effect of blasting a sinkhole on the surrounding, undermined area.

- There are houses that are not far from the site. Therefore, it was possible to observe the effects of blasting on nearby residents.

- The site is owned by the Game and Fish Department which meant that permission to perform research on the property was more easily obtained than if private land were involved.

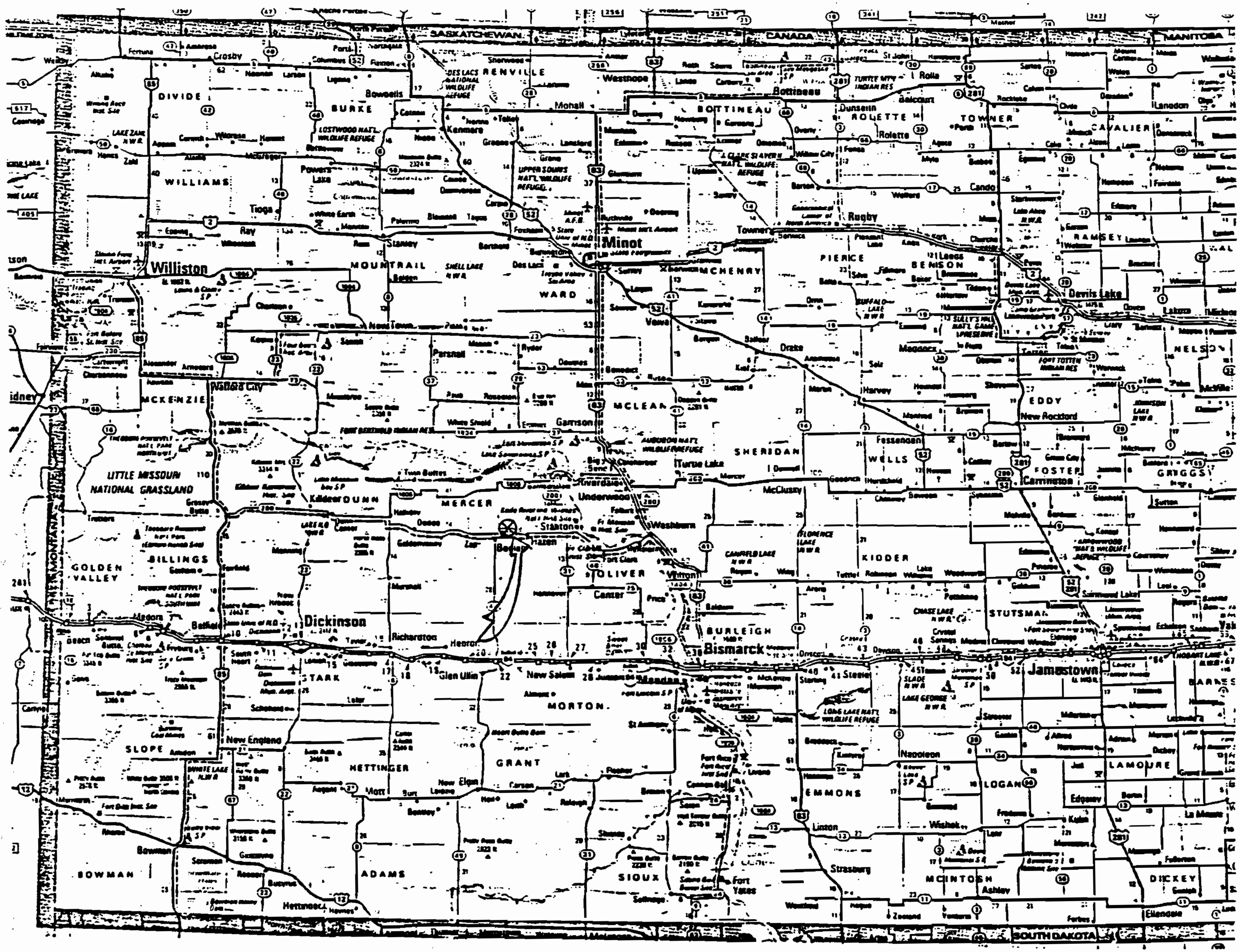


FIGURE 1-1: GENERAL LOCATION OF THE BEULAH, NORTH DAKOTA BLASTING TEST SITE

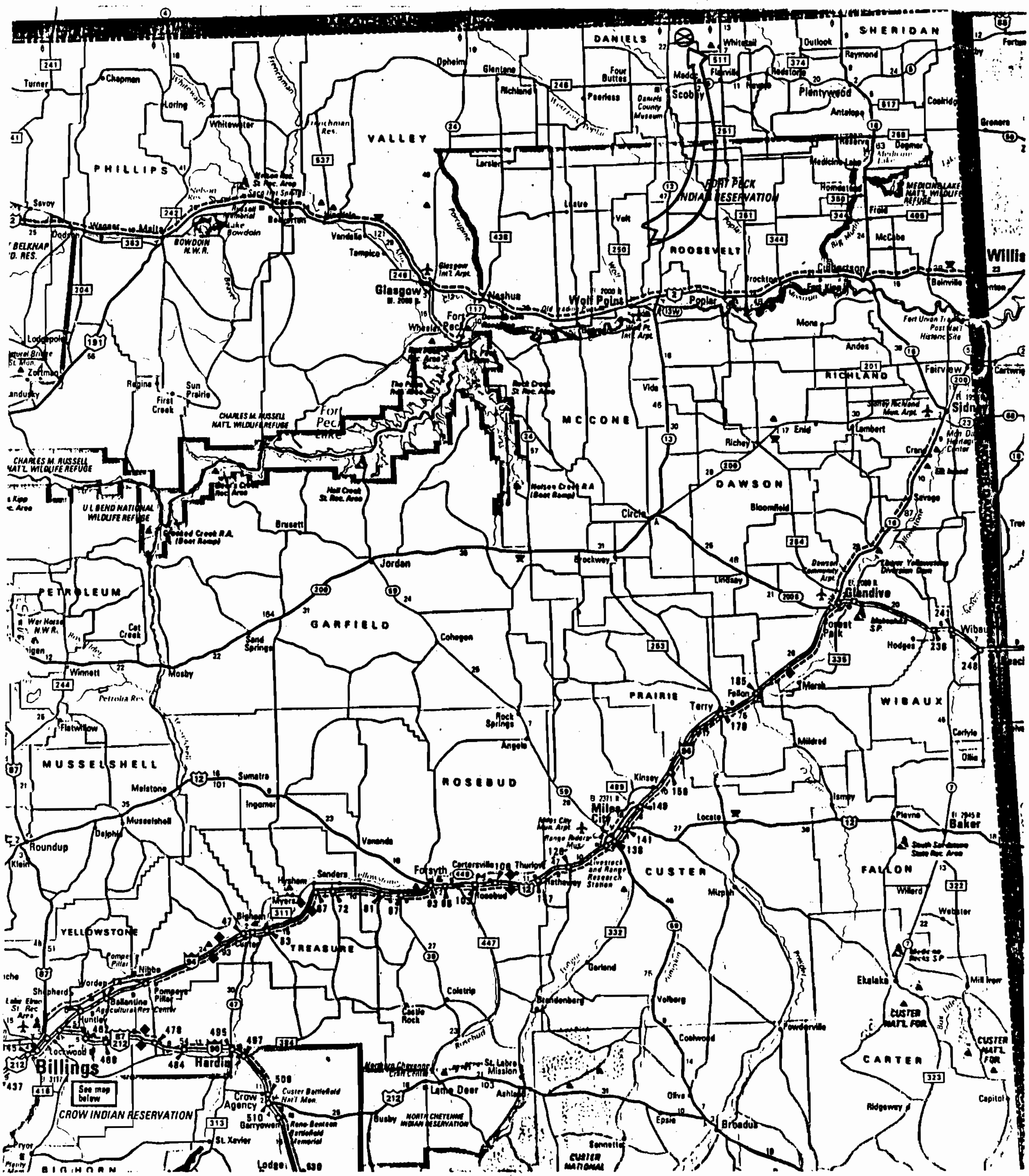


FIGURE 1-2: GENERAL LOCATION OF THE WHITE BLASTING TEST SITE NEAR SCOBEY, MONTANA

- Prior knowledge of the site made initial study and planning easier.

- The site is within reasonable driving distance of our office and therefore expenses for travel and the logistics of supplying the site were reduced.

The Beulah Site afforded a good opportunity to study the use of blasting to close single features such as sinkholes. The North Dakota Game and Fish Department was most helpful in allowing the use of the property.

The total area that has been mined at Beulah is large. For this work we conducted blasting in several areas where sinkholes existed. In total thirteen subsidence features were blasted. These features were found in four separate areas of the site. However, the geology was quite similar throughout the area.

Housing has been built along a county road west of the property. These consist of single family houses and trailer homes. The distance to the houses ranged from about 1,200 feet to 2,000 feet. Six residences were involved in all. The site provided a good opportunity to study blast vibration effects and citizen response. Blast vibration is bound to be an important issue in AML reclamation in more populated areas and the ability to obtain data pertaining to this concern is very useful for future projects.

1.2.2 The White Site

At the White Site research could be conducted into reclamation, by blasting, of very irregular underground workings. This site had been mined by a small group to provide home heating fuel and not by a large company such as was the case at Beulah. There were no mine maps that could be used to initially determine the magnitude of the mining or the location of the mined areas. Therefore, the site provided the opportunity to study the types of exploration work that could be applicable to the many similar sites existing throughout the country.

There was an adit at the site as well. This adit was not fully open to the old workings because of material that had caved into the opening over the years. It did, however, provide an opportunity to give some consideration to blasting around these features. The reasons for selecting this site are listed below:

- It provided the opportunity to perform research on the blasting of irregular mine workings for which there was little documentation.

- The site included an adit which allowed some thought to be given to the blasting of these features.

- A portion of the old workings were water filled which provided the chance to determine the effect of the presence of water on reclamation by blasting.

- The site was not large and was more manageable from a research and development viewpoint.

- The State of Montana, through the AML division of the Department of State Lands was able to provide authority to work on the property.

- The state had done some prior exploration on the property and were prepared to underwrite the cost of additional work. This included the operation of a T.V. camera device that could be lowered down the hole and used to view the direction the workings extended. Availability of the device provided the opportunity to examine the utility of exploration techniques other than just drilling.

The White Site consisted of approximately one acre of land. Given that the mining had been conducted in anything but a systematic manner, the limited size of the site meant there were manageable limits to the research while, at the same time, leaving the site in an acceptable final condition. The one-acre site size also meant that a good understanding of exploration techniques could be obtained but the exploration needs would not become so large as to cause timing or cost problems.

The White Site is more remote than the Beulah Site. Neighbors in the area were individual farmsteads and the closest were at least a mile from the blasting area. Therefore, no problem was expected from blast vibration. However, a seismograph was still used for each shot to gather additional data about ground vibration and airblast that could be used to predict the effects in future work.

In all, work at the White Site consisted of seven blasts. These ranged from shooting individual rooms to area blasts, shot on a regular pattern, that encompassed several rooms and crosscuts. Several variations in technique were evaluated.

1.3 PURPOSE OF THE STUDY

The primary purpose of this study was to ascertain if blasting could be effectively used in certain types of AML reclamation. Further, one wanted to determine whether the costs to perform such reclamation would be reasonable.

It was believed that conclusions could be best supported if actual field work were performed. This provided first-hand knowledge of success or failure. Also, design recommendations and cost expectations could be based on actual data rather than assumptions. For this reason the two sites described above were chosen for conducting actual blasting experiments. The data and experience gained form the basis of the analysis presented in this report.

There were several questions that this research was designed to answer. These are listed below:

1. To study the blasting of individual vertical openings and to assess the success rate of this technique.

2. To study the blasting of quite irregular workings and the success obtained.

3. To study the blasting of an adit and techniques by which this might be effected.

4. Determine blasting theory appropriate to the project and evaluate the success of these techniques. To calculate patterns, explosive loadings and delay timing that would be effective.

5. To assess the degree to which infilling or caving of the opening had occurred.

6. To evaluate the usefulness of explosives, explosive accessories and systems for work of this type.

7. Examine the effects of blast vibration when blasting in old works.

8. Determine whether blasting could be a successful method for reclaiming these features.

9. Determine the cost of blasting vertical openings and irregular underground workings.

10. Provide recommendations for others who may wish to use blasting in AML reclamation of these types of features as to the utility of the method and for appropriate methodology.

These considerations are all discussed in this report.

1.4 METHODOLOGY USED IN THE STUDY

The initial work was to determine methods of blasting to employ at each site that would best suit the characteristics of the area and would provide the best chance of success for the abandoned mine features being blasted. This work was done at our offices using resources such as maps, aerial photography, literature and in-house knowledge of blasting theories and techniques. A visit was made to each site to gather further data to incorporate in the preliminary work.

Once the methods and procedures had been established field work commenced. The first site studied was the Beulah location. Vertical openings were the features dealt with. Each blast was designed and staked in the field. Blastholes were drilled as marked out. Explosive loadings, millisecond delays and tie-ins were determined and the designs effected in the field. Most shots were filmed by high-speed camera and all shots were monitored by a seismograph. Appropriate photographs were taken and all data was recorded for further use.

During the field work ongoing evaluation of the results was utilized to make changes as necessary. This was especially true at the White Site where the irregular workings and lack of information led to the need to modify the program from shot to shot based on observation of the results of each blast. The T.V. camera equipment available at this site was quite useful in evaluation of the daily blasting results. Evaluation of blasting vertical openings at Beulah was easier to perform because the blasting results were not hidden by overlying strata.

Once the field work was completed all of the data was analyzed in order that the questions listed above could be answered. This included the analysis of technical factors and also the study of costs. From the data analysis procedures were developed for future use. Also estimates of the cost of reclaiming AML lands by blasting have been established. The conclusions and recommendations are based on the data obtained from the field studies and other inputs, where these are appropriate.

CHAPTER 2

BLAST DESIGNS FOR RECLAIMING VERTICAL OPENINGS AND IRREGULAR WORKINGS

There are similarities in the blasting requirements for vertical openings and irregular workings but differences as well. In the former case it is desired to throw strata from around the opening into the void thereby filling the sinkhole. Sufficient material must be cast into the void so as to fill both the void and leave the surrounding area filled as well. Therefore the rock must be displaced and swelled in order to successfully fill the area to near the original surface.

To reclaim irregular openings it will be necessary to blast material down into the workings to fill them. Again swell of the broken rock will help to keep the topography from being excessively depressed after shooting. In this case a form of crater blasting will most likely provide the best result. Previous work by the authors on regular rooms and crosscuts have indicated this (4). A primary concern regarding irregular workings is the ability to stop the shots at the desired location and therefore be able to get the drill located properly on the next pattern.

The blasting of vertical shafts would require a similar approach to blasting vertical openings caused by collapse of strata above the mined opening. The blasting of horizontal or inclined adits would require a design that might often combine the applicable theory for both blasting in sinkholes and underground workings.

2.1 BLAST THEORY FOR CLOSING INDIVIDUAL VERTICAL OPENINGS

To close an individual vertical opening such as a sinkhole requires that strata around the hole be fragmented and thrown into the central void. In a sense the requirement has similarities to blast casting as employed in surface mines. Blast casting uses designs and explosive loadings capable of throwing substantial amounts of overburden into the adjacent pit previously stripped by a dragline. However, in the current application one does not generally wish to employ an equal degree of violence for reasons explained below.

To throw the material into the void requires that upon detonation the surrounding strata be displaced toward the opening with adequate velocity. Then sufficient material will be propelled toward the opening to result in infilling the feature with attendant closure.

During displacement the rock or clay will increase in volume. The swell generated is important if both the void and the surrounding area are to have a profile near to the original topography. Insufficient displacement will inhibit throw of the overburden into the hole and will also reduce the swell obtained.

Unlike blast casting in strip mines, however, one does not want the blast around a sinkhole to be too violent. First, heavy charges, shot on an individual delay period would generate very heavy blast vibrations in nearby workings. Usually the sinkhole resulted from caving into an otherwise open room. Initial field observation showed that many such voids opened into mined out rooms where there was little caving beyond the sinkhole. Other open rooms were twenty feet or less from the void.

The important consideration is that strong, sustained ground vibrations close to the detonation have greater potential to cause unwanted caving into the workings near to the shot. Thus closing one vertical opening could create another. The result would be a project of greater magnitude and cost than intended.

Second, a blast containing more energy will have a greater tendency to cause cracking of the overburden around the void. When the overburden is soil or overconsolidated clay the phenomenon is even more pronounced. The cracking requires more regrading to obtain the desired result. It may also be hazardous for personnel working around the shot area, or, depending on the amount of post blasting reclamation, the cracks may pose a hazard to citizens subsequently using the blasted area. Shots that employ less energy in the bank help to reduce the amount of cracking associated with gas pressures and ground vibration.

Third, a blast which generates high face velocities and large amounts of throw into the vertical opening may lead to a blast profile that is heaped in the center and exhibits a power trough around the perimeter of the shot. This is not a major problem for regrading, but an after-blast profile with a shallow central depression and some swell around the perimeter may well be preferred. Material can then be graded into the center quite easily to bring the void level with the surrounding area.

Finally, one must consider the effect of blast vibration on any nearby structures. These must not be damaged. Further, people respond to blasting at considerably lower levels of ground vibration than do their structures. Consequently very low levels of ground vibration will help eliminate citizen complaints due to the blasting.

Often, it can be expected that AML blasting will occur in areas where housing and businesses are located nearby. Blasting methods that allow successful closure of the sinkhole but also minimize blast vibrations at these buildings will therefore be preferred.

In addition to moving the overburden adequately, without undue violence, a sufficient volume of material must be blasted to fill the disturbed area. The disturbed area consists of the vertical opening being filled plus the surrounding overburden in which the blastholes are placed. Therefore, the bank volume (of the overburden drilled off) multiplied by one plus the swell factor, expressed as a percent decimal fraction, must equal the total volume of bank plus void. That is:

$$V_T = V_B(1+S)$$

V_T = volume of bank plus void (cubic feet)

V_B = volume of bank (cubic feet)

S = swell factor, percent decimal fraction

The volume of a caved void is difficult to measure exactly because of its irregular shape. Therefore the value usually must be estimated from approximate measurements.

The swell factor will vary depending on the material being blasted and the degree of displacement achieved. We have completed considerable previous work on the blasted swell factor (4) that pertains to the Beulah Site and which should also provide reasonable approximations for the White Site. This information has been used as needed in the design of individual blasts.

A prime concern then is to move and disrupt the bank enough to obtain the necessary volume for closure that maintains a near surface profile. Failure to accomplish this will result in a deep depression that will be more costly to further reclaim.

Considering all these needs one concludes that using decked charges, independently delayed will likely be the best approach. The individual charges crater off the side of the hole, displacing material into the void, provided the depths of burial, of the charge, are suitable. Decked charges will allow the energy input to the bank to be controlled. Using independently delayed decks reduces the explosive weight per delay and therefore the blast vibrations experienced, at a distance from the blast.

In general the depths to the underground works are modest. Therefore, designing the charges as discrete spherical cratering charges appears appropriate in most cases. If the depth to the mined areas is large then cratering off the side of linear charges, using square root scaling would be most appropriate. In the latter case charge weights increase and this needs to be considered in relation to the vibration factors discussed above.

Cratering theory and its application to mine blasting was first discussed by Livingston in the 1950's (5). The theory was further explained and applied by others (6, 7). Originally cratering related to the application of short charges (relative to the diameter) which could be considered to represent a spherical point charge buried at some depth in the ground. Using the theory one could equate the crater dimensions to the cube root of the explosive weight for a given explosive in a given material type. These scaled dimensions could be plotted against the depth to the center of the explosive charge divided by the cube root of the explosive weight, which is usually termed the scaled depth of burial of the charge. Thus, for a given explosive the dimensions of the expected crater could be predicted for a given depth of burial and were related to the dimensions of the charge.

At some depth, usually termed the critical depth, surface failure just starts. As the depth is reduced, for a given explosive charge, the amount of disturbance increases being, at first, the production of large blocky material followed by increasingly improved fragmentation. Eventually the depth will pass through an optimum which produces the greatest volume of broken material, isolated from the surrounding strata. Further decreasing the depth of burial changes the shape of the crater and results in finer fragmentation and more flyrock. These relationships are illustrated in figure 2-1.

When the depths are divided by the cube root of the explosive weight the resulting value is termed the scaled depth of burial. The burial at which the optimum value of the specified crater dimension occurs is then the optimum scaled depth of burial. A typical cratering curve for a brittle material is shown in figure 2-2. A curve in a more plastic material can be seen in figure 2-3. In plastic materials, such as overburden and frozen till the mechanism of breakage differs. The result is a very limited difference between the optimum burial and the critical burial. Design must be performed carefully. The nature of the breakage in plastic materials is well described by Bauer (7). Since the materials at the test sites are overconsolidated clays the rapid change from optimum to critical depth in plastic materials needs to be considered.

It is to be noted that the direction in which the material moves will be dependent on the relationship of the depth of burial to the side and the depth of burial to the top. In general if the charge has a high degree of burial to the top relative to the face, movement will be to the face. If the burial to the top is less movement will be upward. In a decked hole around a vertical opening movement from the lower decks can be expected to be toward the face while movement from the top

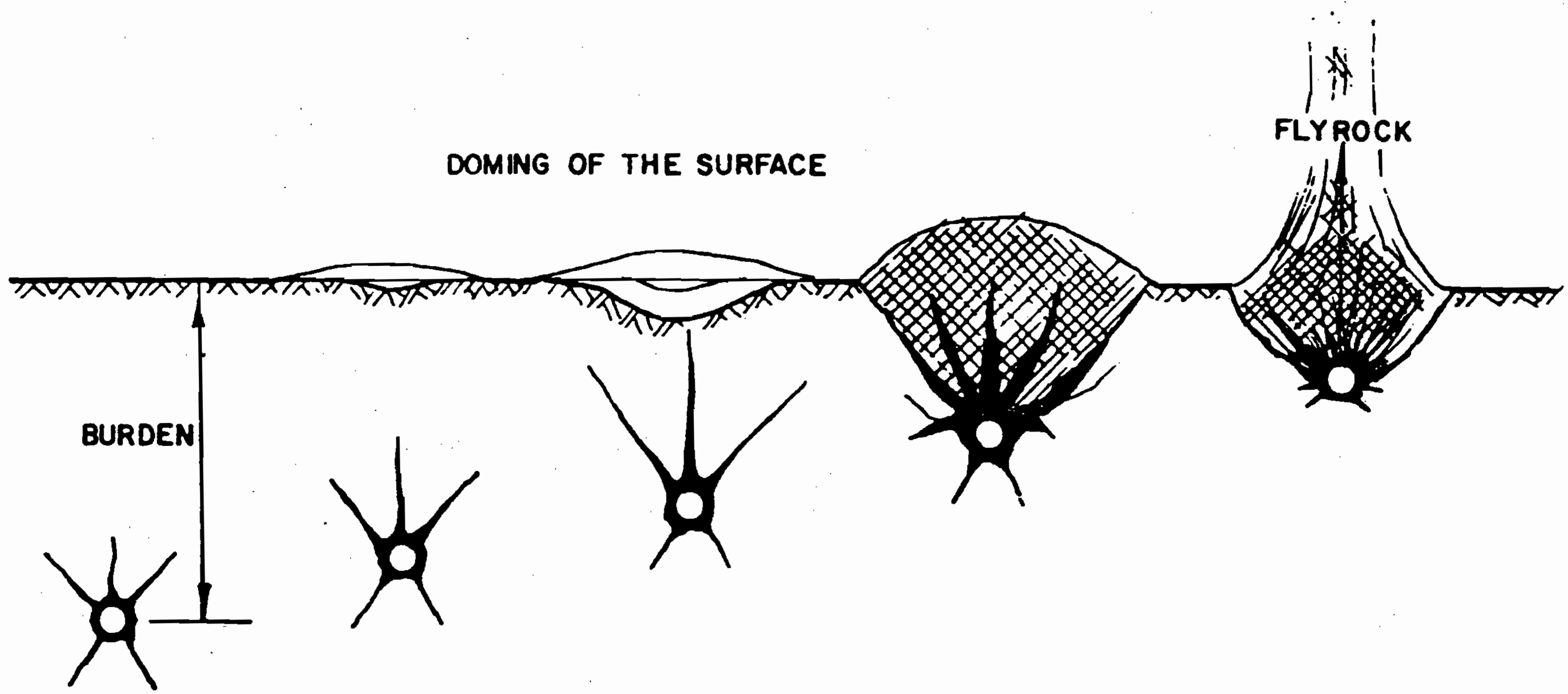


FIGURE 2-1: SCHEMATIC OF THE EFFECT OF DECREASING THE BURDEN ON CHARGES FIRED IN ROCK.

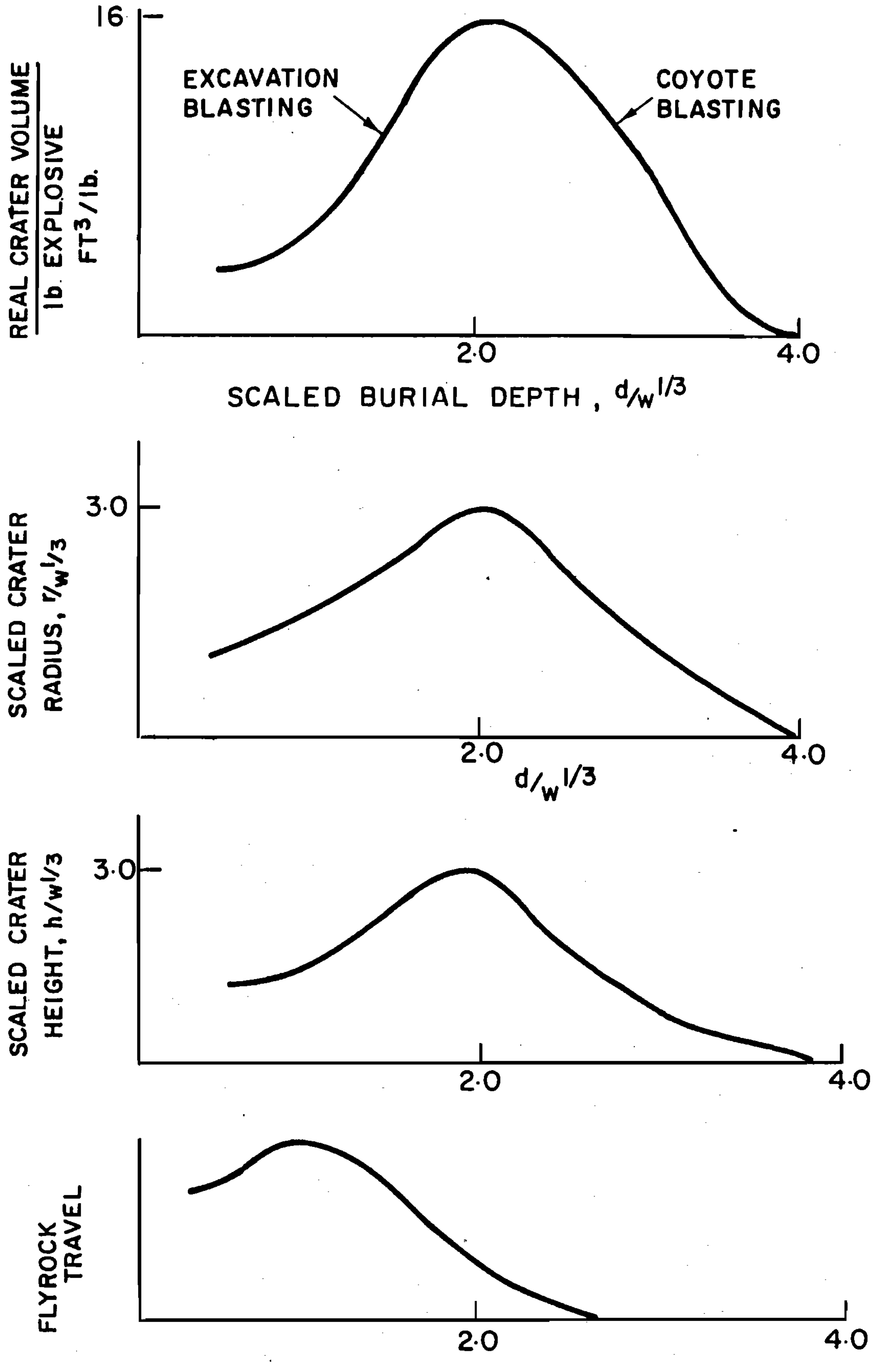
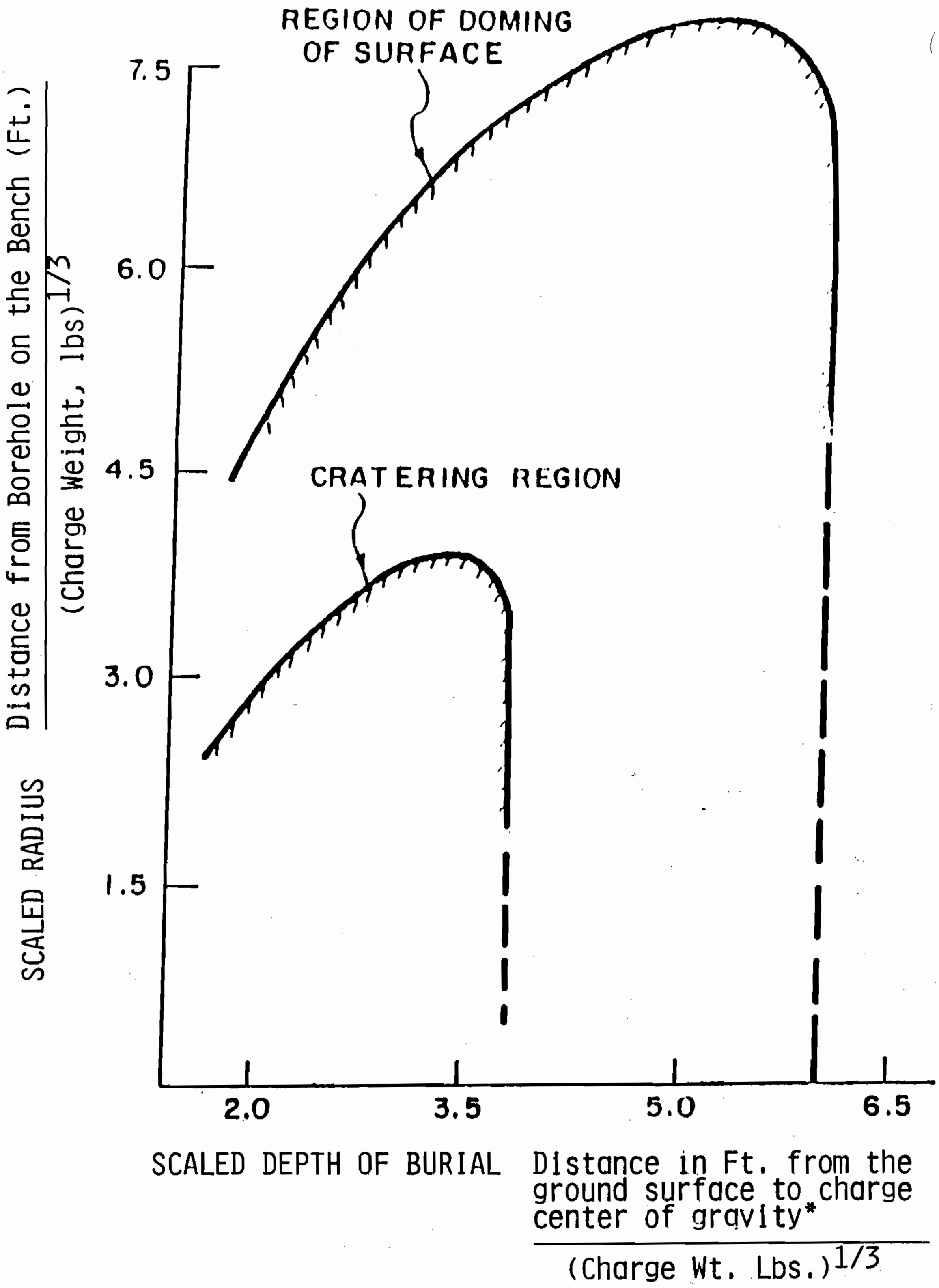


FIGURE 2-2: CRATERING RESULTS IN BRITTLE ROCK WITH SPHERICAL CHARGES.



* FOR LONG CHARGES THE EFFECT OF THE TOP END OF THE CHARGE CAN BE APPROXIMATED BY THE WEIGHT OF THE TOP 8 DIAMETERS OF CHARGE LOCATED 4 CHARGE DIAMETERS FROM THE TOP

FIGURE 2-3: SCALED CRATERING RESULTS IN SANDSTONE (LIVINGSTON deck may be more toward the top surface. Since there is some angle of trajectory on the displaced material from the top a considerable amount of the upper overburden can be expected to be thrown over into the void area.

In general, then, the approach has been to design a series of cratering charges in the hole to displace the material into the sinkhole. These have been designed as spherical cratering charges for which the depth of burial scales to the cube root of the charge weight.

Often, in overburden, the scaled depths of burial for ANFO are quite low. In ditching and trenching work in overburden using ANFO scaled depths of burial as low as 1.30 $ft/lb^{1/3}$ are possible (8). These are for ejection craters and represent the apparent crater. For work in AML blasting in overconsolidated clays optimum burial depths can be expected to be greater. It was not expected that the scaled depth of burial would exceed 2.5 $ft/lb^{1/3}$ however. Typically it might be about 2.0 $ft/lb^{1/3}$.

Since these charges are intended for spherical cratering it is important that the length of the charge not be excessive relative to the diameter. In general one finds that to approximate a spherical point charge the length of the charge should not be greater than eight times the diameter of the charge. For a six-inch diameter hole the length should not exceed four feet.

In reality the charges must be fit into the depth of hole with an adequate amount of stemming left between decks. Therefore it may often be necessary to vary the length of some of the deck charges to fit everything into the hole depth. This is not considered to be a great problem. To some extent, the charge may start to act like a linear charge, however, it will still move material toward the face. It has been considered for the study that the charge length should not exceed 12 to 14 times the diameter for reasonable results.

It is also important to consider how much stemming should be placed between the decks. If too much is used movement will be sluggish and breakage between the explosive decks will be poor. If too little is used there is an increasing chance that cross propagation between decks will occur. Therefore, one should provide adequate stemming to negate or minimize this problem.

One study showed that the sand gap across which a slurry might be expected to cross propagate 5 percent of the time in 6-inch diameter was 11 feet (8). For an emulsion the value is estimated to be 5.5 feet. Thus the emulsions are less sensitive to cross propagation than slurries. ANFO was not tested in this study.

Considering cross propagation possibilities it was decided to use 6 feet of deck stemming whenever possible. This would not eliminate all possibilities of propagating through the deck but would eliminate most. At the same time the configuration would allow for adequate breakage and movement. However, the hole geometry will dictate different deck stemming heights in many cases.

Another factor to be considered was the spacing between holes. These need to be spaced to allow adequate energy input to the bank and also to allow good overlap of the craters forming around each hole. Thus, the strata could be uniformly broken and displaced.

Assume that the optimum scaled crater radius $R/W^{1/3}$ is 2.5 ft/lb$^{1/3}$. Then for ANFO in a 6-inch hole the radius is 8.7 feet when the ANFO is placed in a 4-foot deck. If $R/W^{1/3}$ = 2.0 ft/lb$^{1/3}$ then the radius is 7.0 ft/lb$^{1/3}$. Therefore, it was determined that initially, at least, hole spacings would be kept at 8 to 10 feet. This would allow for reasonable overlap between holes. The distance from the hole to the face was also to be maintained at 8 to 10 feet. For ANFO this leads to a depth of burial of 2.3 to 2.8 ft/lb$^{1/3}$. However, where a bagged heavy ANFO product is used the scaled depths of burial become 2.0 to 2.5 ft/lb$^{1/3}$. To improve movement and protect against any water in the bottom of the hole it was expected to use the HANFO or equivalent product in the bottom deck of most holes. For ANFO and 20-foot holes drilled 10 feet from the crest of the void and 10 feet apart the nominal powder factor would be about 1.14 lbs/cyd. The actual powder factor can be expected to be somewhat less because of breakage behind the holes. For this work, however, powder factor is not likely to be the best measure of expected performance. Rather, the scaled depth of burial relationships should provide a better basis on which to design these shots. In any event this powder factor is quite typical of that used in blast casting shots for coal mines.

Another important consideration is the millisecond delay timing. It is essential that there be sufficient relief toward the vertical opening to allow material to move preferentially in that direction. Restricted relief will decrease the amount of overburden thrown into the hole. It is also likely to cause material to heave up along the perimeter of the shot. Further, substantial back break and cracking behind the blast may well occur.

Therefore it is desirable to have each deck detonate at a different time. This will give successive decks in a hole good relief to crater to the void. Each hole will have good relief to detonate toward the vertical opening.

It was decided to use 42 millisecond delays on surface and Nonel down the hole delays of periods that would yield 50 ms between decks. This had worked well in the past (4). With this delay pattern the shot would detonate such that the decks fired in a diagonal fashion. In a three-deck hole this means that the bottom deck fires, then the middle deck in the previous hole and finally the upper deck two holes back. Then the sequence begins again. Therefore, as each deck fires it has good relief to the side as well as to the free face.

Another reason for delaying each deck is to minimize blast generated ground vibration, both in the immediate vicinity of the shot and also at further distances where housing may be located. With the proposed delay system no decks overlap, unless there is random inaccuracies in the delays which result in overlaps. Each deck fires at least 8 ms from the deck before it. It was our expectation that this approach would lead to minimum ground vibration.

In summary, it is expected that independently delayed deck charges that crater material toward the vertical opening will work best. Usually these will be spherical cratering charges designed to a scaled depth of burial between 2.0 $ft/lb^{1/3}$ and 2.5 $ft/lb^{1/3}$. This yields a powder factor typically between 1.0 and 1.20 lbs/cubic yard. In some cases, depending on hole depth, decks may be longer and approach a linear cratering situation. Then the scaled depths of burial should be 2.1 $ft/lb^{1/2}$ and 2.45 $ft/lb^{1/2}$ where the weight of explosive is now the weight per foot of explosive column. For the above depths of burial the holes should be 8 to 10 feet behind the face.

The radius of crater formed is expected be 7.0 to 9.0 feet. Spacings between holes can be calculated accordingly and are expected to be 8 to 10 feet. Whenever possible the stemming between decks should be kept as close to 6 feet as possible. However, to fit the decks into the hole it may be necessary in some cases to reduce the deck stemming. Figure 2-4 illustrates the proposed blast layout, loading and tie-in. Figure 2-4 shows that adjustment to designs may be necessary to fit the holes around the opening. Figure 2-5 is a typical vertical opening to be blasted.

2.2 BLASTING IRREGULAR OPENINGS

When blasting closed mine workings the first question that arises is whether to blast out the pillars or shoot in the strata above the open rooms. If the pillars are shot then a general collapse of the area might be expected. Conversely, blasting material into the mined rooms and crosscuts will cause collapse only over these features. The resulting topographic variations may be more pronounced.

Even if the pillars were designed to be quite regular these may well have partially failed over the ensuing years. Therefore, the pillars are quite likely to be hard to find with exploration drilling. Exploration costs may increase. This is especially possible when the mine was not worked in a systematic

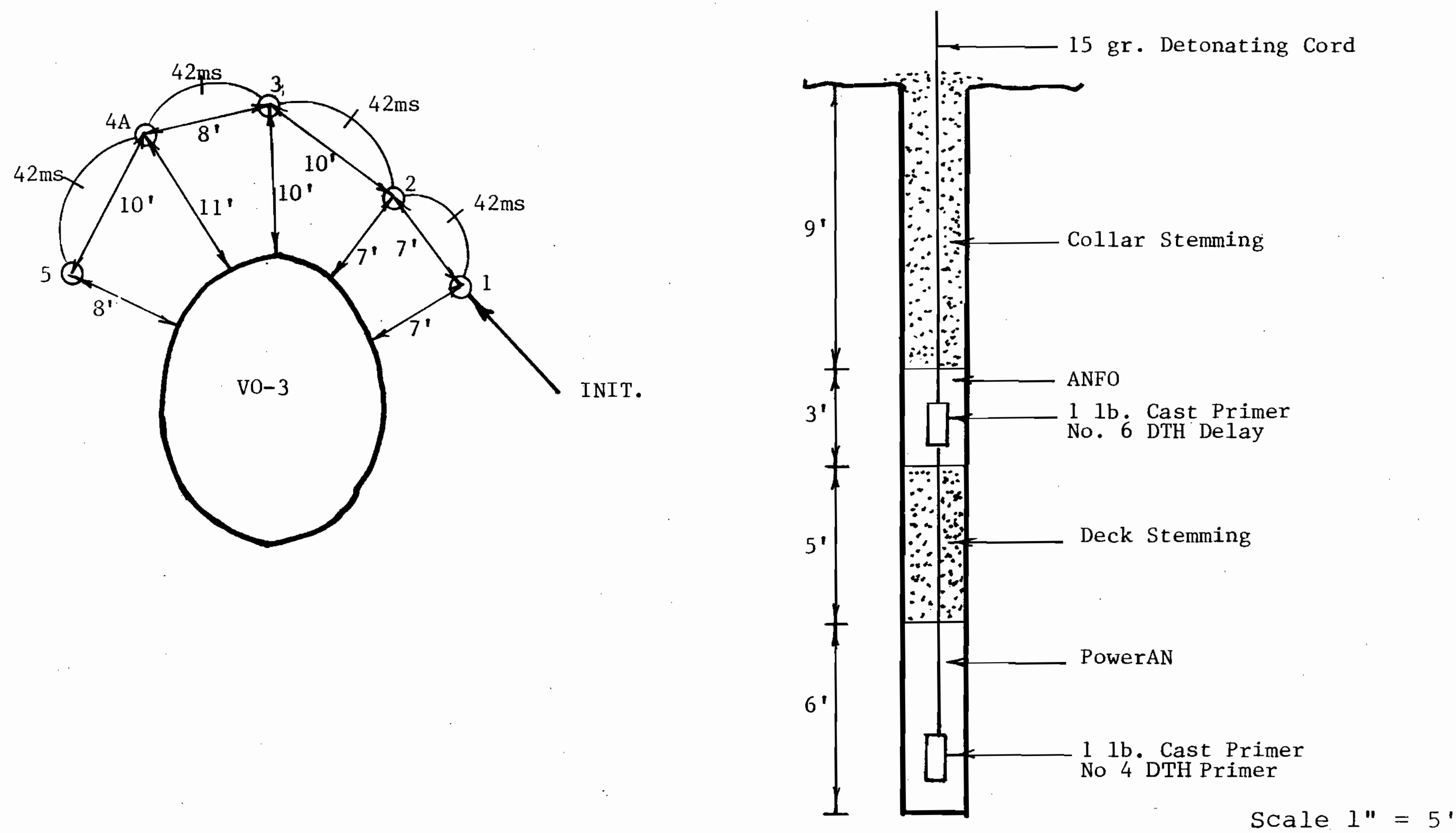


FIGURE 2-4: ILLUSTRATION OF BLAST LAYOUT AROUND A VERTICAL OPENING (VO-3) AND CORRESPONDING HOLE LOADING DESIGN

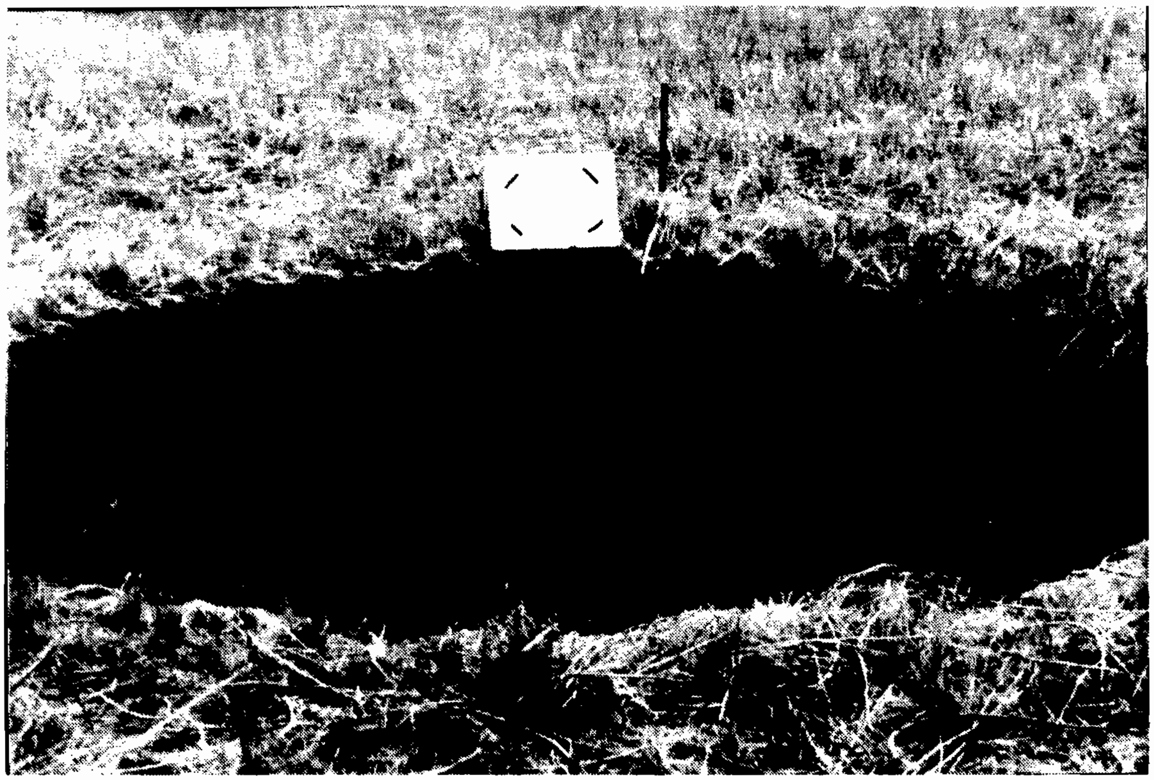


FIGURE 2-5: EXAMPLE OF A VERTICAL OPENING TO BE BLASTED AT THE BEULAH TEST SITE

fashion. Pillars are then of varying size, shape and spacing. They become very difficult to define exactly.

If the pillars could be found and blasted, one does not believe that it is clear that the whole area would necessarily cave. It is possible that the blasted pillars might still provide some roof support. Variations in overlying strata could also result in non-uniform caving. Having blasted all the pillars it would be very difficult to go in and do further work if the area did not cave as planned.

For these reasons it is believed better to collapse the material above the voids. The rooms can more likely be found. Also, the assurance of success is better. If a room does not fully collapse it does not impede work on other rooms. In fact it may well be possible to go in and further cave the opening in question. The overall area disturbed will be less. The main disadvantage is that, depending on the size of the void below and the amount of material above the room, there may be a considerable drop of the surface upon blasting. In some cases this would increase the post-blasting reclamation requirement.

To collapse the voids a cratering technique can be expected to work best. This method was tested in a previous study and found to work very well on long straight rooms that had been mined according to a well developed mine plan (4). These rooms were as much as 200 feet long and were typically 22 feet wide with regular rib pillars between rooms. Given the wide span the strata above tended to collapse quite readily upon blasting. The crosscuts were also straight and easily found. However, these were narrower and were found to be more difficult to blast successfully. Success was achieved by changing the pattern of the blastholes.

At the White Site the rooms and crosscuts were very irregular in direction, length and width. Therefore, it is quite likely that the drill patterns would be smaller than in previous work, for the same 6-inch blasthole diameter. Also, because the rooms are so variable the blast patterns are not likely to be as regular as was the case in prior research.

To move material into the void it is necessary to design the charges to crater overburden down into the void below. Further, unless the blasthole diameter is very large, it may be necessary to use more than one cratering charge in the hole. As the depth to the workings becomes greater the number of individual cratering charges increases.

The charges are independently delayed so that each crater has formed and begun to eject material into the room below before the next cratering charge in the hole fires. Consequently, the strata overlying the void is successively blasted down, filling the room with blasted material. The system has similarities to Vertical Crater Retreat (VCR) mining in underground operations but has significant differences as well.

Most notable is the fact that in VCR mining single cratering charges are used to blast off the ore in the lower region of the blasthole. Once the ore is mucked out of the stope the same blastholes are used for the next cratering shot and so on until the top of the stope is reached. For blasting old mines, on the other hand, all the cratering charges are placed in the hole at once. The detonations are separated by appropriate delays so that the lift below will have relieved before the next one begins to move. Therefore, overall movement downward can be maintained.

The exception will be the top cratering charge. This explosive deck will experience greater freedom to move upward, depending on the depth of burial, and may therefore crater to the top surface. Often, this is a good result as it insures that the top is broken and avoids a thin layer of relatively undisturbed material with a void underneath it. In cases of very deep cover above the mined area it may be possible to bury the top deck much deeper, yielding little surface disturbance but still filling the void so that it is not transferred upward. This will be discussed in more detail later in the report.

The design of the blasthole requires that the bottom charge be buried above the room at a distance that generates a good, well fragmented crater of material that falls down into the room. The scaled depth of burial on this charge is often the most conservative because coal left in the roof increases the difficulty of blasting the strata into the void. The cratering charges above the bottom one are buried a distance above the previous charge that again provides for an optimum crater. The charges are separated by deck stemming that consists of drill cuttings. The top of the hole is stemmed to provide good heave of the surface. At the same time the stemming height is designed to provide minimum fly rock and airblast. Both of these latter points are quite important when working in proximity to residences and businesses.

As a result of work in tar sands it appears that scaled depths of burial of up to 3.1 $ft/lb^{1/3}$ are possible. However, our previous research has indicated that scaled depths of burial less than this actually result. A value of 2.0 $ft/lb^{1/3}$ could be a good number for design. Some variation around this value will occur as the depths to the workings vary. For a 6-inch hole and ANFO a scaled depth of burial of 2.0 $ft/lb^{1/3}$ means that the standoff from the void to the bottom of the charge should be 5 feet. This gives a depth of burial to the center of a 4.0 foot charge of 7 feet. For the bottom charge a more conservative depth of burial may be needed, especially if there is coal in the roof of the void. Each successive charge should be buried approximately 5 feet above the top of the previous charge.

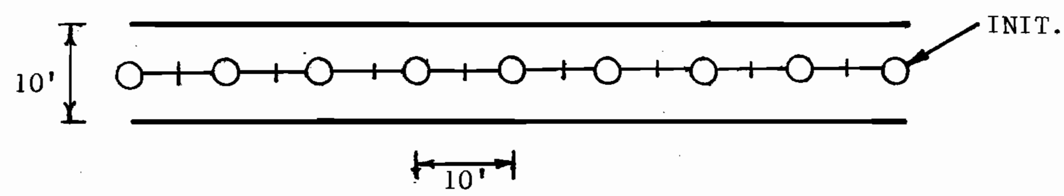
Again, because the crater charges must be accommodated in a hole depth that is fixed by the height of overburden above the coal seam there will be some variation in the depths of burial actually employed. The primary concern is not to get depths of burial much greater than the optimum because performance may well drop off rapidly. Depths of burial below the optimum will change the shape of the crater and increase the velocity at which the strata is ejected but will not cause the magnitude of problem potentially associated with burial depths beyond optimum. This is because crater dimensions do not fall off as rapidly in plastic materials for depths of burial less than optimum as for those greater than optimum.

At the upper surface it is usually desired to get good heave but not to throw flyrock large distances or to cause bursting of detonation gases through the top surface which would add significantly to airblast. Therefore scaled depths of burial should approach 3.5 $ft/lb^{1/3}$. Stemming heights can be expected to be in the 9 to 10 foot range for 6-inch holes and ANFO. Greater depths of burial will decrease surface disruption but one has to watch carefully not to simply transfer the void further up the hole. Figure 2-6 illustrates a typical blasthole design to crater into the void below.

To design the blasts, as described above, it is important to recognize that one must have good knowledge of the location of the rooms and crosscuts. Where the mining was systematic and accurate mine maps are available this need can be met by a limited amount of exploration drilling. However, at the White Test Site the rooms are neither regular nor well documented. Therefore, exploration drilling alone could be expected to require much drilling and to be costly. Other techniques that have a greater area of influence than a single drill hole would be helpful. One such technique is the use of a T.V. camera in the borehole. This is described in Chapter 3.

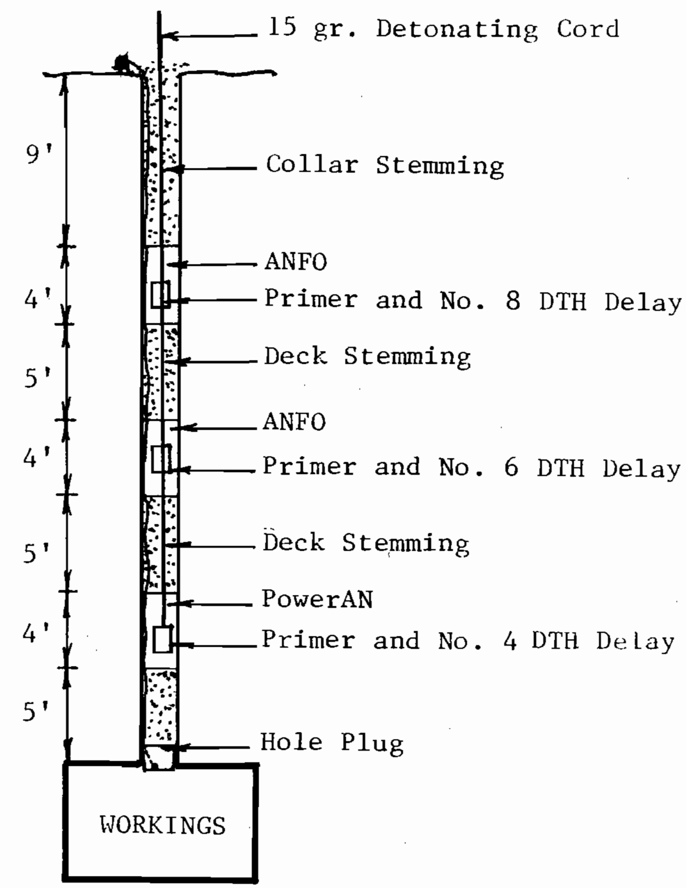
The primary point is that working in these conditions requires a carefully developed and executed exploration plan. In essence one has to draw a mine map from the results of the exploratory program. To successfully shoot in the overburden above the rooms the need for exploration work must not be underestimated when irregular, poorly documented workings are encountered.

Blasting irregular workings creates the concern that it will be difficult to stop the shot at the desired location. The concern centers on the fact that the pillars are often small and the intersections quite large. Therefore, the ground vibrations from closeby blasting may precipitate unwanted collapse of other areas. In other words back break may be hard to control. Backbreak related collapse may not be complete. Equally important it becomes difficult to position the drill on the borehole locations for the next shot. The blasting process can become less well controlled and the results may be more variable.



Scale: 1" = 20'

(a) Plan View of Blastholes Located Over a Mined Room Showing Hole Spacings and Firing Sequence



(b) Cross Section Showing Planned Blasthole Loading for Holes Intersecting a Mined Room

Scale: 1" = 10'

FIGURE 2-6: DIAGRAM OF BLASTHOLE LOCATION, LOADING AND TIMING FOR HOLES PLACED OVER A MINED ROOM FOR THE PURPOSE OF COLLAPSING THE VOID

Initially the intent was to design the shots as though the above problem did not exist. However, if poor control of the collapse was experienced it was intended to increase the stemming to reduce surface disruption. Also careful consideration would be given to where to start and stop the shot to minimize unwanted collapse. Since one did not know in advance exactly what would happen it was considered that if the problem occurred solutions would have to be developed as part of the field research.

Another factor was what to expect in terms of surface configuration after blasting. For successful caving the result would be controlled by the swell factor, void dimensions and depth of overburden. Considerable work on swell factor and surface profiles was completed in the previous research (4). Reviewing this work and recognizing that the void dimensions were fairly small it was expected that major depressions in the surface were unlikely. Rather the topography after blasting should vary from shallow depression to moderate swell of the surface. This is an attractive result if correct because it minimizes post blasting reclamation needs.

Ideally one wants to minimize changes in the after blast topography to the extent possible. Then additional reclamation needs are reduced or eliminated. It may be possible to leave the topsoil in place and still have good vegetation growth potential after blasting.

At the same time one must not simply transfer the void up from the mining excavation to a higher level. This could be a more hazardous result than the original condition. Therefore, simply using greater depths of burial to reduce surface disturbance is not wise. Due regard must be taken for ensuring one hundred percent collapse of the workings. Within this framework, however, there is advantage to burying the top charge as much as possible and thereby reducing surface disruption. During test work at the White Site it was therefore appropriate to vary collar heights to observe surface disturbance and to determine if the void was filled or transferred up the hole. Success in minimizing the surface disturbance could have major implications for the overall cost of AML reclamation using blasting.

The need for extensive exploratory work when blasting in the old works has been discussed above. This exploration time and cost could be reduced if an area blasting approach were taken. In this case the area to be reclaimed would be drilled off on a regular pattern without regard for the location of pillars or rooms. Detonation of the blast should cause settlement of the entire area and this settlement may be quite modest.

The holes could be deck loaded as before and an acceptable pattern developed. A suitable delay scheme would be designed on a row-by-row basis or hole-by-hole depending on vibration and breakage considerations.

Exploration is reduced to finding the perimeter of the workings only. The pattern could be drilled within the perimeter to cave the works in an area manner. To avoid shots which are very large it might be necessary to lay out and shoot several such blasts. Again, careful consideration would need to be given concerning where to start and stop the shot to avoid unwanted caving that would make it difficult to set up on the next pattern.

These patterns contain two types of holes. Those which intersected a void and those which are in a pillar and do not encounter a void. The former holes could be loaded as before to crater into the room below. The remaining holes, drilled to the bottom elevation of the coal seam could be loaded in the pillar to break it up. One deck above the coal could also be used to provide further volume of material to fill the voids.

A prime question is whether it is less costly to specifically blast in the voids or to area blast. Research at the White Test Site has included both methods. Technical and cost comparison will be presented later in the report.

A portion of the White Site contains water filled workings. Therefore, it has been possible to examine the effect of water on AML blasting techniques.

One effect is that waterproof explosives may be necessary in the bottom of the blastholes. Although the water should not rise up the holes unless it is under pressure in the workings use of such an explosive may well be prudent to insure that the performance of the product is acceptable. Non-waterproof explosives, such as ANFO could be degradated by water attack and the energy output of the detonation reduced. The loss of performance could lead to incomplete collapse of the workings.

Another consideration is what happens to the water when the blast is detonated. If open workings remain nearby the water will presumable be forced into these uncollapsed areas. If this is not the case the water must either mix with the falling material or be forced out into the surrounding ground if this is sufficiently fractured. Most likely a combination of the two will occur.

If the water is trapped there is the question whether pressures generated will inhibit collapse of the overburden into the rooms. This seems unlikely given the force with which material will be propelled downward. The question is expected to be answered as waterfilled areas are blasted.

Another question is what effect the water would have if considerable amounts of it mix with the blasted overburden. The primary concern was that if the material became slurried it would tend to flow or creep away from the blasted area leaving a major depression.

Since many abandoned underground mines contain water the results of blasting into water at the White Site provides important information for future projects.

2.3 EXPLOSIVES SELECTION

2.3.1 Introduction

There are several factors to be considered when selecting explosives for AML work. Those include energy output, shelf life, water resistance, ease of use, critical diameter and factors that affect it and the velocity of detonation including the range of diameters over which the ideal velocity is maintained. Temperature and hydrostatic pressure effects may also be important.

Strata over coal seams usually are not strong rocks and the uniaxial compressive strengths seldom exceed 15,000 psi. Much of this overburden is somewhat plastic in nature. When a charge is detonated beneath the free face the explosion cavity expands by compaction and plastic deformation. Outward expansion causes doming and stretching of the surface. As doming proceeds the material starts to break up by a shearing action. However, considerable surface displacement occurs before breakage begins. This has been seen quite clearly in high-speed camera studies of tar sand crater blasting for example.

Therefore, in overburden exhibiting plastic behavior the ability to sustain the borehole pressures for significant time periods is important. Explosives that generate large gas volumes will be best suited to this work. Explosives with very high detonation velocities will not provide better results because the associated shock energy passes rapidly and is not as effective in breaking plastic acting strata as it may be in the breakage of brittle rocks.

2.3.2 Ammonium Nitrate-Fuel Oil

As stated above the strata found above coal mines is not usually very hard. Therefore, ANFO, which has moderate energy output because of its low density is quite adequate for blasting this type of strata. ANFO also generates substantial gas volumes which is an advantage for blasting in the overburden typically found above coal.

In AML work the quantity of product needed at one time is usually not great. Therefore, it will be most appropriate to obtain the product in standard fifty-pound bags rather than in bulk. The bagged ANFO can easily be poured from the bag into the hole providing a fully coupled charge in the hole. The disadvantage is that bagged ANFO purchased in small lots, such as 50,000 pounds, will cause the price to be several cents per pound higher than ANFO purchased in large quantities for bulk use.

The quality of the product is very important. If ANFO is not mixed properly the energy output falls off and blasting effectiveness is reduced. For a proper mix the product should be 94.4 percent ammonium nitrate by weight and 5.6 percent fuel oil. This mixture reacts upon detonation to yield the highest energy output. Usually the weight percentages are rounded to 94/6 since the small increase in fuel oil does not significantly affect the energy output. The relationship of energy output to percent fuel oil is seen in figure 2.7.

In table 2-1 the energy outputs, pressures and composition of the detonation gases is listed. If the fuel oil percentage is reduced to 5 percent the energy loss is 3.6 percent. The drop in thermochemical pressure (equals borehole pressure in a fully coupled hole) is 2.8 percent. At 4 percent fuel oil the energy loss is 15 percent and the pressure loss is 8 percent. One can see that as the product becomes fuel lean the loss of performance is rapid. Much the same is true for fuel rich situations. When fuel lean, the reaction will produce NO and NO_2. The amounts of these gases has not been computed in the table but these will increase with decreasing fuel oil.

The type of prill used can also affect performance. The common product is the porous prill. This form of AN will absorb and hold 6 percent fuel oil indefinitely. Another type of AN prill sometimes used is the high-density prill. This product does not have sufficient porosity to hold 6 percent fuel oil unless it is crushed prior to adding the fuel.

The prill should be strong enough to not break down under normal conditions of handling and temperature cycling. If the prill breaks down caking is likely to occur and the product will not pour freely. The bagged ANFO should have minimum fines Fines will cake readily in humid conditions. The resulting caked product will not pour freely.

For ANFO the ideal velocity of detonation is developed only at large diameters. In figure 2-8 it can be seen that the diameter must be eight to ten inches for the development of ideal velocity. It can also be seen that at 6-inch diameter the velocity of detonation is about 13,500 ft/sec versus 14,500 ft/sec ideal velocity in confined charges. This means that the pressures produced will be less since the detonation pressure is proportional to the square of the detonation velocity. By the same token at 6-inch the velocity is high enough to provide quite good results.

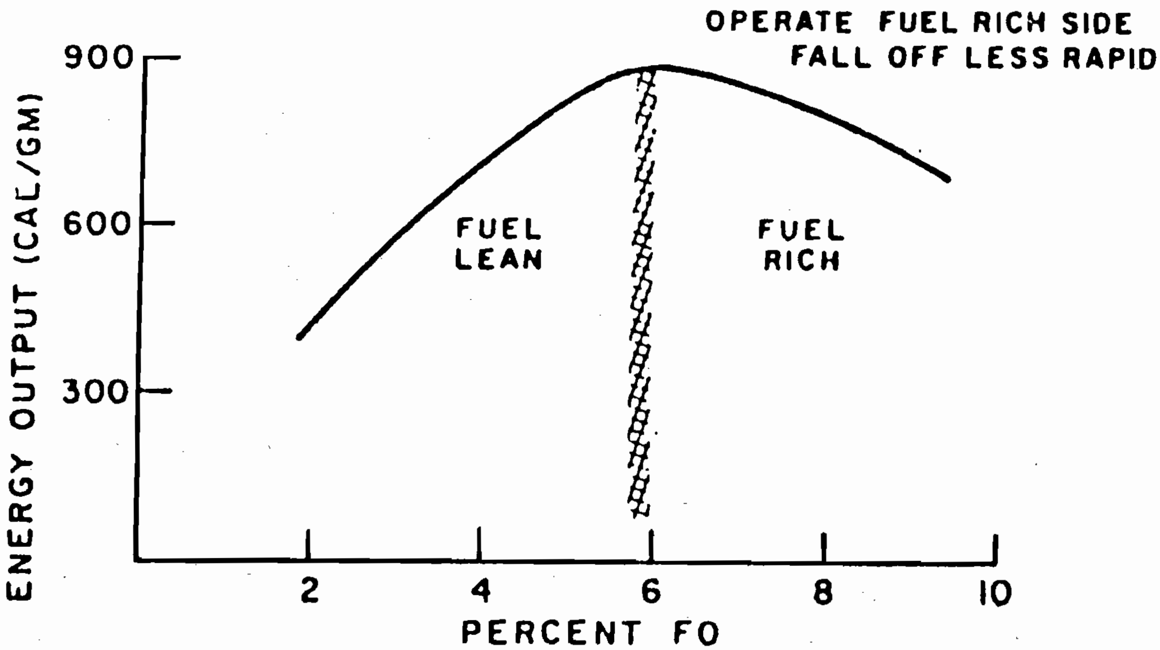


FIGURE 2-7: ENERGY OUTPUT VS. PERCENT FUEL OIL ADDED TO AMMONIUM NITRATE.

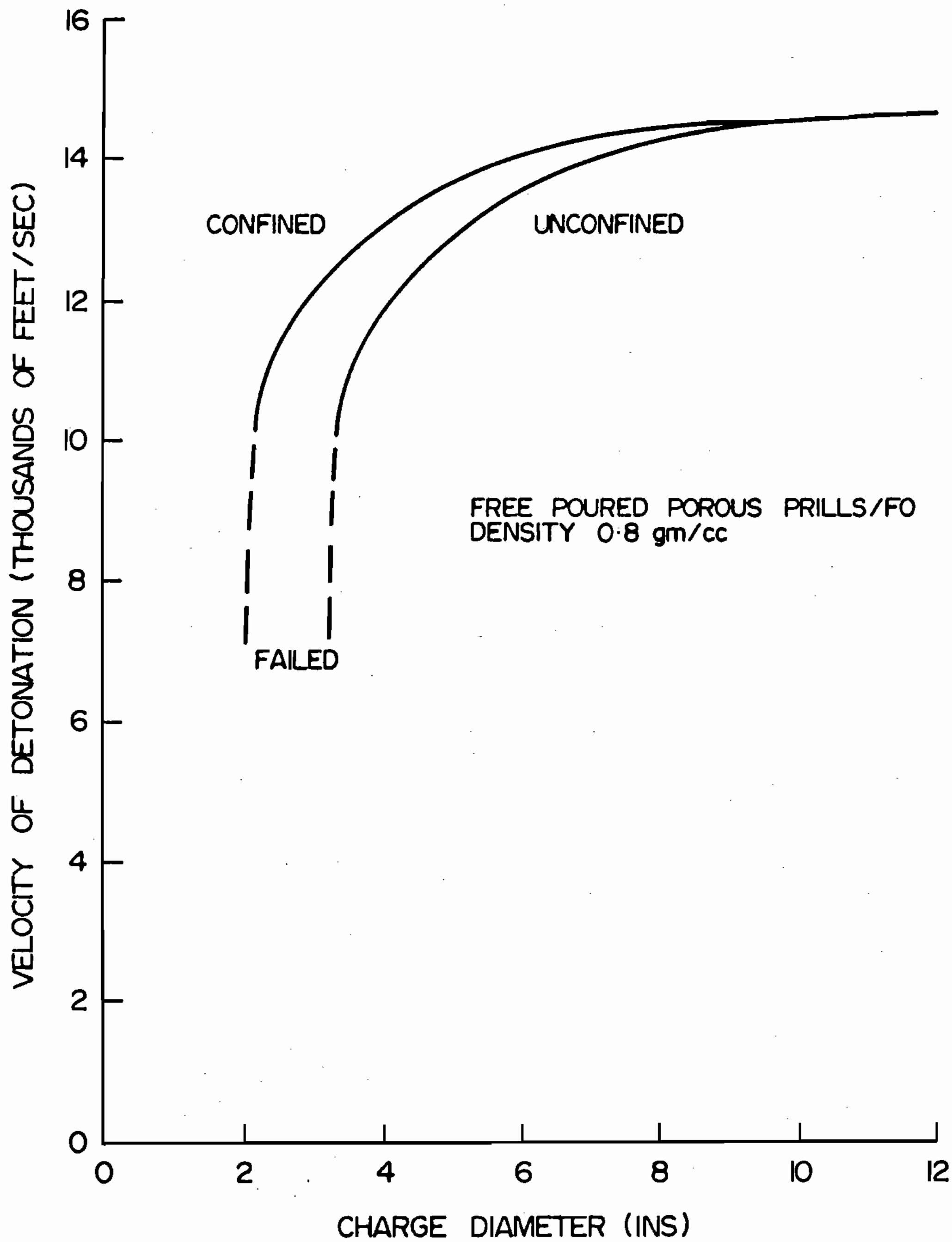


FIGURE 2-8: VELOCITY OF DETONATION VERSUS CHARGE DIAMETER CURVES FOR POROUS PRILLS/FO CONFINED AND UNCONFINED

The critical diameter for ANFO can also be seen in this figure. In confined charges it is 2 inches. For unconfined charges the critical diameter increases to 3 inches. Even in the unconfined case the critical diameter of ANFO is quite acceptable relative to a shooting diameter of 6 inches.

The primary disadvantage to ANFO is that it has no water resistance. Even in cold water (0°C) the solubility of AN is 118.3 parts by weight per 100 parts water. Therefore, if loaded into water the product will rapidly degrade and the energy outputs will fall off. Evidence of water attack is given by the presence of orange fumes after the shot which are nitrogen dioxide and are present when the ANFO is fuel lean, in this case because the prill is dissolved away from the fuel.

In AML blasting as proposed above there will not be water in the hole because the blastholes penetrate the void and are open ended. However, in overconsolidated clays there may be significant moisture in the clay and therefore seepage on the side of the hole. The AN, being very soluble, will attract the moisture and loss of quality will set in. We have seen this in previous AML work. Also, for sinkhole blasting and area pattern shots not all blastholes are drilled to void so water accumulations are possible.

Usually, the seepage and loss of product quality are not sufficient to significantly affect performance, although some orange fumes may be seen. However, if the overburden is quite wet the problem may become serious.

Figure 2-9 shows what happens as water is added to ANFO. Even small amounts cause a marked reduction in detonation velocity. When 12 percent water by weight is added ANFO will typically fail.

Table 2-2 also shows the effect of water on ANFO; in this case when small amounts are added and the product sits for a week. Marginal, oscillating detonations resulted even when the product was quite heavily primed. Therefore, in AML work it will be wise to load and shoot ANFO as quickly as possible.

Overall ANFO has several characteristics which are advantageous in AML work. These include adequate energy output, easily loaded and low cost. However, the product should not be used when water is present.

2.3.2 Slurries, Emulsions and Heavy ANFO

If ANFO may be attacked by water then the best solution in AML work will be to use a waterproof explosive. Waterproof products include the slurries, the more recently developed emulsions and explosives which are a combination of emulsion and ANFO and are usually called heavy ANFO.

VOD = VELOCITY OF DETONATION

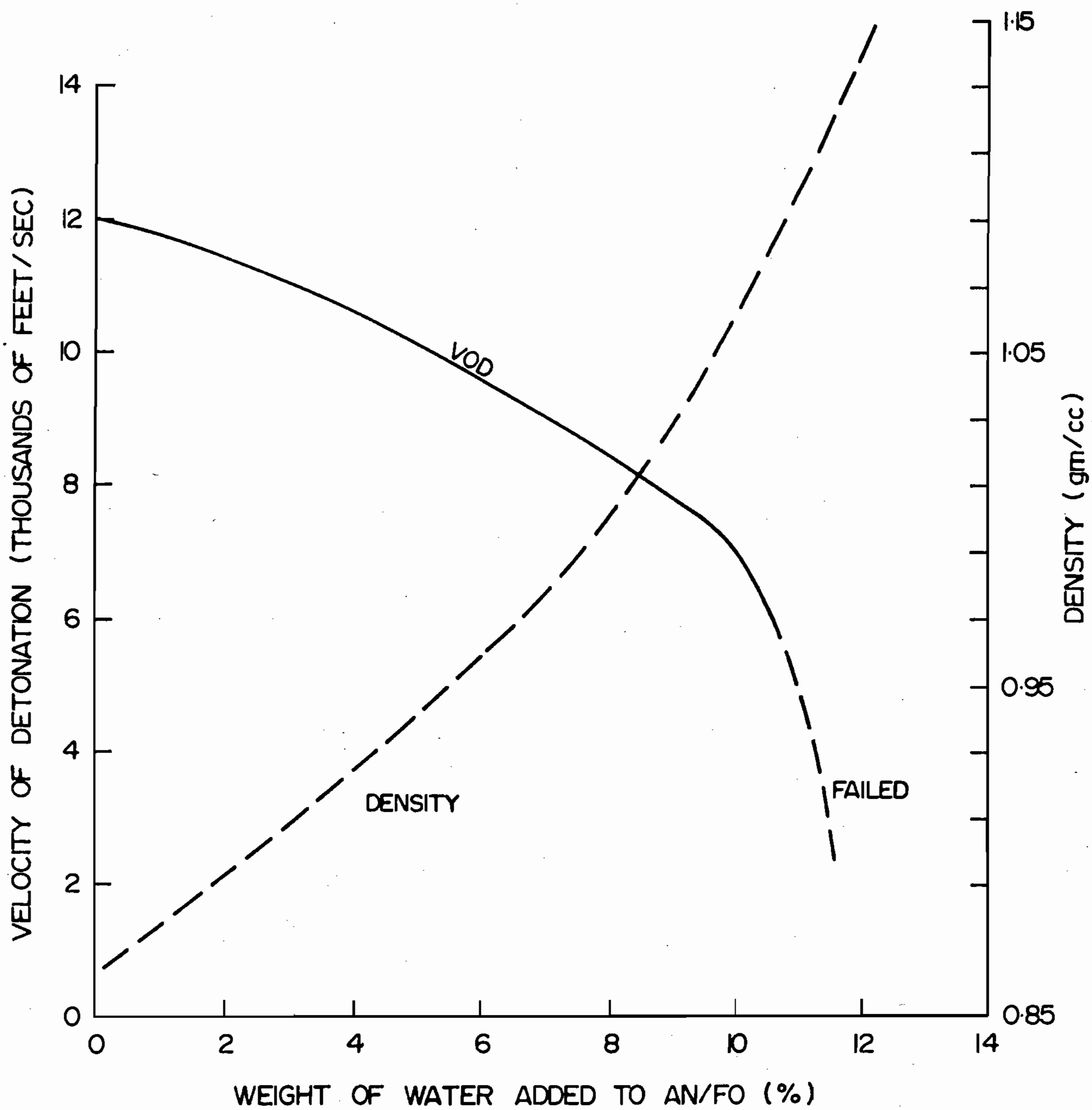


FIGURE 2-9: THE EFFECT OF WATER ADDITION TO AN/FO FIRED WITHIN 5 HOURS OF MIXING IN 4" DIAMETER PLASTIC PIPES USING 1# PROCORE PRIMERS.

SAMPLE	DENSITY AS MIXED 4% H_2O gm/cc	DENSITY AFTER 1 WEEK	PENTOLITE PRIMER WT. (gms)	RESULT
1	0.96	1.10	454	Unstable Detonation
2	0.93	1.02	318	Unstable Detonation
3	0.92	1.00	40	Failed
4	0.94	1.05	18	Failed

Storage Temperature 60°F

The 2% H_2O results were similar. The detonations in both cases were oscillating. Typical of marginal shots. Water therefore has a more pronounced effect on standing. (Ref. 8)

TABLE 2-2

THE EFFECT OF 2% AND 4% WATER ADDITION ON THE SENSITIVITY OF AN/FO IN 4" DIAMETER, 4' LONG PLASTIC TUBES AFTER STANDING FOR 1 WEEK.

All of these products can be purchased in bags. Again, bagged in small lots the product is more costly than it would be in large lots for bulk loading. However bulk loading will seldom be applicable in AML work.

Heavy ANFO (HANFO) has become quite popular. For the product to be waterproof it should contain 50 percent or more of emulsion by weight. Then there is sufficient emulsion to fully coat the prills and prevent water attack. Table 2-3 shows a typical emulsion formulation used in waterproof heavy ANFO. Also shown are the compositions of waterproof heavy ANFO formulations. Several of these include aluminum as more energetic fuel. In AML work products with no aluminum are of most interest since increased energy is not needed to break the rock.

The density and bulk strength of HANFO products are greater than ANFO. When used in bags these increases are partially offset by the decoupling of the explosive from the hole. Typically, in a 6-inch borehole a 5-inch diameter bag will be loaded. A larger bag will tend to hang up in the hole and make proper placement of the charges difficult. The decoupling inherent in the loading procedure may be overcome by opening the bags and dropping the product down the hole in chunks. This is time consuming. Reduction of the decoupling can also be achieved by splitting the bags and then dropping them down the hole. However, our experience with 6-inch holes has been that the explosive tends to squeeze out of the bag and the bags will hang up in the hole. Therefore, the usual procedure is to drop the unopened bag down the hole and accept the decoupling. Given the greater bulk density and strength this does not usually pose a problem.

When HANFO is used there is the question of sensitization. Emulsions depend upon the dispersal of minute air bubbles throughout the mix for much of the sensitivity of the product. These bubbles are introduced either by chemical gassing or by using minute hollow spheres called microballoons. In bagged products microballoons are most often used.

Heavy ANFO with small percentages of emulsion can be used with an unsensitized emulsion. Air bubbles trapped on the prill surfaces provide adequate sensitivity. However, as the emulsion percentages increase beyond 30 percent the performance of an unsensitized product decreases markedly. Sensitized products by contrast maintain performance near theoretical. These results can be seen in table 2-4 where performance of sensitized and unsensitized HANFO can be seen. The effects on velocity of detonation and detonation pressure are recorded in the table. The result is that HANFO containing 30 percent or more of emulsion should be sensitized. If selecting one of these products sensitization should be checked with the manufacturer.

TYPICAL CALCIUM/AMMONIUM NITRATE EMULSION USED IN HEAVY AN/FO PRODUCTION

INGREDIENT	WEIGHT %
Emulsifier	2.0
Calcium Nitrate (containing 2 moles of water crystallization)	38.4
Ammonium Nitrate	38.4
Fuel Oil	10.8
Water	10.4
	100.0

TYPICAL WATER RESISTANT FORMULATIONS WITH VARYING ALUMINUM CONTENTS

INGREDIENT	WEIGHT PERCENT			
Calcium Nitrate	21.1	21.1	21.1	21.1
Ammonium Nitrate	59.1	61.1	64.1	66.1
Aluminum	7.0	5.0	2.0	0
Water	5.8	5.8	5.8	5.8
Fuel Oil	5.9	5.9	5.9	5.9
Emulsifier	1.1	1.1	1.1	1.1
TOTAL	100	100	100	100

TABLE 2-3: TYPICAL WATER RESISTANT HEAVY AN/FO FORMULATIONS.

WEIGHT % AN/FO	WEIGHT % EMULSIÓN	DENSITY (gm/cc)	THEORETICAL V.O.D. (m /SEC)	MEASURED V.O.D. 10"STEEL PIPE (m /SEC)	THEORETICAL DETONATION PRESSURE (K bars)	ACTUAL DETONATION PRESSURE 10"STEEL PIPE (K bars)
100	0	0.83	5,090	5,000	52	50
80	20	1.01	5,370	4,630	72	54
70	30	1.1	6,010	4,330	88	51
60	40	1.23	6,310	4,390	121	58
50	50	1.3	6,460	4,300	134	59
	PLUS 1.6 % MICROBALLOONS					
80	20	0.99	5,370	5,730	70	70
70	30	1.10	5,730	6,540	89	89
60	40	1.20	6,220	6,340	115	115
55	45	1.21	6,280	5,700	118	92
50	50	1.24	6,340	5,670	123	98
0	100	1.29	6,400	6,560	130	130

TABLE 2-4: COMPARISON OF THE THEORETICAL AND ACTUAL DETONATION PRESSURE FOR HEAVY AN/FO MADE WITH UNSENSITIZED AND SENSITIZED EMULSION.

Heavy ANFO, even when waterproof, can be adversely affected by water. This happens in bulk loading when the mix is augered through water. Mixing of water with the product causes the prill to be stripped away and bridging of the HANFO can also occur with the explosive separated by columns of water. When bagged explosive is loaded this is less of a problem. However when using HANFO it is not wise to cut up the product and drop it down the hole to achieve a greater coupling. Table 2-5 shows the effect of adding various amounts of water to HANFO in 12-inch diameter charges. Loss of performance is pronounced, even when relatively small amounts of water are added to the mix. It is to be emphasized that HANFO with 50 percent or more emulsion is waterproof and it is the loading techniques that must be addressed to ensure integrity of the product.

The other products that can be used are slurries and emulsions. The slurries are waterproof products that have been available for twenty-five years. These products provide good service and are fully waterproof. They are less common now as emulsions and HANFO have taken a substantial share of the market. Slurry explosives do tend to be more costly than the others. The mixes can be provided in bulk or in bags. Slurry bags of 5-inch diameter are usually 3-feet long and contain 30 to 50 pounds of product depending on the density of the mix.

When using slurries, especially in 5-inch diameter or less it is important to be aware that cold temperatures affect the sensitivity of the product. Figure 2-10 is a graph of velocity of detonation as a function of charge diameter for two slurries. When the temperature of the basic NCN slurry is reduced from 68°F to 0°F the critical diameter rises from 2 inches to 5 inches. If the slurry were used in cold weather and had a temperature of 0°F, then in 5-inch bags performance of the explosive would be marginal at best. Below 0°F it was difficult to get this product to detonate at all.

Figure 2-10 also shows that TNT sensitized products are much less sensitive to reduced temperature than those sensitized by other means. However, for reasons of cost and availability it is very unlikely that these products will be used in AML work.

Basic NCN slurries are also sensitive to overpressure. As explosive and stemming are added to the hole the explosive low in the hole is subjected to increased loading. The density increases, air is forced out of the mix and the sensitivity decreases. This is seen in figure 2-11. To recover the sensitivity in deep holes the starting density of the product is reduced by introducing air bubbles. This is usually done by chemical gassing but microballoons can also be used. Microballoons are more costly but can substantially increase the shelf life of the product when it is used in bags. If an AML project requires the loading of slurry in the bottom of holes

EXPLOSIVE	REDUCTION IN ENERGY OUTPUT	VELOCITY OF DETONATION IN 12" DIAMETER STEEL PIPE (FEET/SEC)
SO CALLED WATERPROOF HEAVY AN/FO, A, MADE WITH SENSITISED AN/EMULSION + AN/FO	0	18900
A + 10% WATER	17%	12000
A + 15% WATER	27%	9000 (SOME FAIL)
A + 20% WATER	FAILS TO DETONATE	FAILS

TABLE 2-5: EFFECT OF WATER ADDITION ON THE ENERGY OUTPUT AND VELOCITY OF DETONATION OF "WATERPROOF" HEAVY AN/FO.

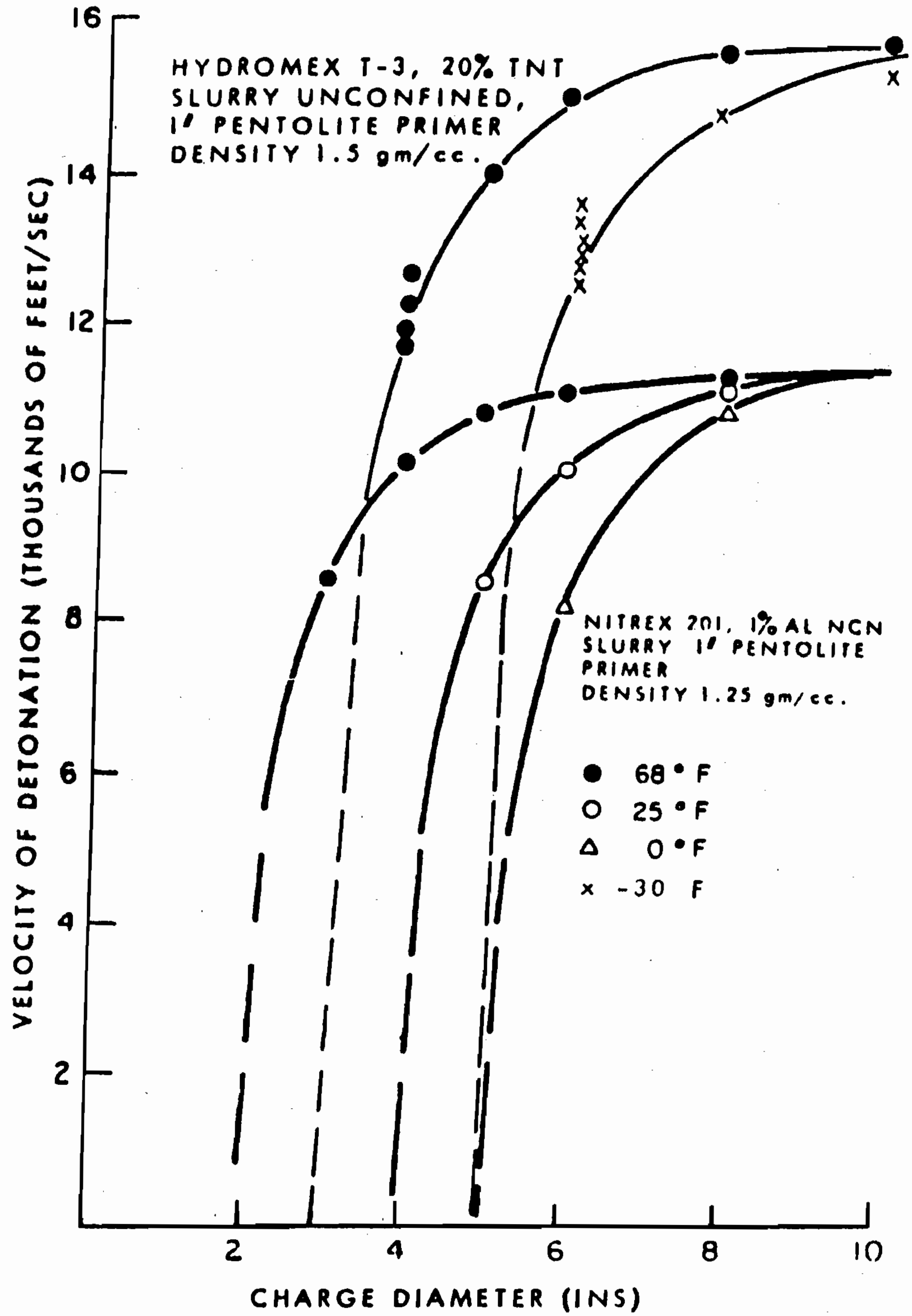


FIGURE 2-10: VELOCITY OF DETONATION CHARGE DIAMETER CURVES FOR A 1 % AL NCN SLURRY , AND A 20 % TNT SLURRY FIRED AT DIFFERENT TEMPERATURES .

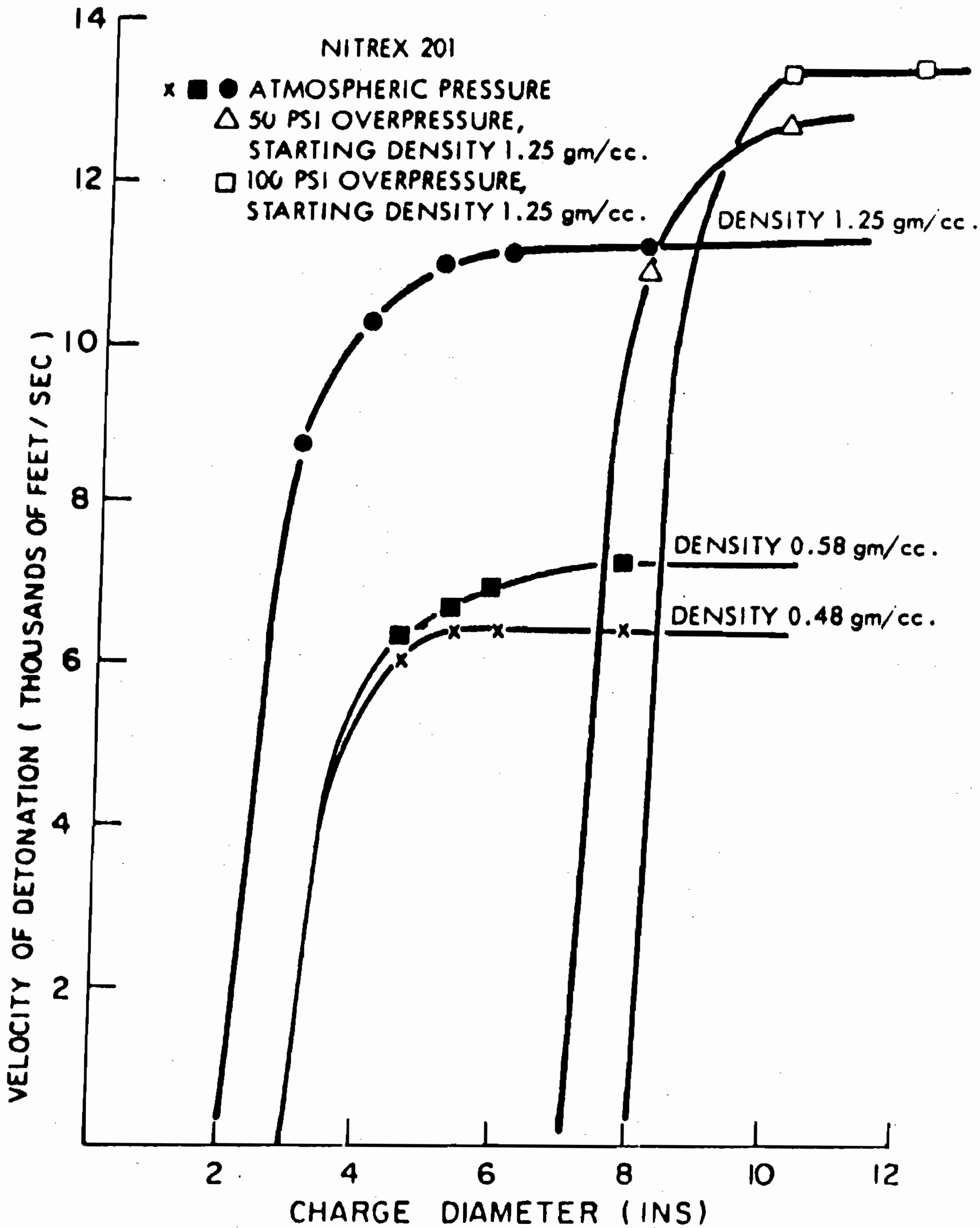


FIGURE 2-11: EFFECT OF PRESSURE AND DENSITY ON THE VELOCITY DIAMETER CURVE OF NITREX 201, @ 1% AL NCN SLURRY.

more than 40 feet deep care should be taken to insure that the product is suitably gassed to give proper performance when loaded in the hole. This needs to be discussed in advance with the explosive manufacturer.

Emulsions are less sensitive to temperature effects than slurries. Therefore these products can often be used at lower temperature than equivalent slurries with acceptable results. Also bagged emulsions are often sensitized using microballoons and therefore hold their sensitivity under overpressure conditions better. If chemically gassed then the pressure sensitivity effects are similar to slurries.

Emulsions provide an intimate mix between the fuel and the oxidizers. Therefore these products rise to ideal velocity more rapidly and have higher ideal velocities of detonation than the slurries. For small diameter charges these results are observed in figure 2-12. These advantages of emulsions may well make the selection of a basic, non-aluminized product preferable to the selection of a basic fuel oil slurry having virtually the same bulk strength.

Table 2-6 compares the strengths of ANFO, ALAN/FO, HANFO and a slurry. The shaded products have similar bulk strengths and will provide similar blasting results on the same patterns. The basic fuel oil slurry has a bulk strength not too different than ANFO and could be a reasonably priced alternative to ANFO in wet holes.

The waterproof HANFO shown has a bulk strength much greater than ANFO. This added energy is not really needed in AML work unless there is particularly hard strata directly overlying the coal or in a layer higher in the overburden. For use in wet conditions the cost of the product would have to be considered relative to other choices. Since the HANFO products contain appreciable amounts of ANFO the pricing can be attractive.

In table 2-7 the properties of emulsions are compared to those of similar slurries. The basic emulsion at 1.2. gm/cc density has a bulk strength relative to ANFO of 1.1, very similar to the basic fuel oil slurry. The basic emulsion would also provide a good substitute for ANFO in wet holes.

Therefore explosive selection for AML work consists of ANFO for virtually all dry hole applications. Where water is a problem the best choices are a basic emulsion, basic slurry or a waterproof HANFO which will generally be in the range of 50 to 70 percent emulsion. Final selection should be based on cost, availability and the technical factors discussed above. The chosen product will usually be purchased in bags and the bags should be 1 to 1½ inches less in diameter than the blasthole.

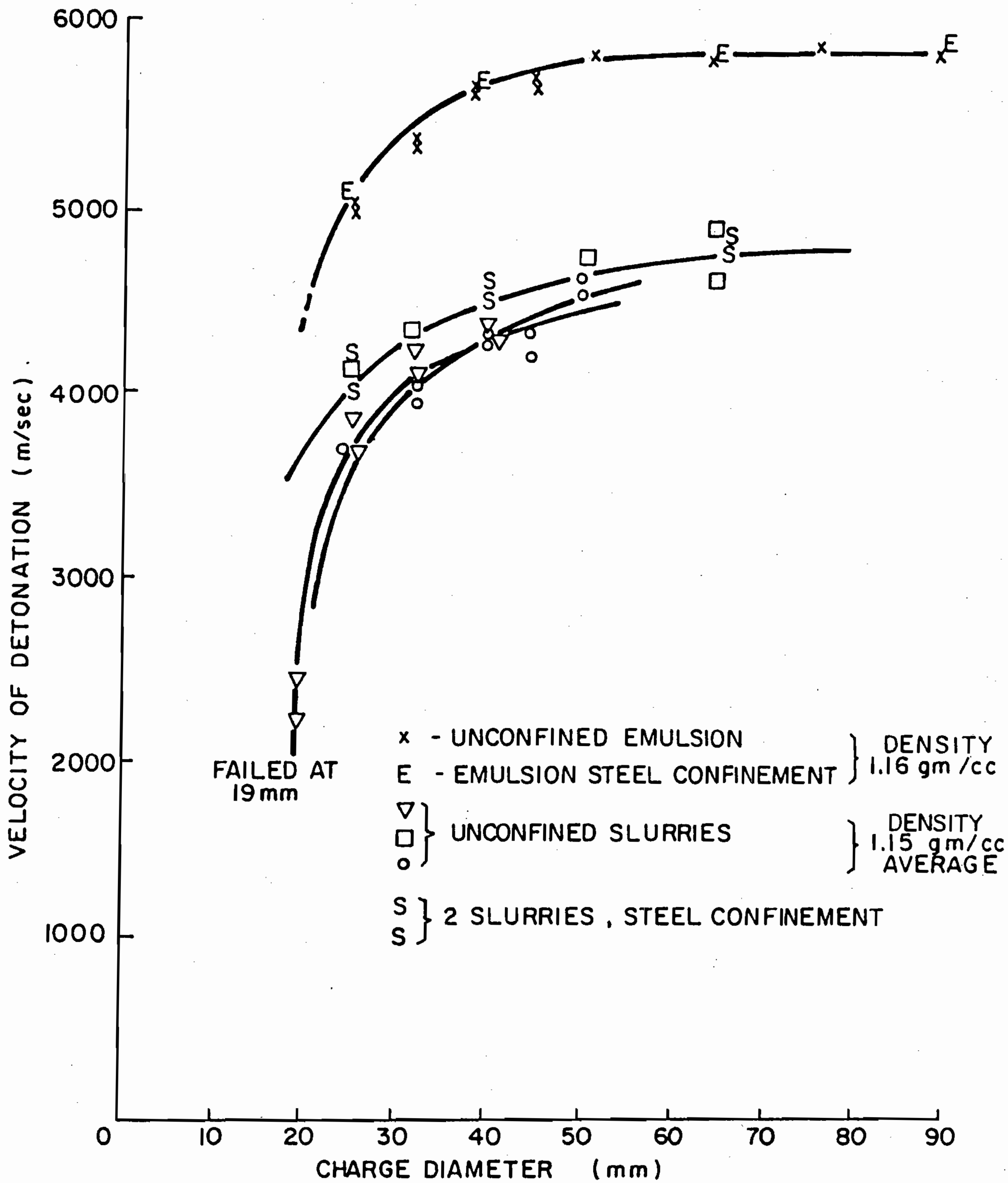


FIGURE 2-12: CONFINED AND UNCONFINED VELOCITIES OF DETONATION FOR SMALL DIAMETER COMMERCIAL SLURRIES AND EMULSION EXPLOSIVES UNDER AMBIENT CONDITIONS. (65°F)

EXPLOSIVE	DENSITY gm/cc	RELATIVE WEIGHT STRENGTH	RELATIVE BULK STRENGTH (RELATIVE TO AN/FO AT A DENSITY OF 0.83 gm/cc)
AN/FO	0.83	1.0	1.0
5% ALAN/FO	0.87	1.13	1.18
7% ALAN/FO	0.88	1.18	1.25
10% ALAN/FO	0.91	1.24	1.36
15% ALAN/FO	0.94	1.35	1.53
AN/FO + 20% AN EMULSION	0.98	0.96	1.13
+ 30% "	1.10	0.92	1.22
+ 40% "	1.20	0.91	1.32
+ 50% "	1.28	0.89	1.37
BASIC FO SLURRY 0% AL	1.15	0.78	1.08
5% AL	1.21	0.90	1.31
9% AL	1.25	1.02	1.54
13% AL	1.30	1.12	1.75

TABLE 2-6: STRENGTHS OF AN/FO, ALAN/FO, HEAVY AN/FO AND 1 SLURRY TYPE.

COMPOSITION	DENSITY (gm/cc)	VELOCITY OF DETONATION (Km/sec)	BOREHOLE PRESSURE (atms)	TOTAL ENERGY (cal/gm)	%ENERGY IN SOLID PRODUCTS	EFFECTIVE VOLUME ENERGY* (cal/cc)
EMULSION,0% AL	1.2	6.44	50,700	68.4	1.0	817
+5% AL	1.32	6.56	56,500	862	7.4	1096
+7% AL	1.33	6.60	58,000	948	10.0	11.98
+10% AL	1.34	6.60	58,800	1016	14.8	1270
+14% AL	1.35	6.50	60,000	1150	20.8	1412
TNT SLURRY	1.45	6.10	50,000	740	8.3	1028
TNT SLURRY+10% AL	1.47	6.10	55,00	1538	23.1	1360
BASIC FO SLURRY 0%AL	1.2	6.4	50,000	680	1	812

*HAVE USED ONLY HALF OF THE ENERGY IN THE SOLID PRODUCTS OF REACTION.

TABLE 2-7: COMPARISON OF THE THEORETICAL ENERGY OUTPUTS OF ALUMINIZED EMULSION WITH VARIOUS SLURRIES.

2.4 BLAST VIBRATION

2.4.1 General Discussion

When a blast is detonated the associated high gas and shock pressures transmit energy to the ground. In the immediate vicinity of the blasthole high compressive stresses crush the rock. The pressure attenuates rapidly and compressive failure does not occur but tensile failure due to the hoop stress does result. If the compressive wave is reflected at a free face further tensile failure occurs at the boundary.

At greater distance the shock front attenuates to an oscillatory wave. Particles in the ground move along cyclically repeating paths. The radiated energy causes particle movements that are within the elastic limits of the rock. The particles will move with certain velocities and at a frequency representing the repeating orbit of the particle motion.

The vibration is a combination of several body and surface waves. It is not necessary to know what the different wave characteristics are in a given event. However, the particle motion is three dimensional, so it is necessary that blasting seismographs measure the motion in three orthogonal directions. These are termed the longitudinal, transverse and vertical direction. Transducers in the measuring head are capable of sensing and recording vibration in each of these modes.

It is common to determine the peak particle velocity in any direction to evaluate the magnitude of the vibration and its potential effect on structures and human response to the blast. However, the resultant peak particle velocity may also be used. Most modern seismographs provide a direct printout of the peak particle velocity in each mode and the resultant peak particle velocity as well.

The original work by the U.S. Bureau of Mines clearly demonstrated the correlation between peak particle velocity and damage (9). This became the standard criteria for many regulatory agencies in the United States and abroad. The onset of plaster cracking was considered to occur at 2.0 ins/sec of peak particle velocity for residential structures. A conservative scaled distance, $D/W^{1/2}$, of 50 $ft/lbs^{1/2}$, where the explosive weight, W, was the weight detonated per delay period was chosen as the value above which 2.0 ins/sec would not be exceeded. This was for cylindrical charges, where the length to diameter ratio was greater than 8 to 1. The original work did not consider that the frequency of the vibration would affect how a given structure responded to the ground vibration. The damage criterion was said to be valid for vibrations from 1 to 500 hertz.

More recently the role of frequency has been carefully studied and found to be of basic importance to the way in which a structure responds to the ground vibration (11). In very general terms, as the ground vibration frequency approaches the natural response frequency of the building the effect of the vibration is greater. Different buildings, such as one story or two story structures, have different response frequencies. However, these frequencies are usually low. Taller houses tend to have lower frequencies.

As the elastic wave radiates away from the detonation the frequency typically reduces. Closer in it is higher. The strata through which the wave travels affects the frequency and is the dominant factor dictating wave characteristics at larger distances.

It was found that predominant frequencies in coal mine blasting tended to be lower and the time durations longer than for other types of blasting (10). This is attributed to the large overburden layers usually found above coal seams. Frequencies are typically less than 20 hertz. By contrast quarry blasts have dominant frequencies between 20 and 40 hertz. Construction blasting shows a wide spread of frequencies from 15 to 80 hertz. Values between 30 and 60 hertz are quite common.

The Bureau of Mines study determined that vibration frequencies below 40 hertz had a considerably higher damage potential than vibration frequencies above 40 hertz (10). This is because the frequencies of midwalls and structure corners are seldom above 40 hertz.

Therefore, practical safe limits for blasting have been recommended by the USBM to be 0.75 ins/sec for modern homes with gypsum wallboard and 0.50 ins/sec for plaster on lath construction. These values are for frequencies below 40 hertz. Above 40 hertz a peak particle velocity of 2.0 ins/sec is considered safe for all buildings. The USBM recommendations are reproduced in figure 2-13.

In AML blasting it must be considered that the frequencies typical of blasting coal mine overburden will be experienced. Therefore, the criteria above for frequencies less than 40 hertz should be used. It must be emphasized that any state or local government regulations, which may be more stringent, must be complied with. A contractor performing AML reclamation must be well aware of local blast vibration requirements before commencing work. A state administering such a contract must be satisfied that the contractor knows the criteria in force and also is capable of designing blasts that will meet the criteria.

2.4.2 Predicting Peak Particle Velocity

Figure 2-14 is a plot of peak particle velocity versus the scaled distance to the point of measurement. It is immediately evident that there is much scatter in vibration data. There are numerous reasons for the scatter, which have been well explained

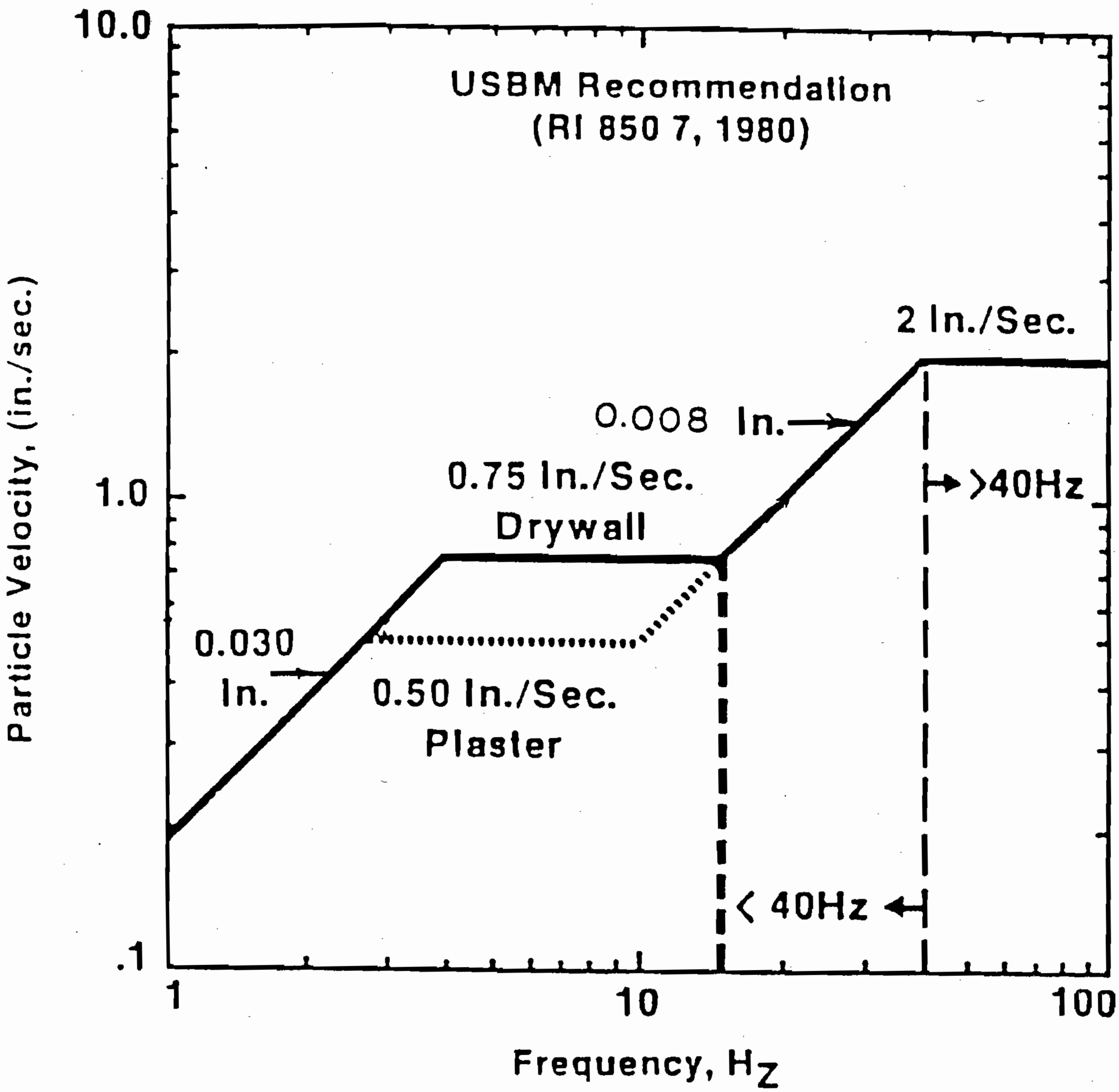


FIGURE 2-13: USBM RECOMMENDATION FOR SAFE BLASTING

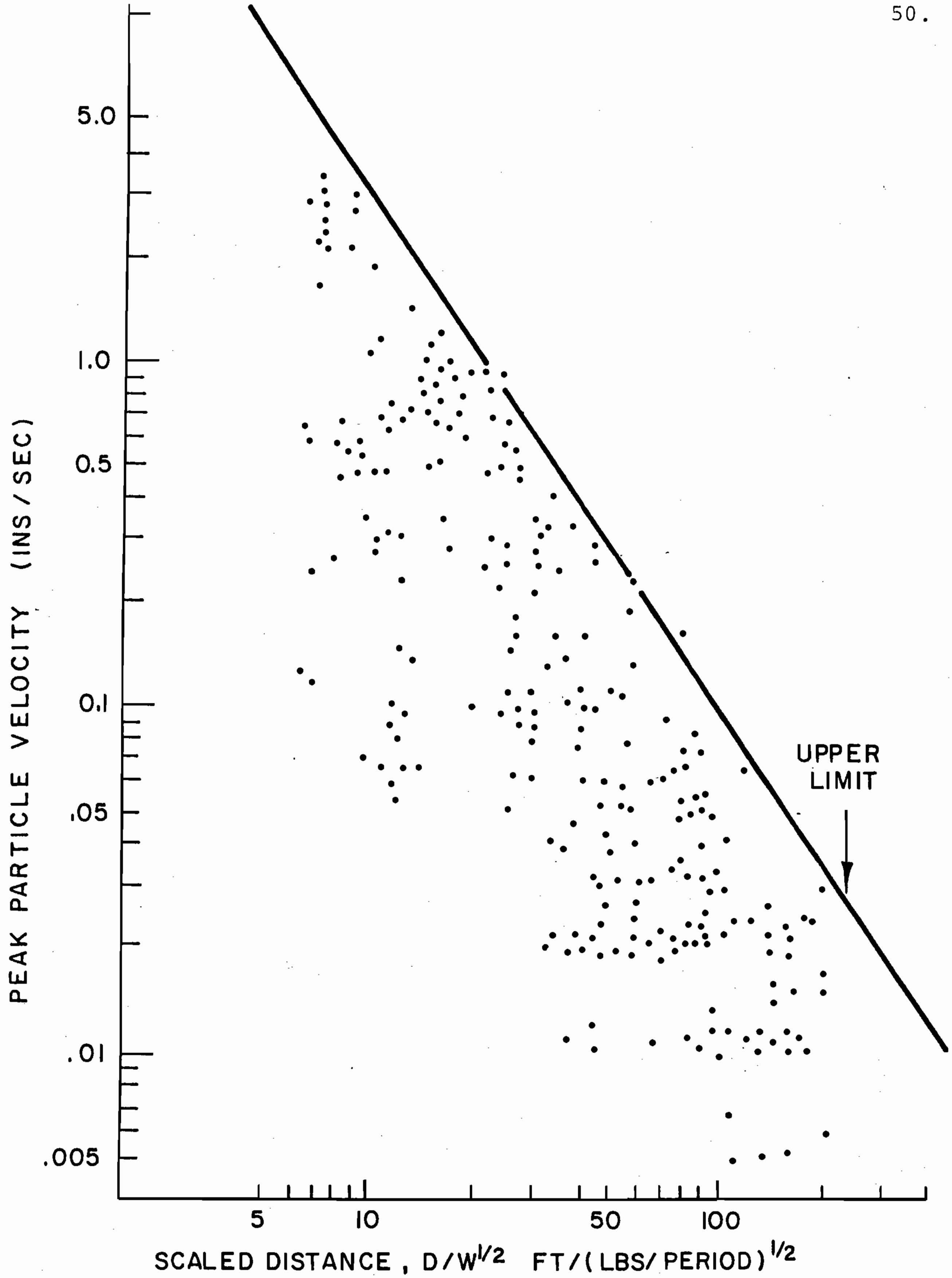


FIGURE 2-14: TYPICAL VIBRATION DATA FROM MULTIPERIOD DELAY BLASTS IN OPEN PITS AND ROCK STRIP MINES.

by Dowding (12). The presence of the scatter means that there is not a definitive peak particle velocity for a given scaled distance. It is not possible to say what the vibration will be before shooting the blast. What is possible is to determine a particle velocity which will not be exceeded. This is represented by the upper limit line in figure 2-14. A best fit line is determined for the data and the upper limit line is parallel to the best fit line and encompasses all of the known data. The data in figure 2-14 is based on large scale open pit and strip mines (13). The maximum charge weight per delay is 60,000 pounds; the maximum total charge weight is 700,000 pounds. The upper limit line is based on some 1500 recordings, a representative sample of which is shown in figure 2-14.

For a given scaled distance the maximum peak particle velocity can be found. Conversely if one desires not to exceed a certain peak particle velocity the scaled distance which must be maintained can be determined. Once the distance to the point of interest is accurately known the weight per delay that can be detonated can be found. For example, if it is found that a scaled distance of 70 $ft/lb^{1/2}$ must be maintained and there is a home 1000 feet away then no more than 204 pounds of explosive can be shot per delay period.

2.4.3 Human Response to Blasting

In 1976 Medearis reported that the human body perceives vibration well (14). However, he also found that the human perception was unrelated to vibrational levels associated with structural damage. This fact had also been indicated by prior USBM work. The results of this work are superimposed on the peak particle velocity versus scaled distance plot in figure 2-15 (13). The important thing to note is that persons find blasting perceptible at about 0.1 ins/sec. The vibrations are unpleasant at a particle velocity of 0.5 ins/sec. These values are necessarily general since many social, economic and psychological factors will affect the manner in which any individual will perceive blast vibration.

The foregoing indicates that to eliminate citizen complaints due to blasting one must keep the vibration levels below 0.1 ins/sec. Reducing these vibrations below 0.2 ins/sec will negate almost all complaints. Siskind et al report that vibration levels below 0.5 ins/sec ought to be tolerated by 95 percent of people that perceive this vibration level as distinctly perceptible (11).

The fact is that AML blast designs often will need to employ human response as the criteria to design to. Fortunately the nature of AML blasting will often allow such criteria to be met.

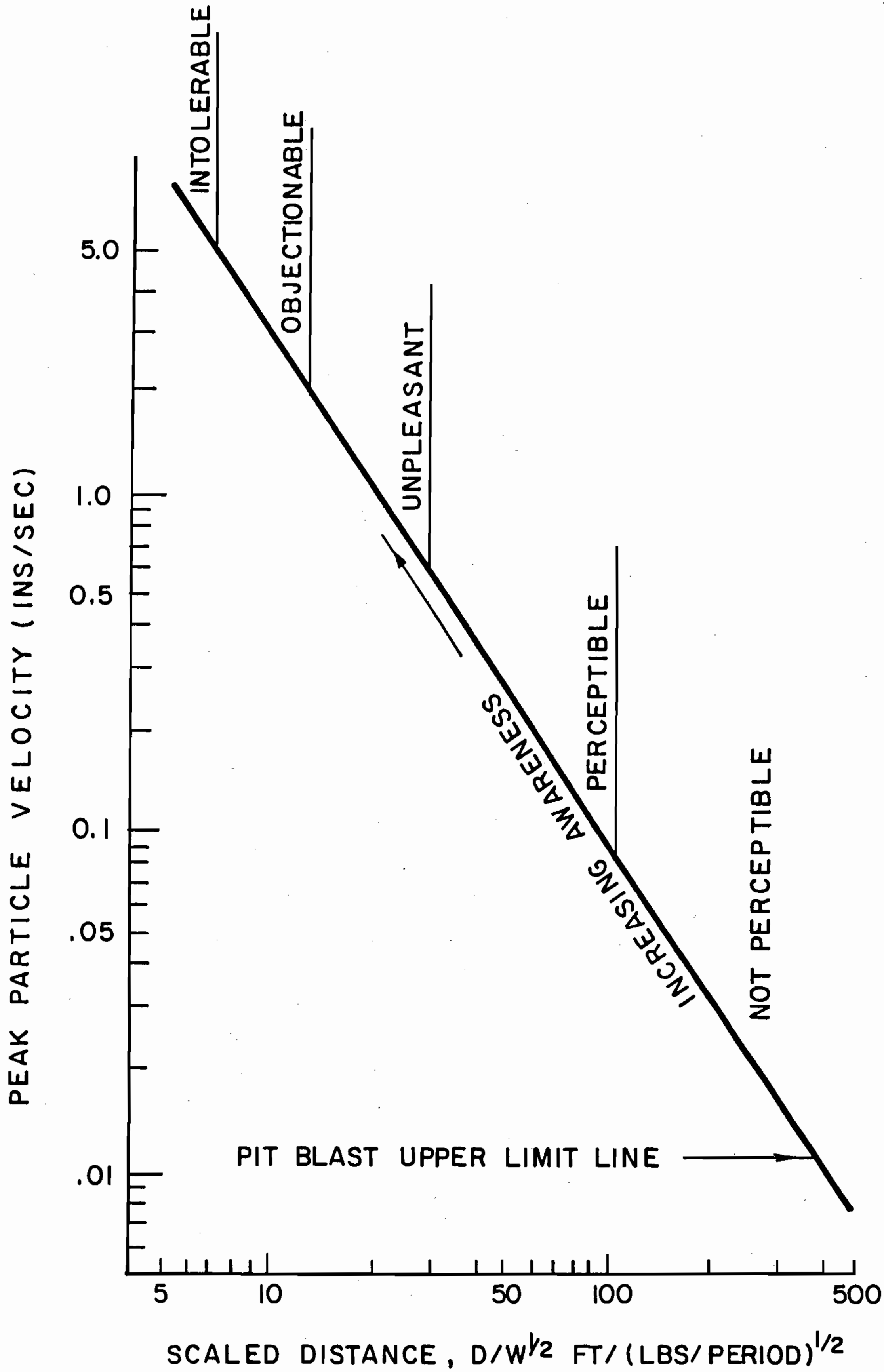


FIGURE 2-15: ANTICIPATED HUMAN RESPONSE AT DIFFERENT SCALED DISTANCES FROM PIT BLASTS.

2.4.4 Airblast

Another consequence of blasting which can cause damage and citizen complaint is airblast. There are several sources of airblast important in AML work. These are listed below:

- Surface delays and primacord
- Inadequate stemming height and lack of proper stemming material
- Burdens near the crest of a vertical opening are too small due to the slope of the sinkhole
- Atmospheric conditions such as temperature inversions and winds in the direction of concern
- Gas escape through fractures in the ground
- Poor millisecond delay sequence

The results can be a high airblast level which can cause window and structure damage and, at a lower level, can invite citizen complaint.

Figure 2-16 relates the peak overpressure to the scaled distance where cube root scaling is now employed. Also shown are criteria for the onset of damage of various kinds. Some window damage may occur at 0.03 psi or 142dB. The graph also shows that increasing confinement reduces airblast for the same scaled distance.

For AML work we recommend that airblast not exceed 133dB when measured on a 2.0 Hz high pass system. This corresponds to an overpressure of 0.015 psi. With citizen response in mind reducing the airblast below this level is a worthwhile goal. Values in the range of 110dB to 120dB ($8x10^{-4}$ to 2.8×10^{-3} psi) should be aimed for.

The following are recognized methods for minimizing airblast:

- Insure adequate charge confinement at the collar and at the face
- Cover all surface delays and detonating cord with drill cuttings
- Examine weather conditions. Avoid blasting when there are temperature inversions, heavy clouds or stiff winds in the direction of the structure
- Use proper delay sequencing. Do not use very short delays

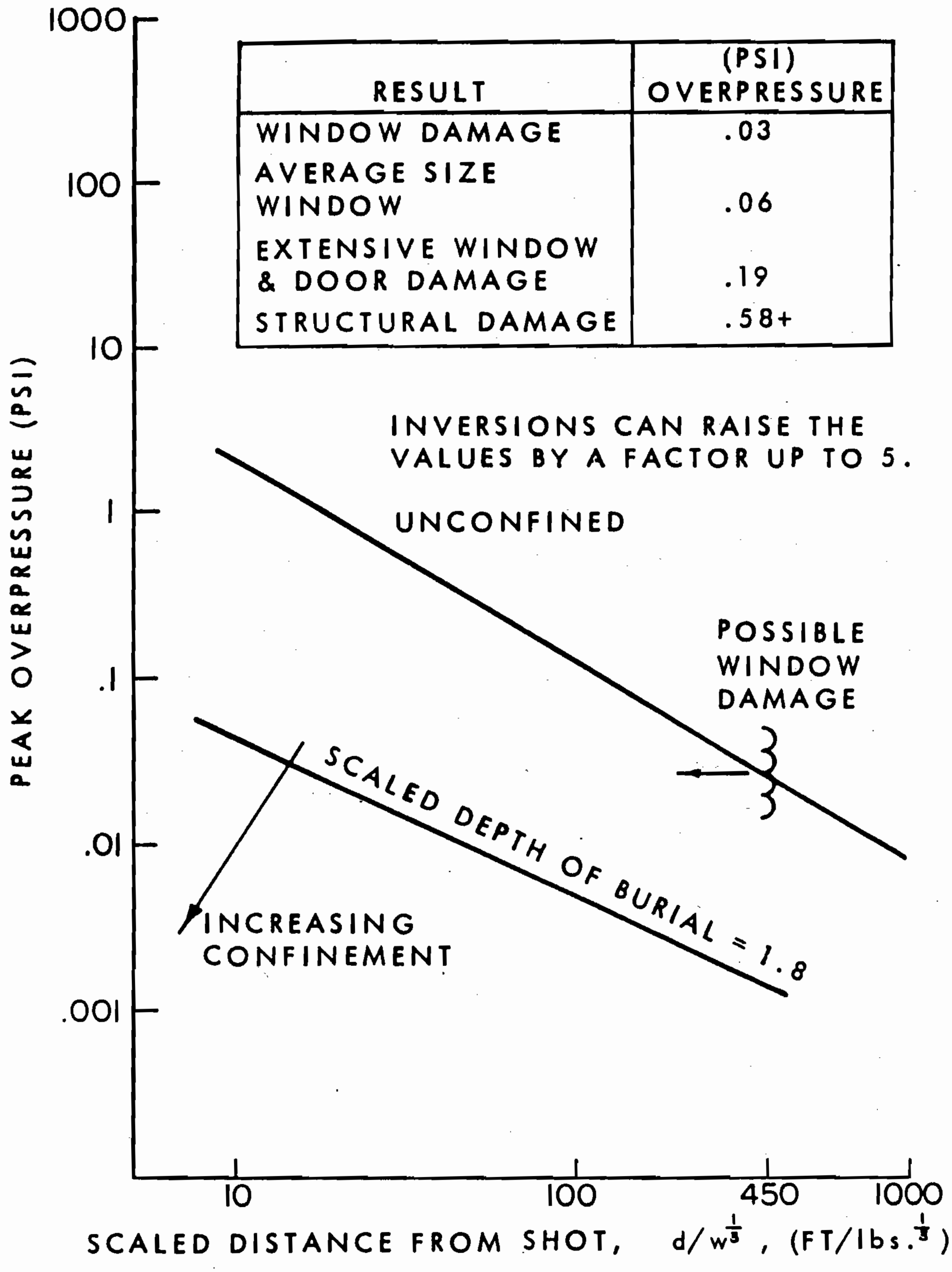


FIGURE 2-16: PEAK AIR BLAST PRESSURE GENERATED BY SPHERICAL CHARGES.

. Reduce the charge weight per delay

. Use low energy detonation systems on surface

2.4.5 Impact of Blast Vibration and Airblast on AML Blast Reclamation

In discussion with various people involved in AML reclamation one senses that many are concerned about the use of blasting because of blast vibration considerations. There is the feeling that the method is only applicable when the site is quite remote.

First, let it be said that there are cases when blasting is not the appropriate mitigation technique. For example, if a subdivision were undermined and subsidence was occurring blasting would not be an acceptable way to solve the problem. Hydraulic or pneumatic stowing would be the appropriate approach.

On the other hand if the area is not undermined reclamation by blasting is feasible quite close to structures. Certainly blasting to within 500 feet of structures is possible. At 500 feet detonating a 42 pound charge (equivalent to 4 feet of ANFO in a 6-inch hole) would give a scaled distance of 77 $ft/lb^{1/2}$ and a maximum expect particle velocity of 0.2 ins/sec. For a structure 1000 feet away detonating 100 pounds of explosive would give a scaled distance of 100 $ft/lb^{1/2}$ and an expected peak particle velocity not exceeding 0.1 in/sec. These results are well within criteria for both structural damage and citizen response. Therefore, blasting, even when close to structures should not be discounted solely on the basis of vibration concerns.

Airblast can also be controlled when close to structures. We have obtained many readings between 110dB and 120dB at measuring points 300 to 400 feet from the shot. Achieving low levels of airblast is a matter of applying the principles above. Especially important are eliminating the use of detonating cord on surface and burying all surface delays and detonating cord downline pegtails.

CHAPTER 3: SITE EVALUATION AND EXPLORATION

3.1 INTRODUCTION

To successfully fill the vertical openings and the mined rooms and crosscuts requires preliminary evaluation and planning for the site. Also, exploration work is required during the field studies to further delineate the workings so that the blastholes can be correctly located.

The nature of the preliminary investigation differed considerably for the two sites. At Beulah vertical openings were to be reclaimed and these could be found by visual inspection. The White Site involved the blasting of underground workings. There were no mine maps except for a very preliminary view of a small portion of the workings, based on prior exploration. Therefore, while initial evaluation and engineering of the Beulah Site could be accomplished readily, there was little that could be done at the White test area until exploratory drilling began.

At both sites rotary drilling was a principle exploration method. Much more drilling was needed at the White Site than at Beulah, North Dakota. Also at White a T.V. camera arrangement was used in exploratory work which was not required at Beulah. Because the Beulah Site consisted of vertical openings, which were open to the surface, considerable measurement of the sinkholes could be performed prior to commencing work on the site.

3.2 BEULAH SITE

3.2.1 Preliminary Evaluation

The location of the test areas at Beulah is shown in figure 3-1. This figure is an aerial photograph of the site produced by the AML Division of the North Dakota Public Service Commission prior to any mitigation efforts. Reclamation was subsequently performed, as indicated on the figure. The reclamation consisted of filling the sinkholes with overburden from nearby sources, regrading and seeding the areas. Therefore, many of the sinkholes shown in figure 3-1 no longer exist. The sinkholes tested had appeared after the initial reclamation had been completed.

Before beginning work at Beulah the area was surveyed to determine the number of vertical openings available for test blasting. Thirteen openings with a variety of configurations were found. The dimensions of each opening were measured and sketches were drawn up for use during planning and blasting. The sinkholes varied from those that were quite circular in nature to those that were long, because a section of a room had collapsed, and open into the room below at one or both ends. In

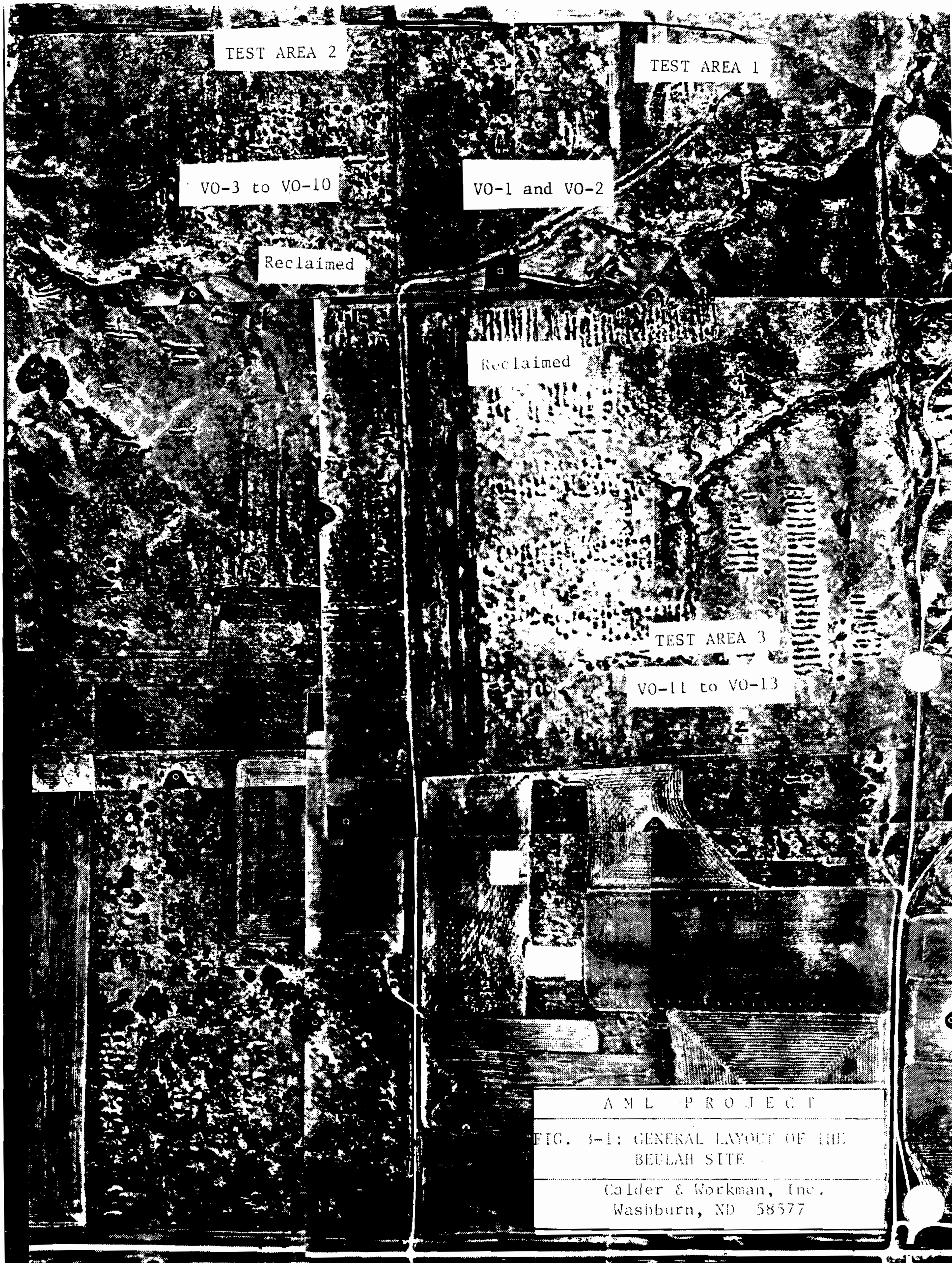


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FIG. 3-1: GENERAL LAYOUT OF THE BEULAH SITE

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some cases the opening was undercut, being narrow at the surface and of much larger dimensions below.

The map in figure 3-2 is a map of the underground workings produced in 1948. Test areas 1 and 2 are to the north and contain the majority of the sinkholes. Area 3 contained three vertical openings which were closed during the field study.

Figure 3-3 is a map of test area 1. This is the area on which previous work, studying the complete collapse of rooms and entries had been performed. This area contained two sinkholes.

From distance and azimuth readings in the field, taken using a Brunton compass and 100-foot tape, the vertical openings were located on the map in figure 3-3. One was located in an area of prior blasting and one was in a room that had not been previously blasted.

The sinkhole designated VO-1 was located where room N-14 and entry NH had been blasted more than one year previously. A hole 25 feet deep had opened up on the entry due to ratholing of the fill into uncaved workings. The vertical opening was circular and about 10 to 11 feet in diameter.

The vertical opening VO-2 had opened up on room S-2 sometime after the completion of prior research work. This room had not been blasted in the previous study. This sinkhole was narrow at the top but considerably wider below. It was 6 to 8 feet in diameter and 14 feet deep. One could not see the open room at the bottom of the sinkhole.

Figure 3-4 is a map of the main area where blasting of the vertical openings was performed. This map is a portion of a larger map produced by the Public Service Commission when an inventory of AML sites was being taken. The sinkholes represented by the broken line features were subsequently reclaimed by filling. The numbers inside the broken lines are the depth of the depressions prior to filling.

On plotting the new vertical openings, all of which had developed after the area was initially reclaimed, it was found that the majority were close to prior caving activity. The new features had developed either on adjacent rooms or further along a room already experiencing collapse.

VO-3 was a sinkhole that had developed near to the access road. This opening was circular at the top with a 6 to 7 foot diameter. It dropped down a few feet and then angled off to the workings below. The height of the slope was approximately 3 feet and the slope length was 20 feet.

The next vertical openings were designated VO-4A and VO-4B. These were two openings that had developed on the same

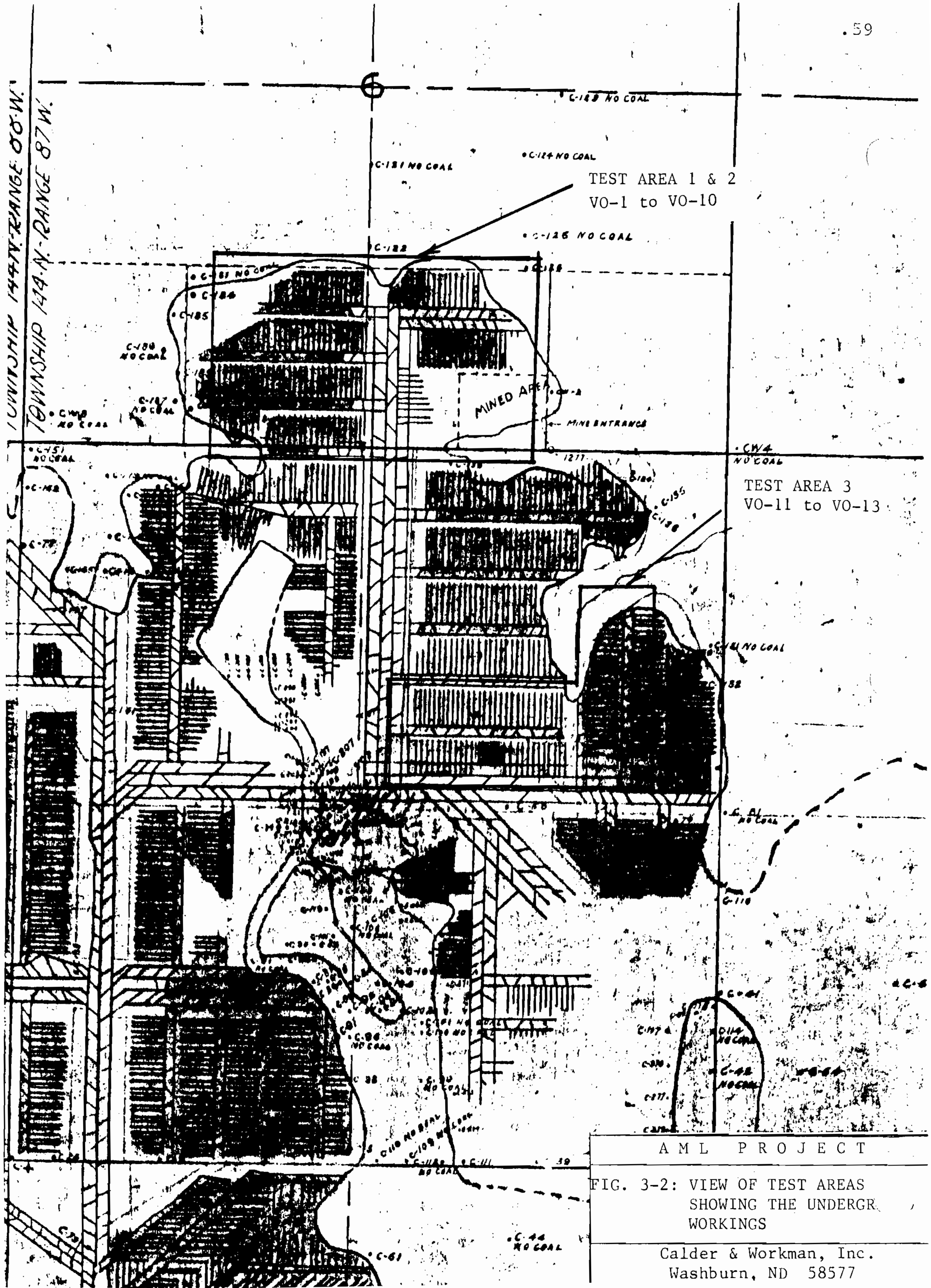

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FIG. 3-2: VIEW OF TEST AREAS SHOWING THE UNDERGR WORKINGS

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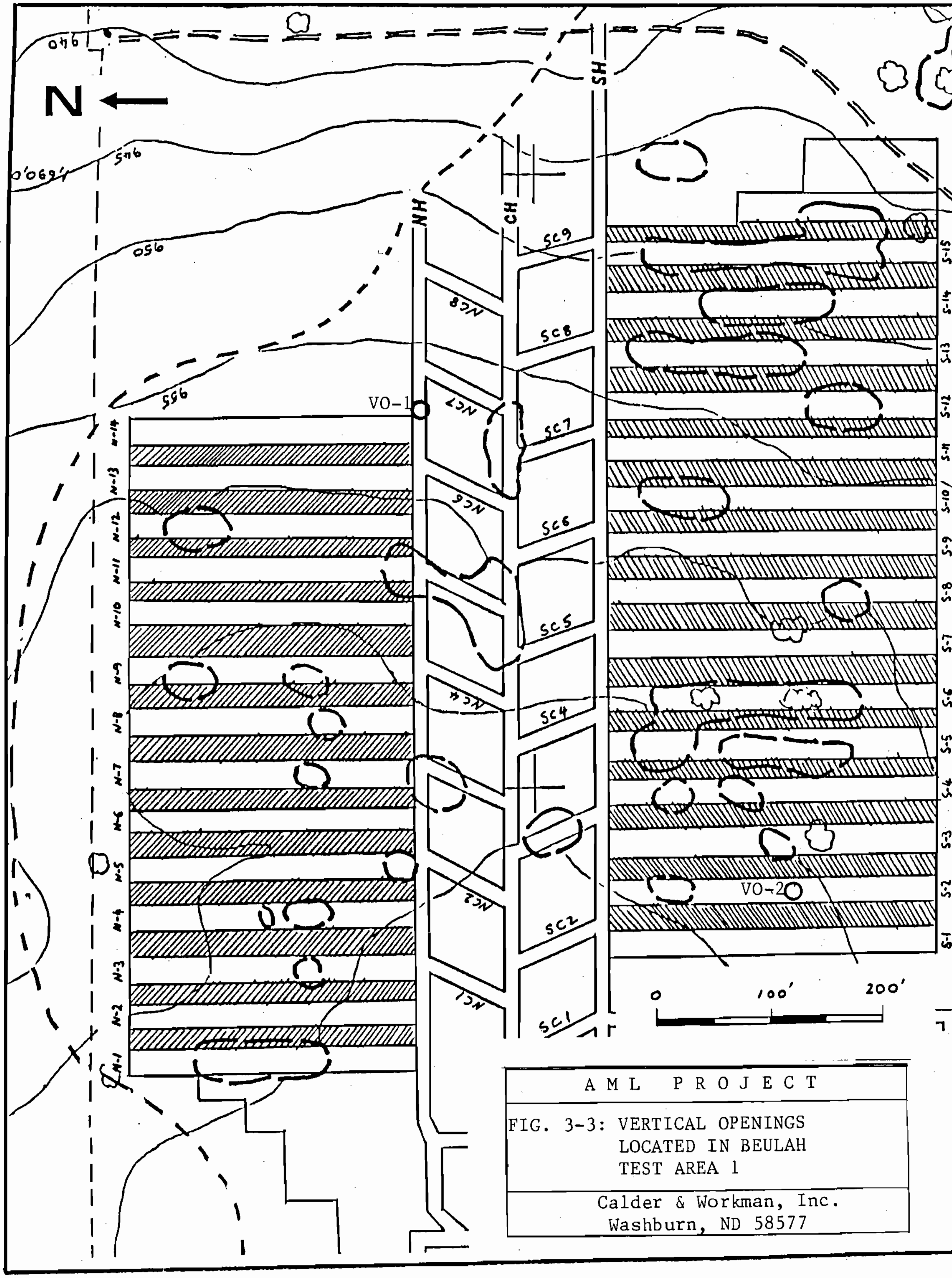

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FIG. 3-3: VERTICAL OPENINGS LOCATED IN BEULAH TEST AREA 1

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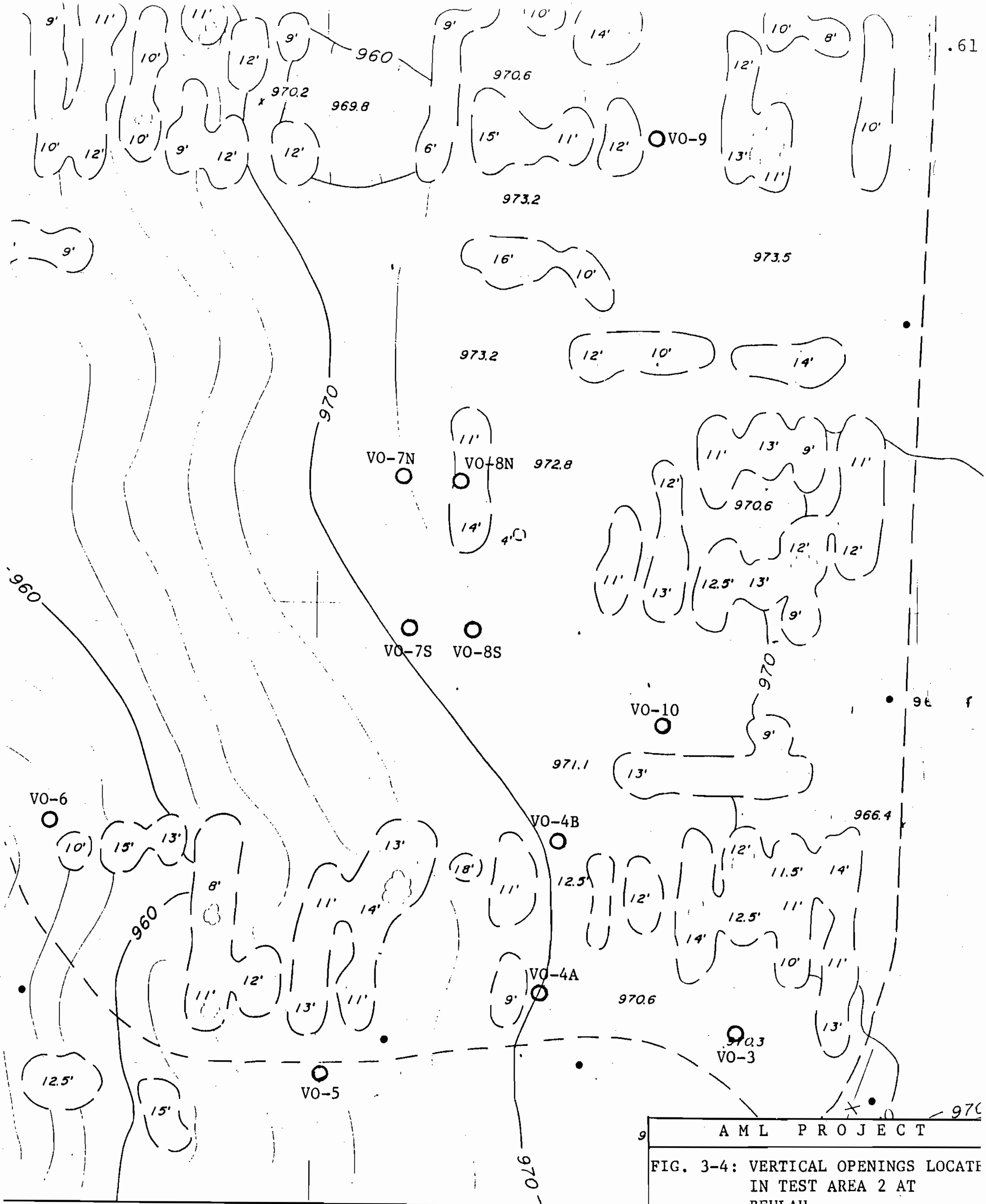

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FIG. 3-4: VERTICAL OPENINGS LOCATE
IN TEST AREA 2 AT
BEULAH

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room about 128 feet apart. Both sinkholes were open to the works below.

Opening VO-4A was a shallow depression about 40 feet long, 25 feet wide and open at the south end. The slope to the workings was 50 feet long and about 26 feet deep.

Opening VO-4B was oval in shape being 17 feet by 6 feet at the surface. The slope distance to the room below was 60 feet. Below surface the opening was as much as 30 feet wide.

The sinkhole designated VO-5 was oblong in nature with dimensions of 27 feet by 16 feet. The depression was shallow and open to the works at both ends. The opening at the north end was most prominent. The slopes down to the rooms were narrow (3-4 feet). The south opening was 50 feet long and the north opening 60 feet long, as far as these could be measured.

VO-6 was the sinkhole furthest to the west in test area 2. This opening was discovered later in the program. It was a circular feature which had settled about 6 feet. There was a narrow opening to the south. Because the opening was small one could not see all the way to the workings.

VO-7 was a long, narrow subsidence feature that was open at both the north and south ends. The feature was 64 feet long and 15 feet wide. The depth along the depression was about 8 feet. The slope distance that could be measured was approximately 60 feet at each end. The slopes were 11 feet high.

The sinkhole designated VO-8 was a long depression parallel to VO-7 and represented collapse of the room to the east of VO-7. This subsidence feature was 50 feet long and 15 feet wide. It was open to the works below at the north end. The slope to the works was 10 feet high, and 10 feet wide and approximately 60 feet long.

The sinkhole VO-9 was a particularly dangerous feature. It had a narrow, circular surface opening of 5 to 6 feet diameter. It dropped 20 feet to the rubble below and then sloped off in both directions to the room. The opening at it base was 30 to 40 feet wide so that the cavity was severely undercut. The surface opening was obscured by tall grass.

Void VO-10 was a near circular opening that was shallow in depth and had a small opening at the north end descending toward the works below. the depth of the overall depression was 6 feet. The length of the ramp to the works was unknown because it was narrow and could not be taped. The circular opening at the surface was 14 to 15 feet in diameter.

The remaining vertical openings were in test area 3. The location of these is shown on the map in figure 3-5. Sinkholes

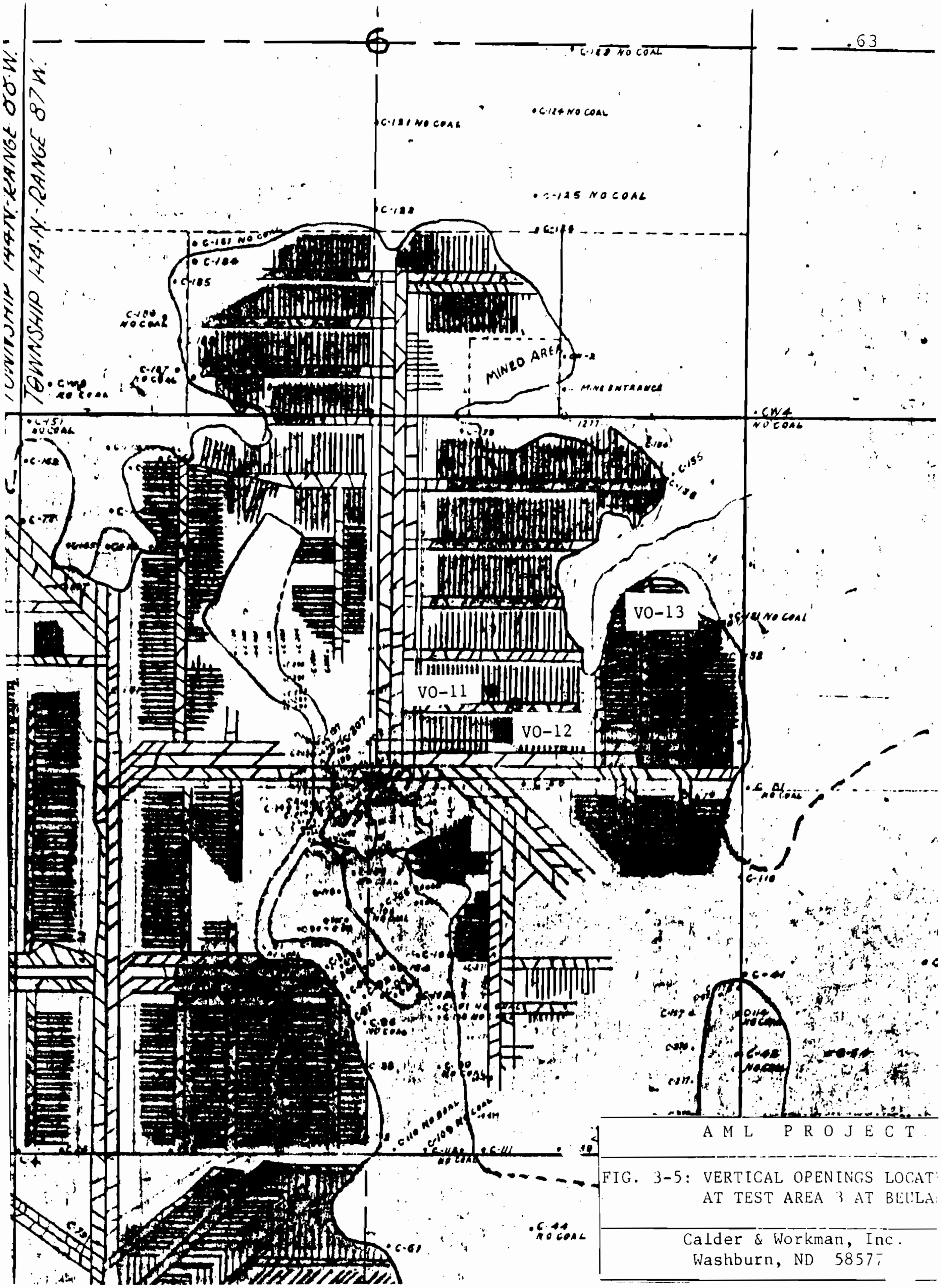

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FIG. 3-5: VERTICAL OPENINGS LOCAT
AT TEST AREA 3 AT BEULA

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VO-11 and VO-12 were quite close together at the south end of an area that had been previously reclaimed. VO-13 was a large subsidence along the side of a hill next to the coal outcrop.

The opening designated VO-11 was quite a large feature that had slumped to a substantial depth, especially in the vicinity of the open void into the works at the south end of the slump. The opening was 50 feet long. It was 32 feet wide at the north end and 22 feet wide at the south. The void was as much as 30 feet deep. At the north end it was as little as 5 feet deep.

Void VO-12 was south and east of VO-11 by 182 feet. This sinkhole was circular with a diameter of about 17 feet. It was open to the workings to the east at about a 30 foot depth. It was also open along the south side. In figure 3-5 one sees that this vertical opening was located over an entry and appeared to be near the intersection of the entry and a room. The west side of the depression was slumped but did not show an opening to the void below. However, the drill intersected a void close to the surface just behind the west side of the opening.

Vertical opening VO-13 was a caved room which had been mined close to the outcrop. From the outcrop east there was a long section of collapsed room. At the east end there was a large, sloped opening fully open to the room below. The depth to the cavity was about 35 feet. The slumped room was approximately 65 feet long.

3.2.2 Field Exploration

Some field exploration was necessary to completely characterize the sinkholes to be shot. This program, however, was very modest relative to the exploration needed when caving underground rooms, for which there is no visual identification.

Usually exploration consisted of one or two holes drilled to the works below at a distance no more than 50 feet from the opening. The main purpose was to identify the trend of the room below in order that the production holes could be placed in the best position.

For example three exploration holes were drilled around VO-3 for a total of 114 feet. VO-6 required two holes, one at each end, for a total of 110 feet. Table 3-1 is a summary of the exploration footage drilled at the Beulah Site.

The table shows that a total of 35 exploration holes were drilled. The total footage was 1,487 feet for a cost of \$1,263.95. The cost is based on the drilling price of \$0.85 per foot. The average exploration cost was \$80.00 per vertical opening. The range in cost was \$29.75 to \$354.45. However, excluding VO-2 the range was much less being from \$29.75 to \$110.50.

TABLE 3-1: SUMMARY OF EXPLORATION DRILLING REQUIREMENTS AND COSTS AT THE BEULAH, NORTH DAKOTA SITE

Vertical Opening	Exploration Holes Number	Exploration Footage Feet	Exploration Drill Cost Dollars
VO-1	2	102	$86.70
VO-2	7	417	$354.45
VO-3	3	114	$96.90
VO-4A	1	45	$38.25
VO-4B	1	39	$33.15
VO-5	2	81	$68.85
VO-6	2	110	$93.50
VO-7N	1	40	$34.00
VO-7S	1	35	$29.75
VO-8N	1	38	$32.30
VO-8S	1	36	$30.60
VO-9	4	130	$110.50
VO-10	2	86	$73.10
VO-11	2	65	$55.25
VO-12	4	105	$89.25
VO-13	1	44	$37.40
Totals	35	1487	$1,263.95

The exploration procedure was quite straight forward. The engineer would examine the opening and, in particular, observe the trend of any visible rooms below. Then one or two exploration holes were drilled. The engineer examined the results of this drilling. If the information obtained was adequate to define the workings below then the production pattern was designed. If questions still remained additional exploration holes were staked out and drilled. Once the engineer was satisfied that adequate information had been received the production blast was designed.

The primary purpose of the exploration drilling was to ensure that one knew where the rooms below were when designing the blast. For blasting closed vertical openings this is important because there is a desire to minimize disturbance beyond the sinkhole in question. Therefore, one does not want to place blastholes where detonation of these holes could generate unexpected caving. Defining the works below is especially crucial when the sinkhole does not provide a clear view into the rooms below.

Another reason for defining the underground rooms is that the configuration of the vertical opening may require drilling beyond the crest of the hole. This occurs when the sinkhole has caved leaving a long, sloping ramp down to the open mined area below. To close this slope requires drilling beyond the surface expression of the hole and this requires additional knowledge of the underground configuration. Figure 3-6 illustrates this need.

3.3 WHITE SITE, SCOBEY, MONTANA

3.3.1 General Description

The White Site had been mined well into the past. The mine was not a major operation. Rather it was a small local mine providing coal for domestic heating use. There were no surviving mine maps and it is unlikely that much underground mapping was performed when the mine was active.

In August of 1988 a visit was made to the site to inspect the area. The site was found to be relatively flat and about one acre in size. Topsoil had been removed in order that the area could be reclaimed to crop use after regrading.

Two mining features were noted during the inspection trip. An adit was in evidence on the side slope which provided entry to the mine. Also, there was a sinkhole about 12 feet in depth toward the east side of the property. Other than these two indications of mining there were no surface disturbances in evidence.

During the preliminary visit information from a previous exploration program, sponsored by the State of Montana was

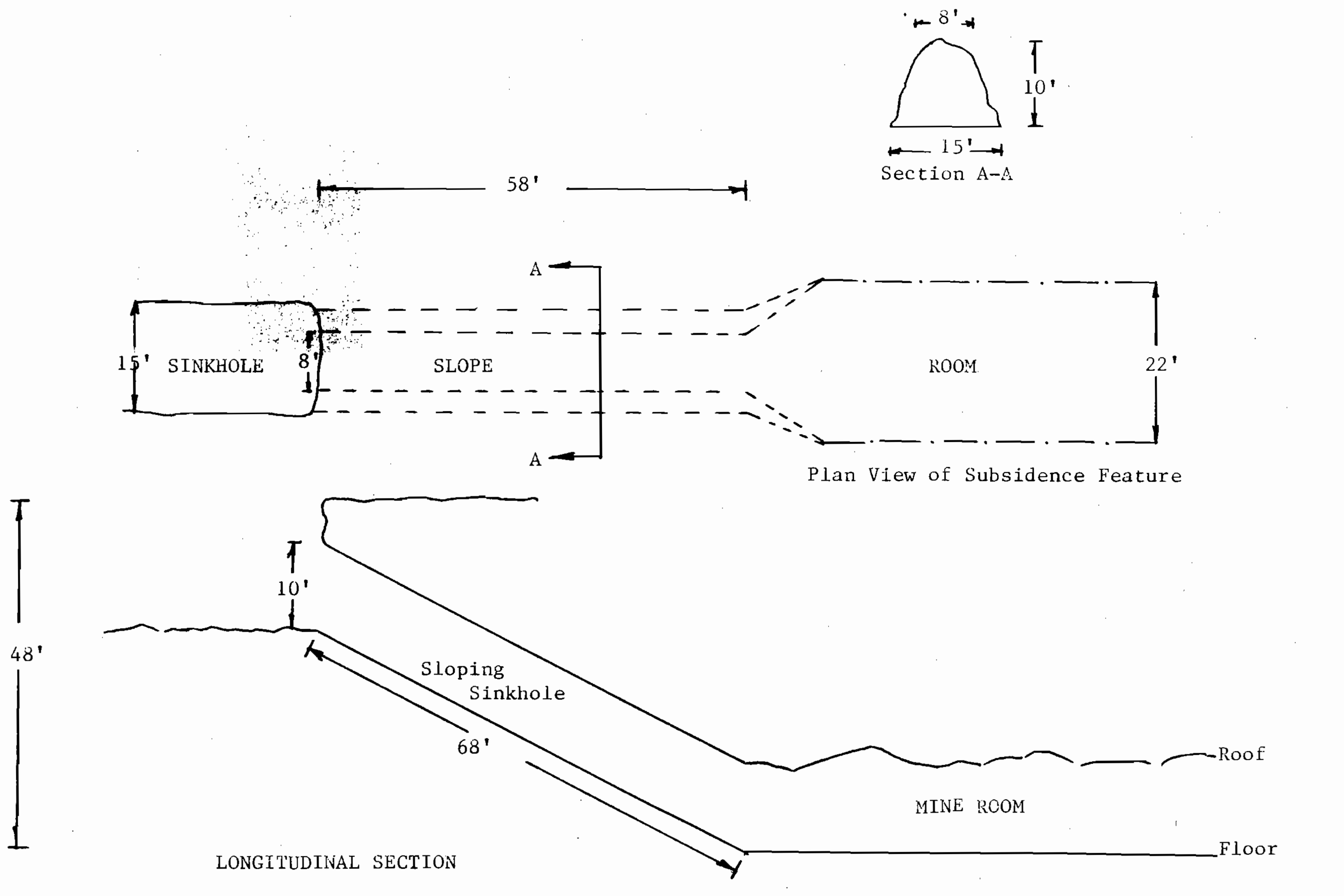


FIGURE 3-6: ILLUSTRATION OF A SINKHOLE WITH **A SLOPING ENTRY** INTO THE ROOM BENEATH. BLASTING **REQUIRED ALONG** THE SLOPE TO COMPLETELY FILL THE **VOID**

provided. This work consisted of some 23 exploration drill holes. A T.V. camera had been used in several holes to determine the trend of the workings in the vicinity of these holes.

The results of this survey had been mapped. This is shown in figure 3-7. The map shown, which was of a very preliminary nature, was the only map of the underground workings. The figure also shows exploration holes where the T.V. camera was used to further delineate the rooms and pillars.

The scarcity of information meant that the majority of the exploration had to be performed during the current project. The additional work to be done included exploration drilling and extensive use of the T.V. camera where conditions permitted. Exploration and production work were ongoing simultaneously, with the exploration staying out in front of the production operations. The White Site was therefore very different than the Beulah Site from the standpoint of exploration requirements and costs. The additional exploration on the White Site was underwritten by the State of Montana.

3.3.2 Preliminary Studies

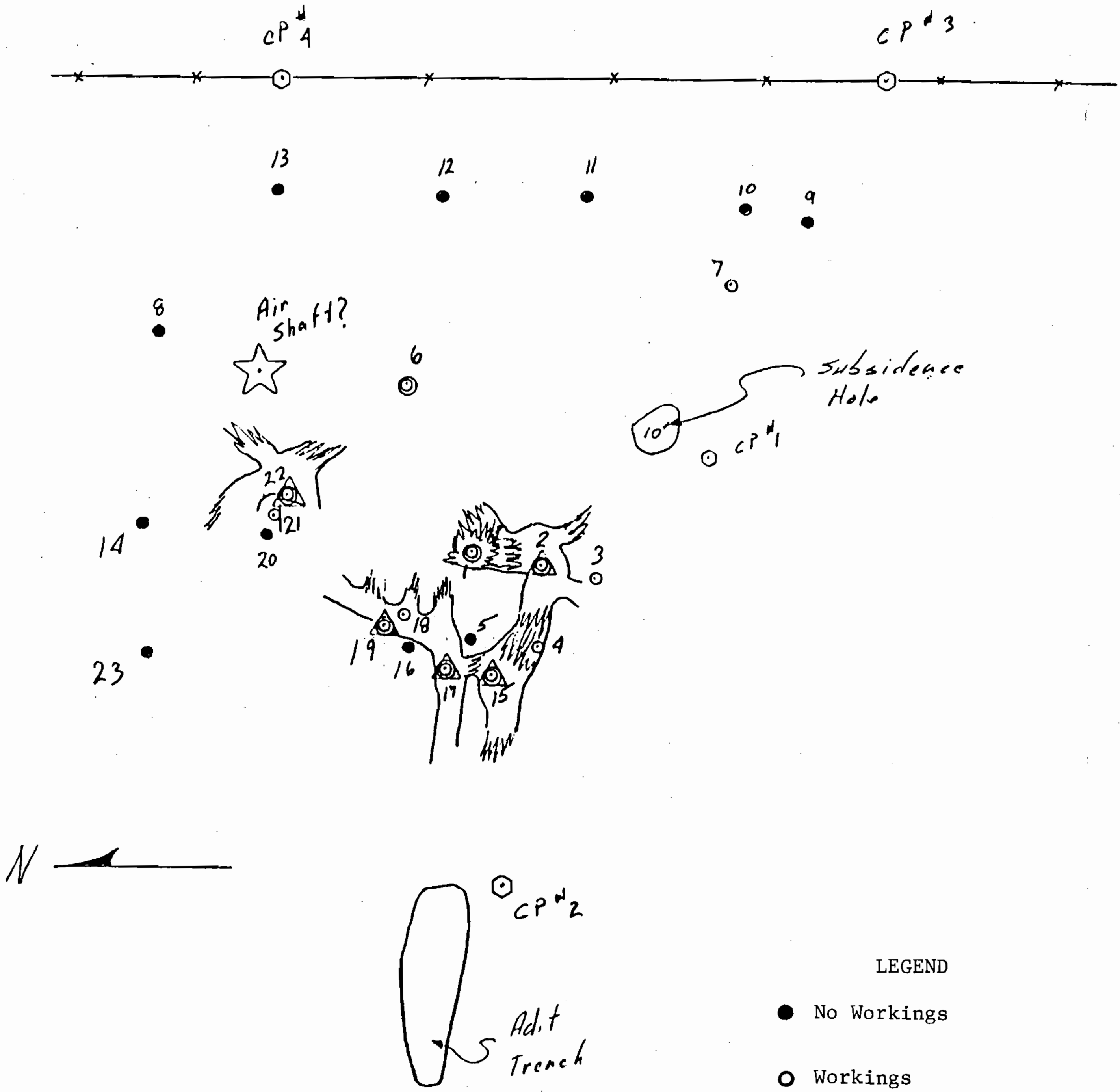
Unlike Beulah there was little preliminary investigation that could be performed at the White Site. As described above there was sparse information available to study and act upon.

What data there was had resulted from studies commissioned by the State a year previous. This information was made available for use in this research project.

Figure 3-8 is a map of the underground workings which had been projected based on the exploration work and the map shown in figure 3-7. Figure 3-8 was an attempt, based on limited exploration, to make preliminary estimations of how the property might have been mined. It was helpful in giving initial directions to the field exploration and research. Subsequent work resulted in major revisions to this map. This was not unexpected given the limited exploration on which the map was based.

A geological cross-section was also provided. This section was proved to be substantially correct by the detailed exploratory work conducted during the current research. The section is shown in figure 3-9. Compared to the Beulah Site there is more till and loosely consolidated clay. The deeper strata is overconsolidated clays similar to the Beulah Site. The differences were not great and preliminary indication was that the blasting design parameters would be similar to Beulah.

Having visited the site and reviewed the results of previous work decisions were made as to how to proceed on the



LEGEND

● No Workings

○ Workings

◎ 8" Dia. Borehole

△ T.V. Survey

Mine Workings Collapsed Area

FIGURE 3-7: EXPLORATION BOREHOLE LAYOUT AND PRELIMINARY INTERPRETATION FOR THE WHITE SITE IN DANIELS COUNTY, MONTANA

Scale 1" = 50'
W.F. Shaw
9-4-87

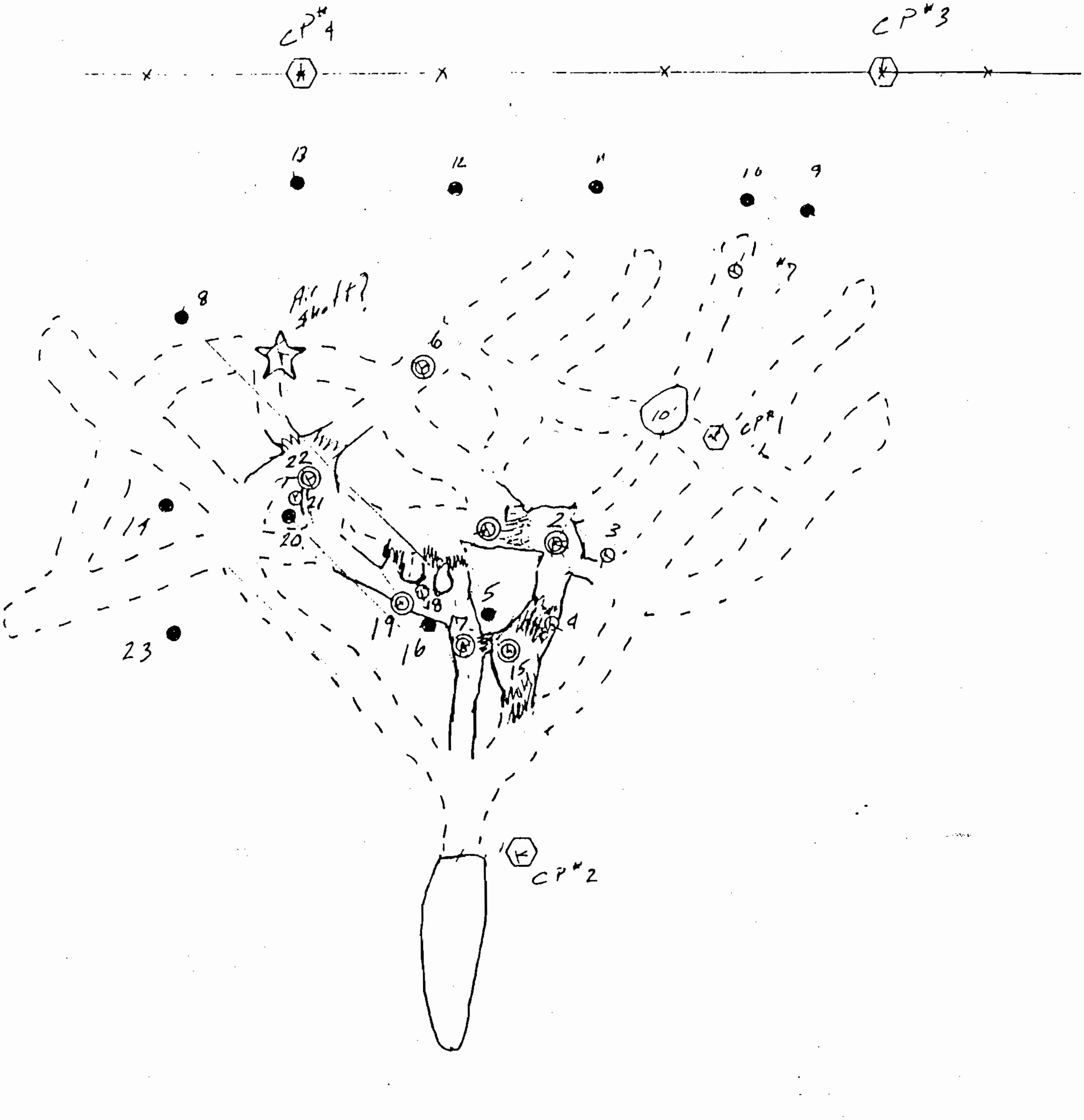


FIGURE 3-8: MINE MAP SHOWING A VERY PRELIMINARY VIEW OF THE UNDERGROUND WORKINGS BASED ON A SMALL EXPLORATION STUDY IN 1987

Scale 1" = 50'

W.F. Shaw

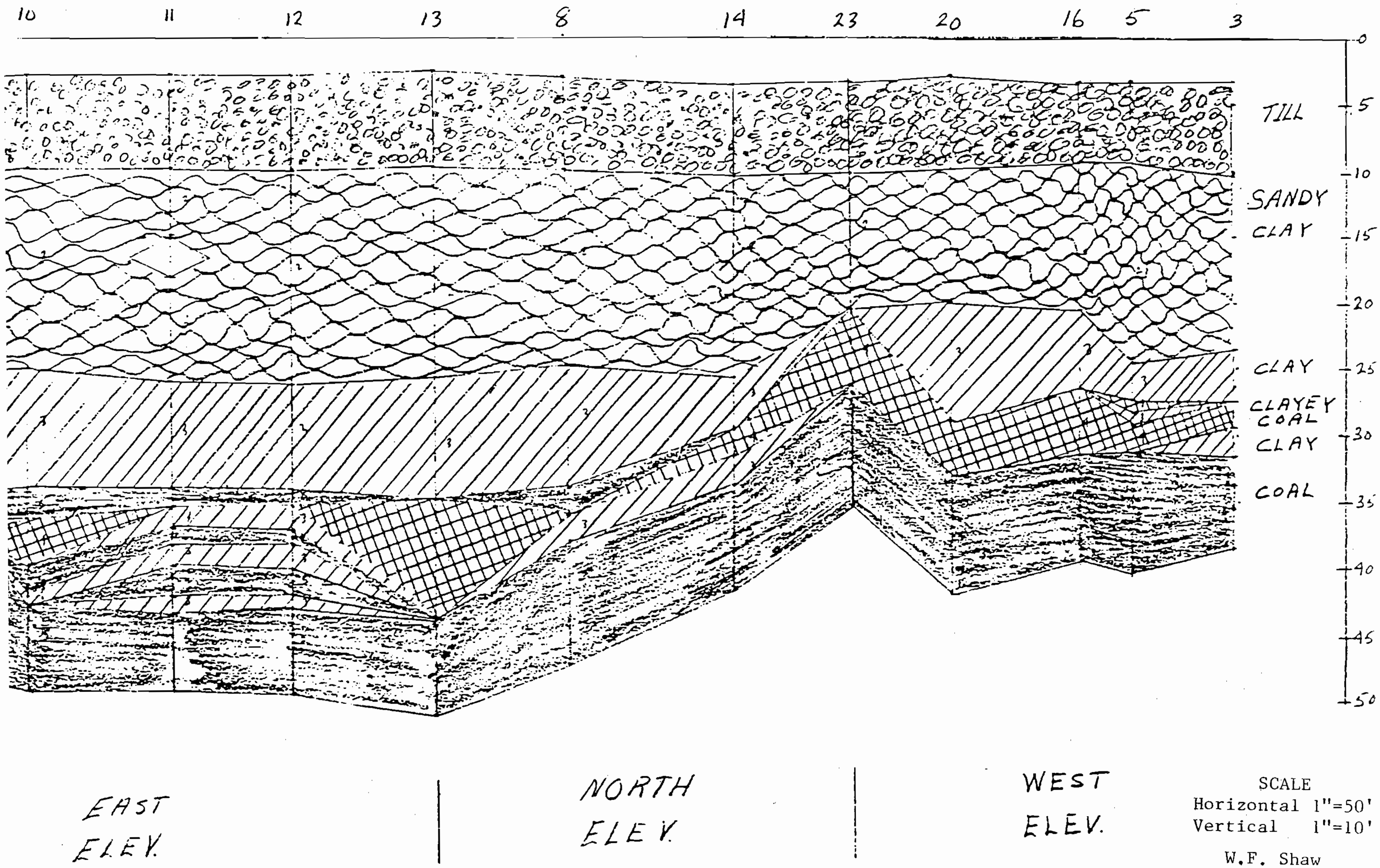


FIGURE 3-9: GEOLOGICAL CROSS SECTION OF THE WHITE SITE S""WING THE TYPICAL STRATA TYPE D THICKNESSES

property. It was decided to start exploratory work and subsequent blasting at the adit and work eastward. The primary reason for this was that the adit was a visible feature and therefore a good point of departure. Also, previous analysis of the exploration results was most detailed near the adit, which would assist initial interpretation. Finally the east end of the mine was water filled and it was considered appropriate to blast the wet workings after experience had been gained at the site.

3.3.3 Field Exploration Studies

Field exploration at the White Site consisted of drilling and viewing the underground works with a T.V. camera. Where the underground rooms contained water drilling was solely relied upon because the camera setup could not work in wet areas. However, it was possible to use the camera for a significant portion of the site.

The basic procedure used was to drill a hole that intersected the old workings. Then the T.V. camera was lowered into the hole and the trend of workings proceeding away from the hole was determined. From this information further exploration holes were staked out. In some cases these were camera holes while in others the purpose was simply to confirm continuity of the void.

When the rooms were water filled and the camera could not be used then drilling was relied upon. Because the workings were mined in such an irregular way drilling was a process of elimination. A few holes could be staked out and drilled. After the results were assessed additional holes could be staked out.

A total of 146 exploration holes were drilled at the White Site. Twenty-three holes were drilled in the prior exploratory program. Many of these holes could still be used for camera work. Five of the holes were used on a post-blast basis to study the extent of caving from the shot. The 146 holes represented a total footage of about 5,855 feet. This was a much greater exploration drilling requirement than that needed at Beulah.

The T.V. camera system used at the site was provided by the State of Montana through L. C. Hanson Co. of Helena, Montana. The system had been developed by Hanson. It consisted of a T.V. camera of the same type used in bank and store surveillance systems and an industrial monitor.

The T.V. camera was mounted on a bracket and could be swiveled through 90° by pulling an attached wire. A tripod was mounted over the hole and the camera and cables were lowered down the hole by a pulley arrangement. When lowering the camera it was swiveled to a vertical position with its long axis aligned with the axis of the hole. When it had been lowered into the void the wire was pulled to rotate the camera into the horizontal position for viewing. It was returned to the vertical position for extraction from the hole. In this way the unit could be lowered and raised easily in a 8-inch diameter hole. Twenty-two 8-inch diameter holes were drilled.

The tripod had a azimuth scale afixed to it. Therefore, the direction the camera was oriented could be determined as the void was studied. The monitor had a canvas hood to protect against glare and dust.

To make the underground workings visible to the camera it is necessary to provide light. This was done by lowering a spot light down a nearby hole to light up the area. The light was usually lowered down a hole at some distance away so that the area between could be viewed. If no light showed up it indicated there was no connection between the holes. Sometimes there was a connection but rubble in the void obscured the light so that viewing any detail was difficult. To power the lights and the camera setup a gas generator was used.

Once the camera was lowered and rotated into position the trend of workings radiating away from the hole could be determined. The distance that could be viewed depended on conditions underground. If there was considerable collapse and rubble this often blocked the view. Where the workings were more complex as where there was a corner or more than one room viewing distance was reduced. Well preserved, individual rooms could be viewed best. Also, dust would obscure viewing but this would usually settle. The ultimate factor limiting the distance that the camera could see was the strength of the spotlight used to light up the room.

The range in which the workings could be viewed was generally of 10 to 25 feet. There was no scale by which to judge distances. However, it was observed that an experienced operator could estimate distances quite closely in most cases.

Where it could be used the camera arrangement was valuable for finding the rooms and pillars. There is no doubt that the availability of this equipment significantly reduced drilling requirements and added to the understanding of the underground configuration. On the other hand use of the equipment can be time consuming. Also, the cost of the equipment and geologist is incurred. Further, because the exploration drilling is stop and go the drill contractor finds it necessary to charge a greater cost per foot to account for the downtime.

As a result of the combined drilling and camera work it became possible to draw a quite representative map of the underground workings. This map is shown in figure 3-10. The

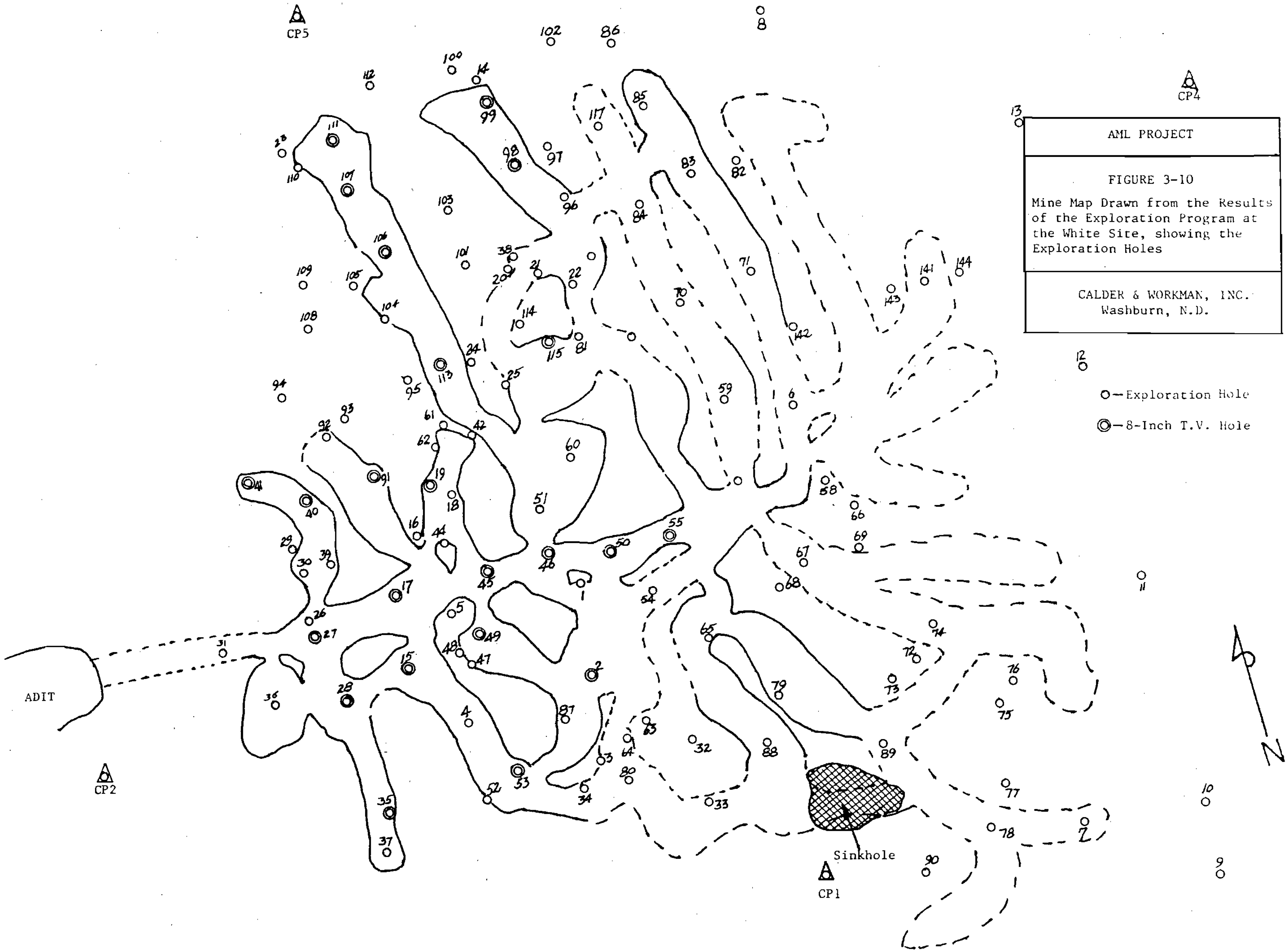


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FIGURE 3-10

Mine Map Drawn from the Results of the Exploration Program at the White Site, showing the Exploration Holes

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exploration holes are marked on this map. At the east end exploration was limited. In this section pattern blasts were shot so less detail was needed. The final shape of the works was filled in from the results of the production drilling (by observing which production holes encountered voids). This area is the least well defined of the mine workings.

In the figure the holes with the double circle were those in which the T.V. camera was used. The remainder are exploration holes from which void/no void information was obtained. Some of the holes were used for lowering the light source.

Examining figure 3-10 one can see that the area was mined with limited advance planning. Both the rooms and pillars are quite variable in dimensions. Some of the intersections show very large spans. Many pillars have been robbed until these provide minimal support. Were the cover deeper numerous of the pillars would have failed long ago and there would have been much more surface subsidence.

A major concern in such a setting is the backbreak that will occur from a blast. The opportunity for unwanted, incomplete collapse is great. Blasting techniques need to account for this problem if the project is to be successful. When the approach taken is to blast in the rooms it will be necessary to characterize the underground workings quite carefully in order to make good decisions about the blast configuration.

3.4 EXPLORATION COSTS

3.4.1 Beulah Site

The costs for exploratory drilling at Beulah, North Dakota were listed in table 3-1. In this case the exploration drilling was charged at $0.85 per foot.

Referring back to the table it can be seen that the exploration costs are modest. The average cost of exploration per vertical sinkhole was $80.00. However, as can be seen there was significant variation in the cost from sinkhole to sinkhole.

The primary factor affecting the amount of drilling is the extent to which the sinkhole is open to the room below. Where the opening is large and there is a good view the trend of the room can be more readily anticipated and drilling reduced. When the view into the workings is obscured more drilling may be required. Also, if the sinkhole is open at both ends more drilling may be needed to determine the direction of the rooms at both ends of the subsidence feature.

At Beulah the underground workings are systematic and accurate mine maps exist. This helps to reduce the exploration cost. Where the underground rooms are not regular the amount of drilling and therefore cost will increase. This increase will be 50 to 100 percent of the footage and cost experienced at Beulah. Therefore, when estimating costs prior to starting the project the nature of the mining must be considered.

The cost will also be dependent on the depth to the underground mine. At least as many holes will be required in deeper cover than more shallow overburden. In fact, because the sinkhole is not likely to be as open to the workings in deeper cover more exploratory holes may be needed, further increasing the footage. This is especially true where the workings were not mined systematically. At Beulah exploration hole depths typically ranged from 30 to 60 feet.

When reclaiming individual vertical openings costs of exploration will be modest. The cost will typically range from \$80 to \$160 per sinkhole on average. When the slump is open to the mined room two holes will usually be adequate. Where the view of the room is obscured or the workings irregular four exploration holes may well be typical.

The above cost is for drilling only. There is also an engineering cost for examining the hole, placing stakes for the drillers and evaluating results. Our experience at Beulah was that these tasks did not exceed one to two hours per sinkhole. Therefore, the cost of technical labor can be estimated at \$40 - \$80 per sinkhole

Also there are some supply requirements needed for the exploration of each vertical opening. This includes stakes, paint, tapes and other expendables. The cost of these supplies can be estimated at about \$5.00 per sinkhole.

Therefore, the total cost per sinkhole will vary from \$115 to \$195 on average. Where there are regular workings and good mine maps estimate the lower cost. If the workings are not systematic and there are no reliable mine maps then the higher average cost should be estimated.

3.4.2 <u>White Site</u>

Exploration at this site was considerably more costly. As described above it consisted of a combination of drilling and T.V. camera observation. The drilling was extensive to characterize the works. Use of the T.V. camera was also extensive to view the workings and to determine directions for additional drilling.

The cost of exploration drilling was also more expensive at this site. One was made aware that as much as \$6.00 per foot had been paid for previous work. The reasons are that the drilling is often stop and go due to the need to use the camera and downtime for the drilling contractor is much greater than for production drilling, or on a site like Beulah where no camera work is involved. Also some 8-inch diameter drilling was involved.

In the current case the drilling cost per foot did not reach $6.00 per foot. Both regular 4- or 6-inch holes were drilled and 8-inch holes were drilled for the T.V. camera holes. The regular exploration holes were charged at $2.00 per foot while the 8-inch holes cost $4.00 per foot. A total of 5,855 feet of exploration drilling was completed. Twenty-two holes of 8-inch diameter were drilled representing 880 feet of drilling. The remaining 4,975 feet was regular exploration work. This represented 124 holes.

The T.V. camera and geologist were on site for approximately 18 days. The camera was not in use all of the time and the geologist had project inspection duties for the State as well as performing the exploration work. However, about 75 percent of the time was spent on site characterization. The T.V. camera and geologist were contracted by the State of Montana through L. C. Hanson Co. and exact costs are not available. A reasonable estimate is that a geologist and equipment for this work would cost about $350 per day.

The engineer also spent time helping to interpret results, stake holes and map the results. This time is estimated at 80 hours. There were also field supplies used in exploration similar to expendables used at Beulah.

Table 3-2 lists the exploration costs at the White Site. It can be seen that these are much greater than for closing sinkholes at Beulah.

The total cost was $21,935. The mobilization charge was one-half the total. The remainder was charged to production studies. The T.V. camera work is estimated on the basis of 75 percent of 18 days having been spent on exploration work. Engineering time was not broken out by exploration and production because these often overlapped, however, 80 hours is a close estimate of the time spent.

The exploration is in addition to the previous program in 1987. This program cost several thousand dollars. Combined the two would yield a total exploration cost of about $30,000.00.

The exploration was conducted on approximately one acre of mined land. Thus the exploration work is a major cost for the total reclamation effort. For many projects it is questionable if such costs can be supported. An alternative is to use an area blasting approach where the block is drilled off on a regular pattern. Then the exact location of the workings is not as important. Holes are loaded according to whether they intersect a void or not. Exploration could be reduced to defining the perimeter of the mine and some internal drilling to give an approximate idea of where the rooms are. The T.V. camera work would not be necessary. Production holes could also provide exploration information. Pattern blasts were tested during this project as a potential way to reduce exploration time and overall costs of a project of this type.

TABLE 3-2: EXPLORATION COSTS AT THE WHITE SITE, DANIELS COUNTY MONTANA TO CHARACTERIZE AND MAP THE UNDERGROUND MINE WORKINGS

Item	Quantity	Unit Price, $	Total Cost, $
EXPLORATORY DRILLING			
4- or 6-inch	4,975 ft.	2.00	9,950.00
8-inch	880 ft.	4.00	3,520.00
mobilization	1 ea.		500.00
T.V. CAMERA STUDY			
Equipment & Geologist	13.5 day	350.00	4,725.00
ENGINEER	80 hours	40.00	3,200.00
FIELD SUPPLIES	1 ea.		40.00
TOTAL COST			$21,935.00

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TOTAL COST			$21,935.00

CHAPTER 4: FIELD PROCEDURES AND TECHNIQUES USED AT THE BEULAH TEST SITE

4.1 BLAST DESIGN AND IMPLEMENTATION

4.1.1 Preblast Design

Each vertical opening was somewhat different from the others. These subsidence features were of three types which were:

1. Circular slump into the works below. There might or might not be an opening into the room.

2. Sinkholes that were long, having dropped into the room over a significant length and with a sloping opening into the workings at one or both ends.

3. Features with a small surface opening and a large subsurface area causing the vertical opening to be undercut.

These three types of subsidence features had to be treated differently for blast design. In addition there were always some individual differences between openings so each had to be designed specifically.

The first step, therefore, was to examine the individual sinkhole in the field. Measurements of the feature were referred to, to give a rough estimate of the void volume to be filled. Reference was also made to the mine maps to give some idea of what might be expected upon blasting.

Once the blasting engineer was satisfied that the opening was as well characterized as possible the best plan for filling the sinkhole was developed. The preblasting design included considering the volume to be filled, the burden on the holes, the spacing between holes, the hole depths, the rows of holes needed and the layout of the holes and the explosives loading.

4.1.1.1 Volume to be Filled

The volume of a sinkhole was difficult to measure exactly. The openings tend to be of variable geometry. A goodly proportion of the hole was inaccessible for safety reasons so that exact measurements could not be taken. Also, the volume needed to fill the void was partially dependent on how much collapse might occur in the room associated with the vertical opening. For these reasons the volume determinations were approximate. Considerable field judgement was required when laying out the pattern to obtain enough blasted material. The detailed work on swell in the previous research (4) report was quite helpful in assessing the needs for the vertical openings.

The only way to further increase the accuracy of the volume calculations would be to employ photogrammetry. Stereo pairs of the sinkholes could be taken and analysis equipment used to determine the dimensions and volume of portions of the holes where access was not possible. The success of this approach would depend on being able to obtain adequate photographic exposures in bad (and variable) lighting conditions. Also this approach would add significantly to the cost of the project and represents a level of sophistication not required for other methods of mitigating these features.

A factor to consider when determining volumes was the backbreak to be expected. During previous work we observed backbreak in these plastic materials of 10 to 12 feet. Therefore, it was reasonable to assume that there would be about 10 feet of backbreak around the sinkhole being blasted. Since much of the material in front of the blast holes was thrown into the opening the backbreak was quite helpful in filling the volume vacated by the thrown overburden.

4.1.1.2 Burden on the Holes

As stated in Chapter 2 the primary goal was to cast overburden from around the void into the open hole, while also maintaining adequate material in the blast zone so that the overall result was a configuration ranging from a shallow depression to a slight rise in the topography. Therefore, the holes had to be placed close enough to the opening so that, upon detonation, considerable velocity could be imparted to the material, thereby casting it into the opening. At the same time the holes had to be placed far enough back from the vertical opening to ensure that the burden would contain the explosion. Otherwise loss of performance would result due to bursting of gases through the face and airblast would be increased.

In Chapter 2 the scaled depth of burial (SDOB) for the charges was discussed. Scaled depths of burial of 2.0 to 2.5 $ft/lb^{1/3}$ were expected to work best. However, higher up the column the SDOB could be somewhat greater but should not exceed 3.0 $ft/lb^{1/3}$. At the collar the SDOB should be adequate to avoid excessive flyrock and noise and to reduce cracking of the surrounding overburden.

With these parameters in mind it was determined that the holes should be placed 8 to 10 feet back from the sinkhole. Usually a heavy ANFO was used for the bottom deck which gave a scaled depth of burial of 2.0 $ft/lb^{1/3}$ at 8 feet and 2.5 $ft/lb^{1/3}$ at 10 feet. ANFO was used in the upper decks yielding a 2.3 ft/lb$^{1/3}$ scaled depth of burial at 8 ft and 2.8 ft/lb$^{1/3}$ at 10 ft. These designs were expected to give good results and in fact this proved to be the case.

Older sinkholes often have gently sloping sides. Therefore depths of burial at the toe can be large. However, the vertical openings blasted in this study were no more than two years old. Therefore, the depressions were steep sided and burdens at the toe and crest were usually quite similar. Thus, holes can be well placed around sinkholes that are new. For older features angle drilling may well be appropriate. Drills that can angle drill are not generally as available on a contract basis, however.

4.1.1.3 Spacing Between Holes

Decisions had to be made about the spacing between the blast holes. Often, it is found that a spacing of 1.5 times the depth of burial works well. For an 8-foot burden this would suggest a 12-foot spacing for example.

To close vertical openings it was believed that the spacings should not be too large. The reason was to insure good movement of all the overburden into the void. Therefore the spacing chosen between holes was often 10 feet. This gives spacings of 1.0 to 1.25 times the depth of burial. Spacings of 12 feet were also used where it was believed that the holes had good relief and the material would displace easily. The 12-foot spacings were 1.2 to 1.5 times the depth of burial. The spacing should not be less than the burden.

It must be recognized that blasting in vertical workings is different than blasting on a mine bench. Here the geometry is particularly variable. Therefore, holes must often be spaced at a distance different than what seems optimum. The goal however is to stay as near to optimum as the situation allows.

Hole spacings also vary because of the need to provide a safe environment for the drill crew and their equipment. Because of the variable nature of the openings it was not always possible to place the drill exactly where one might wish. This is especially true when the sinkhole is of the type that is undercut.

The basic goal was to space the holes at about 1.25 times the depth of burial. The 10-foot spacing also provided a scaled crater radius of about 1.50 ft/lb$^{1/3}$ which was deemed adequate in most cases. When the situation required, adjustments were made to the design while attempting to stay as close to the basic goal as possible.

4.1.1.4 The Hole Depths

The depths of the holes were governed by the height to the workings. The depth to the rooms was determined by measurements with a 100-foot tape and from exploration drilling. Exploration drilling usually provided the most accurate data.

In some cases the hole depths were determined by other factors. For example, some sinkholes were long and narrow with an opening to the works at one or both ends. The depression along the room was usually shallow (12 to 15 feet deep). Since the area below was already filled with rubble there was no point blasting below the surface of the depression. Material below the surface could not be thrown into the void in any event. Therefore in these cases the holes were only drilled to about 2 feet below the surface of the slump.

Where the sinkhole was open to the room it proved generally necessary to drill a depth similar to the depth to the workings. This provided enough swelled overburden to fill the area without having to go too far back from the openings to obtain the material. The further back one goes the harder it is to get the broken rock to move into the void.

4.1.1.5 Rows of Holes Required

Wherever possible the blast was designed to have only one row of holes around the opening. However if the void was particularly large more material would be necessary to fill the depression. Then two rows of holes would be needed.

The prime example of this was VO-11. This was a large sinkhole with a substantial opening into the rooms below at the south end. It was determined that considerable volume of material would be required to fill this opening. Therefore, two rows of holes were used. The result was a large vertical opening that was successfully filled to within a few feet of the surface. One row of holes would surely have left a much deeper depression as less blasted overburden was generated. For VO-11 the two rows were staggered which worked well.

4.1.1.6 Hole Layout

Once the factors above were determined a plan could be produced for the blast layout. Burdens and spacings would be as determined above. The hole depths were obtained from exploration and observation of the vertical opening.

Frequently the sinkhole was a sloping ramp into the workings below. This meant that the blast had to be laid out along each side of the slope until it intersected the works. Hole depths varied along the slope.

Holes were not always drilled over the old works. The concern was that the blast detonated above the works might cause unwanted caving of the room beyond the extent of the vertical opening. This need was another reason for the exploration described in Chapter 3. In other cases however drilling above the works was necessary. Blasting in a circular sinkhole was the usual case when this was required.

Figure 4-1 is a typical blast layout for a sinkhole with a sloping opening to the room below. It shows the typical layout of the holes around the vertical opening and along the room to the extent of the sloped void.

4.1.1.7 <u>Explosive Loading</u>

Once the hole depths were established the explosive loading could be designed. The blasthole depths were double checked after drilling because there were sometimes changes and the loading could be modified accordingly.

To be a good approximation of a spherical charge the length of the explosive decks needed to be not more than 4 feet. The distance between charges should be about 6 feet to avoid cross propagation problems.

In practice some variations on these parameters cannot be avoided. This is because the decked charges must fit into the length of blast hole available. Therefore, some explosive decks will be longer or shorter as required to fit the geometry. This is also true of the deck stemming employed.

The maximum number of decks used in this research was three. Often only two decks were required and in some cases only one deck was loaded. Each deck was independently delayed.

The bottom deck was often PowerAN, a heavy ANFO product. This was loaded to reduce the scaled depth of burial in the toe of the hole to ensure that the overburden around the toe would displace well. Also, this 70 percent emulsion product was waterproof and would protect against any water in the bottom of the hole.

The first explosive deck was loaded directly into the bottom of the blasthole. The next was loaded at a designed distance up the hole The collar deck was kept 9 feet below surface and the hole was stemmed to the top with drill cuttings. Figure 4-2 is an example of a typical hole load. As stated the number of decks was dependent on the hole depth. The figure shows a case where three decks were used.

4.1.2 <u>Blast Related Work</u>

Once the design worked discussed above was complete the blast could be staked out around the hole and drilled. The blast holes were staked using three-foot lathe board. The hole

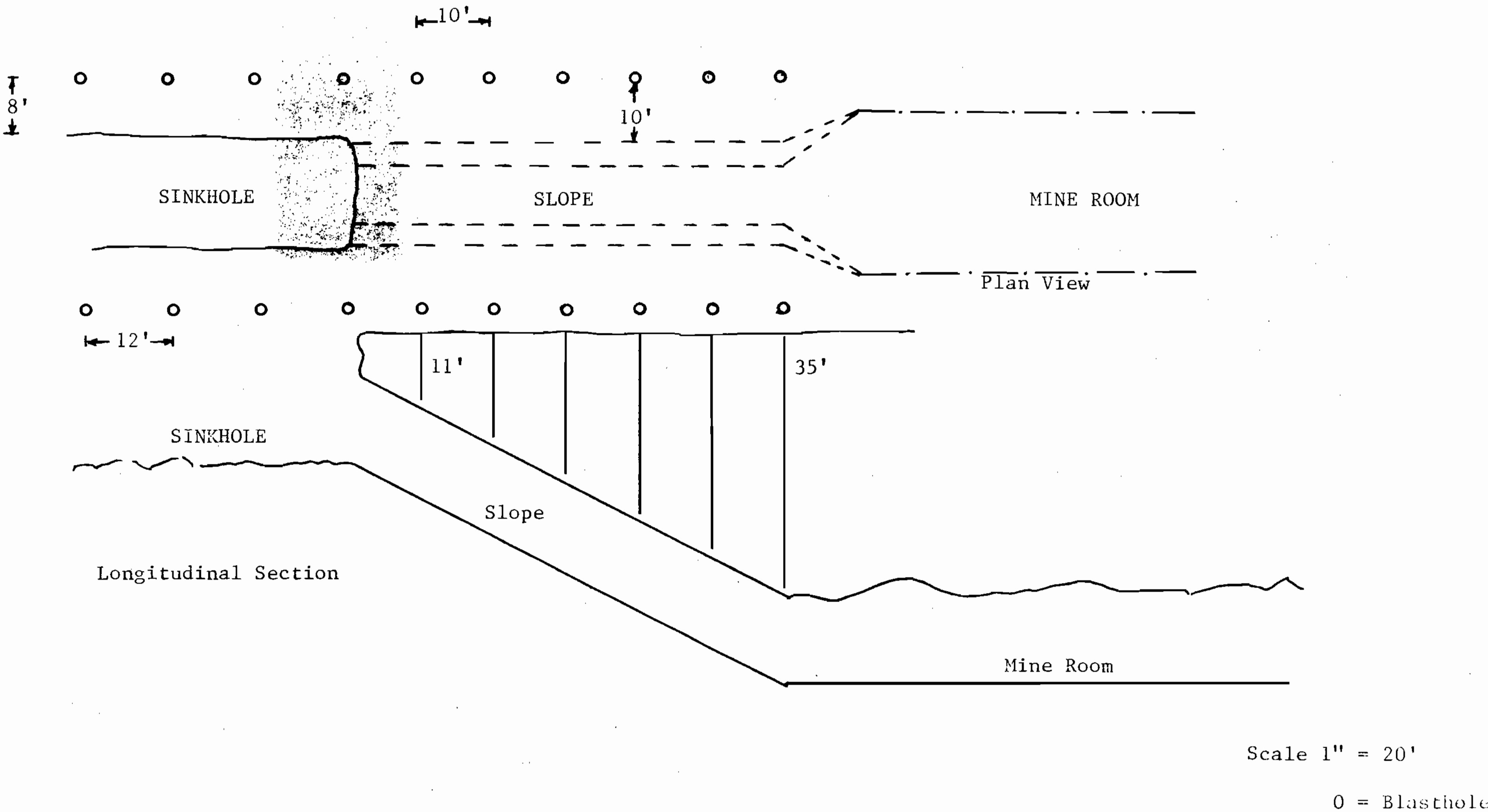


FIGURE 4-1: ILLUSTRATION OF DRILL PATTERN AROUND A SINKHOLE WITH A SLOPING RAMP INTO THE MINE ROOM BELOW

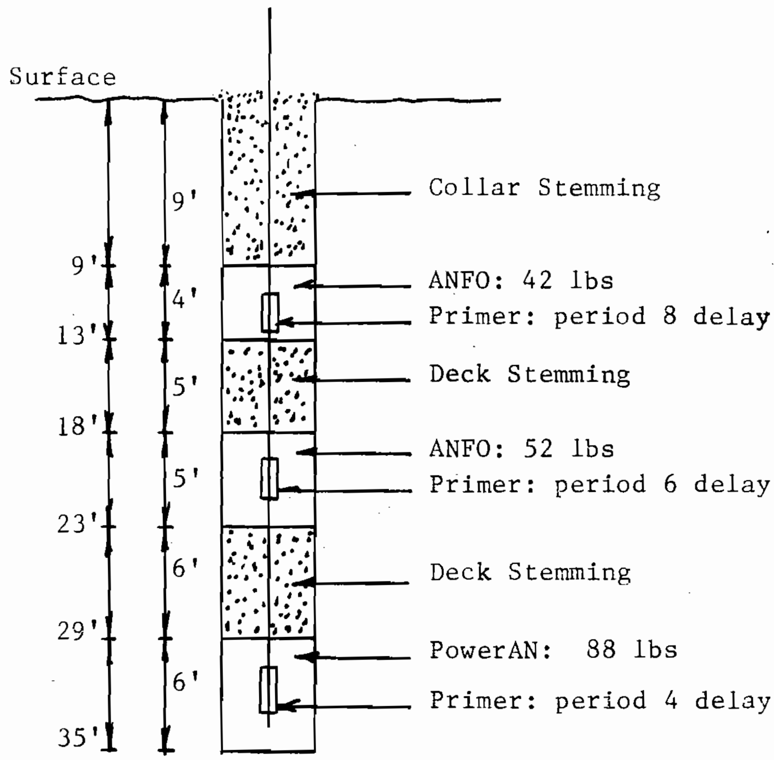


FIGURE 4-2: EXAMPLE OF BLASTHOLE LOADING USED AT THE BEULAH TEST SITE

Scale: 1" = 10'

number and the designed depth was written on the stake with a felt pen.

The drill crew then drilled the blastholes as specified. In some cases the void was intersected at depth. These blastholes had to be subsequently plugged or else drilled at a different location. Drilling normally took less than a day. Blasthole drilling was often one or two vertical openings ahead of blasting which allowed flexibility in the blasting operation. Little difficulty was experienced with the holes caving in after they were drilled so vertical openings could be drilled off in advance without problems. Figure 4-3 shows the drill operating around a sinkhole. The lathe board used to stake out the blast holes can be behind and in front of the drill.

Once the holes were drilled and prior to loading each one was taped to ensure that it was the correct depth. This information was recorded on the blast summary sheet. When a void was encountered it noted. In some cases the hole was moved, but more often it was plugged with a seismic cone and loaded.

Once the holes were taped a loading chart was drawn up for the blast. This chart showed the deck loading locations for each hole depth encountered in the blast. The pounds of ANFO and PowerAN were also shown and the primers, delays and other accessories were listed. This facilitated taking material from the magazine without removing more than necessary for the shot. Table 4-1 is a typical loading chart. This example is the chart used for vertical opening VO-5. The table is typed for clarity; in the field these charts were simply hand written by the blasting engineer. The chart was then referred to as each hole was loaded to get the correct powder loads in each hole.

If there were variances to the plan during loading these were noted on the chart by the blasting crew. For example in the case presented in the table only 12 bags of PowerAN were used. Also only nine number 6 period delays were used and 22 primers were needed rather than 23 as planned. Recording the changes meant that an exact record of the loading was available and magazine inventory requirements were more easily met.

With the load chart prepared the blast could be shot. The necessary explosives were loaded into the blasting pickup as were the required accessories. A cap box was provided in the truck for carrying caps and delays. The vehicle was provided by the explosives vendor and met all regulatory requirements. The explosives were transported to the site and distributed at the blastholes.

The holes were loaded according to the plan. First the blast summary form was referred to. The depth to be loaded was shown for each hole. If the hole was deeper it was backfilled.

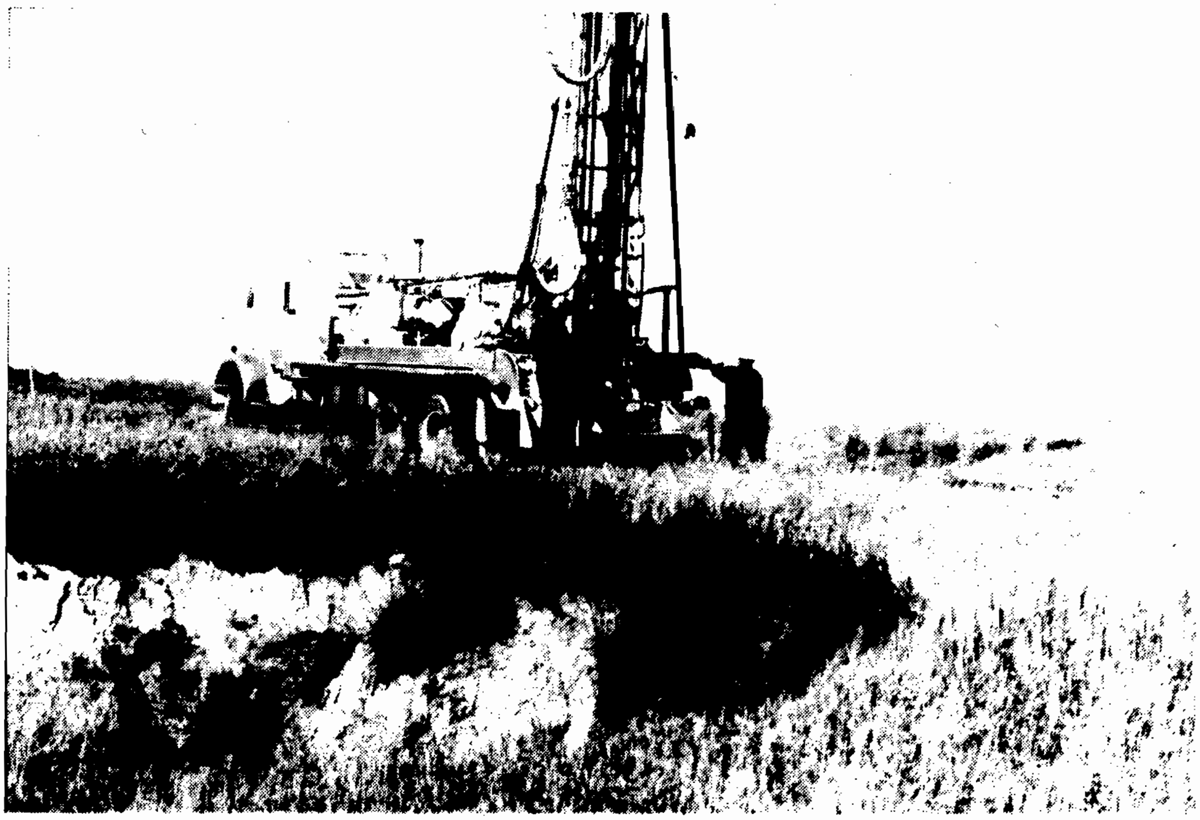


FIGURE 4-3: BLASTHOLE DRILL COMPLETING A PATTERN AROUND A SINKHOLE

TABLE 4-1: A TYPICAL LOADING CHART PROVIDED FOR EACH BLAST AT THE BEULAH SITE

VO-5

L O A D I N G   C H A R T

VALUES IN FEET

0	0	0	0	0	0	0
Stem	Stem	Stem	Stem	Stem	Stem	Stem
9	9	9	9	9	9	9
ANFO	ANFO	ANFO	ANFO	ANFO	ANFO	ANFO
14	13	12	12	16	15	14
Stem	Stem	Stem	Stem			
20	18	15	14			
PowerAN	PowerAN	PowerAN	PowerAN			
26	23	18	17			

ANFO:	500 lbs.	12 bags
PowerAN:	508 lbs	17 bags
Delays:	#4	13
	#6	10
	NTD 42ms	13
Primers:	23	
Caps:	1	

if it had struck a void this was recorded and the depth at which to locate a seismic plug was stated. The plug was then placed at the specified location.

The seismic plugs are used in the oil exploration industry when sealing drill holes. These devices come in various diameters. The base of the device is a plastic cone. At the top plastic leaves radiate out in a circular fashion around the cone. When placed in the holes the leaves grip the borehole walls and hold the plug in place. The plug was placed in the hole using loading poles. It was placed about a foot above the void. Placed closer the unit tends to drop through to the works below. To ensure that it remained in place when the weight of explosive and stemming was loaded above bailing twine was fastened to the cone and tied off at surface. We have experienced no failures after explosive loading with this system.

Usually backfill was required above the cone. The typical amount was 1 to 2 feet. This was indicated on the summary as well. An example of a blast summary sheet is shown in table 4-2. Since this form is intended for blasting underground rooms as well not all columns are used then shooting vertical openings. The example shown is the summary for VO-5. Blast summary sheets for all shots are found in the appendices.

The form provides all information found from taping the holes and from the drillers logs. Rock layers are noted where they occur. When the sheet was reviewed by the blasting engineer other information was added about plugging, backfilling and loading.

In some cases something changes between the time the holes are first measured and the time they are loaded. For example, hole 2 caved further and could not be loaded as planned. The new depth was recorded and the loading adjusted accordingly. Clearly, it is important to tape each hole just before it is loaded.

Other information is recorded at the top of this form. This includes general information about the blast, quantity of explosives used, accessories used, high-speed camera data and seismograph data. The blast summary report was found to be a good way to organize the blast data and to communicate among the crew as to what was to be done.

Each blasthole was then loaded. The specified explosive was loaded in the lower deck. A one-pound cast primer was placed in the explosive. A Nonel down hole delay was placed in the primer. These were Trojan slider primers and once the delay was inserted the primer was slid down the detonating cord. The cord was RX Primaline or an equivalent product. The cord will detonate the Nonel pigtail of the Primadet delay but will not set off the primer. For the primer in the bottom deck

AML PROJECT - BLAST SUMMARY

BLAST #: VO-5

LOCATION: Test Area 2

DATE 8/16/88

HOLE DIAM 6"

NO. HOLES 13

EXPLOSIVES CONSUMPTION

ANFO 12 BAGS = 600 LBS SLURRY 12 BAGS = 360 LBS

BOOSTERS 22 42 M/S DELAYS 13 1 cap

DELAYS #4 13 #5 ___ #6 9 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 300' NW OF VO-5 PPV: _____ IN/SEC

CAMERA USED: ____ TARGET DELAY #: ____ FPS: ____

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1					18					
2			18	18	19	(Void at 14', plugged at 13' Load 12')				Void at 18'fill to 15'
3					23					
4					18					
5					17½					
6					23½					
7					29					fill to 26'
8			21	20	21		29	8		fill to 16'
9			17	16	17		20	3		fill to 14'
10					18					
11					18					
12					23					
13					18					

TABLE 4-2: EXAMPLE OF THE BLAST SUMMARY SHEET USED FOR RECORDING FIELD DATA AND PLANNING THE SHOT

the Primaline was knotted to the loop of the Primadet and the primer was lowered into the hole. As the bottom deck was loaded a weighted blasting tape was used to confirm the correct rise of the explosive in the hole.

Once the bottom deck was placed the planned quantity of stemming was shoveled in to provide the deck separation. Again the hole was bobbed with the blasting tape to insure that the rise was correct.

If more than one deck of explosive was needed more powder was now placed. While the bottom deck explosive was often PowerAN the upper deck(s) were almost always ANFO. The explosive was poured into the hole from 50 pound bags. The blasting tape was used to confirm the correct amount of explosive as before. Once all the decks were loaded the hole was stemmed to the surface using drill cuttings.

When the blast had been loaded and the crew was ready to shoot the holes were connected together according to the design produced by the blasting engineer. The surface delays used were 42 millisecond Nonel noiseless trunkline delays. With these surface delays and 50 ms between decks in the hole no deck was to shoot less than 8 ms after the previous deck.

The delay arrangement used provided good relief. The sequence was that the bottom deck in a hole fired first, followed by the next deck up in the previous hole, followed by the top deck two holes back if a three deck arrangement were being used. Then the sequence was repeated starting with the bottom deck in the next hole. This meant that the decks shot on a diagonal from top to bottom providing relief both in front of and to the side of the deck.

Before shooting each surface delay was buried under cuttings along with any detonating cord pigtails. The electric blasting cap was also buried once it was attached with black tape to the first Nonel delay. Burying these units was to reduce airblast and noise to the minimum level.

Before the blasting cap was wired in the seismograph was set up in a suitable location and programmed to record the ground vibration and airblast from the shot.

In most cases the high-speed camera was also placed to record the event on film. To get a better view the camera and tripod were frequently mounted on five or ten feet of scaffolding to give a better angle on the blast area. Filming these events is often difficult because the blast area is obscured by topographic variation and tall vegetation. The scaffolding was a considerable help for improving the filming.

When the monitoring equipment was in place the blasting area was blocked off and the cap was attached to the first delay. The blasting lead wire had been stretched out to full length so that the end was next to the high-speed camera. Once the cap was attached and the blaster had returned to the shooting location a portable siren was sounded for one minute. At this time the twist type blasting machine was connected to the blasting line. After the siren rang the blaster counted down from ten. On the count of three the operator activated the high-speed camera to record the shot. On the count of zero the blast was initiated.

4.1.3 Post Blasting Tasks

After the shot the siren was sounded for the all clear. The shot was then inspected once dust and fumes had dissipated. Some time was allowed to elapse before inspection to allow for any delayed collapse. Care had to be taken when examining the result because sluffing and breaking away of material can proceed for some time after the shot.

Next the distance from the high-speed camera to the blast was measured with a one hundred foot tape. Also the azimuth was shot with a brunton compass and the vertical angle of the camera was taken.

The distance to the seismograph was usually measured before the shot. The information could then be entered into the seismograph. For both the seismograph and the camera the distances are accurate to less than five feet.

Once the measurements were taken the seismograph and high-speed camera were dismantled and stored. The blasting wire was reeled in and stored as well. All unused explosives and accessories were returned to the magazines which were located between test areas 2 and 3. The magazine inventory sheets were brought up-to-date to comply with federal law.

Later in the day, or the following day, the blasted sinkhole was further examined. By this time one was assured the area had stabilized. The profile was observed and photographs were taken. In some cases a portion of the area might have ratholed and still be open to the works, or just a deep depression was created as blasted material flowed away into the room. If necessary a second shot was planned to accomplish final filling of the area. This was needed after four shots. The problem usually occurred where there was a long sloping void that was open to the room beneath. When a second shot was needed it usually required significantly less holes than the original blast.

4.2 DESCRIPTION OF INDIVIDUAL BLASTS

4.2.1 Vertical Opening VO-1 (August 3, 1988)

This opening had occurred where blasting had been performed during previous research. The void that subsequently developed in the blasted area was at the intersection of room N-14 and entry NH. From the entry the room ran to the north. The entry was open to the west of the opening for about 15 feet where it was terminated by an old sinkhole. The room also showed voids for about 30 feet. From there it was collapsed by prior blasting. Figure 4-4 is a drawing of this sinkhole prepared from measurements taken during preliminary field work.

A pattern was designed to collapse the room to the north, the entry to the west and fill the sinkhole that had formed. Twelve holes were staked. Of these eleven were drilled. One hole was abandoned because the drill could not be safely set up on it.

The pattern was loaded as specified on the blast summary and loading chart. It was tied in as planned and detonated on August 3.

The results of this blast were good. The sinkhole was filled. Shallow depressions had developed on the room and the entry indicating their collapse. It was considered that the shot had successfully achieved the goals set out.

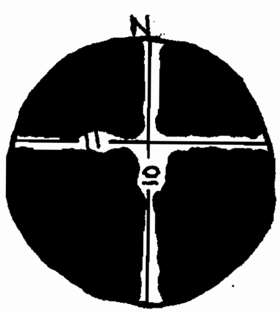
The blast layout and tie-ins for all the remaining blasts are included as an appendix. Figure 4-4 also includes a photograph of VO-1 after blasting.

4.2.2 Vertical Opening VO-2 (August 5, 1988)

This hole was located in test area 1. It had opened up after previous research in the area was finished in 1986. The hole was located on Room S-2. This room had not been previously blasted. Room S-1 directly west of S-2 had been shot in the previous work, however. There was no direct evidence that nearby blasting had precipitated the collapse of VO-2.

This vertical opening had dropped about 14 feet. There was no visible opening to the room. It was about 7 feet in diameter at the surface but considerably wider below. Figure 4-5 is a diagram of the sinkhole.

Seven production holes were staked around this opening. These varied in depth from 18 to 59 feet. Some holes were over the room and others were over the pillars. The longest production hole was over a portion of the room that had not



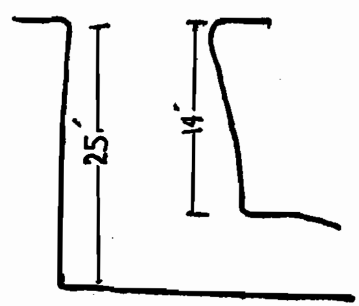
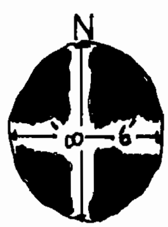


FIGURE 4-4: POST-BLASTING RESULT AND DIAGRAMS OF PREBLASTING DIMENSIONS OF VO-1



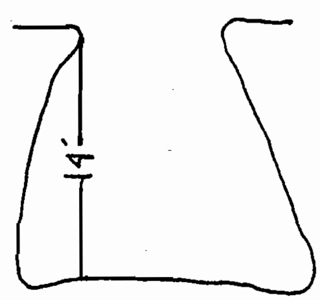


FIGURE 4-5: POST-BLASTING RESULT AND DIAGRAMS OF PREBLASTING DIMENSIONS OF VO-2

caved upward. Other holes were less deep and indicated that caving had progressed toward the surface in the vicinity of the sinkhole.

The sinkhole was blasted on August 5. The results were quite successful. The hole was well filled. There was no unwanted collapse along the room even though some holes were over the void. The surface disruption around the hole was modest. There was cracking in the soil around the opening several feet back from the area of actual disturbance. Figure 4-5 also has a view of the vertical opening after blasting.

4.2.3 Vertical Opening VO-3 (August 8, 1988)

This sinkhole was found in test area 2 as shown on the map in Chapter 3. It was circular with a narrow opening to the works trending to the south. The surface slump within the depression was shallow, perhaps about 4 feet in depth. Figure 4-6 is a diagram of the sinkhole.

A pattern of five holes was staked around the south half of the shot. Hole 4 struck a void at 9½ feet and was not loaded. Hole 4A was used instead. A sixth hole was drilled but not used when it was found that 4A would be usable. The holes were loaded using ANFO and PowerAN 7500.

The blast was detonated at 12:15 CDT on August 8. the results of this shot were very good. The hole was filled and the slope down to the workings was filled and sealed off. There was a small amount of surface swell that could easily be graded with a small dozer if required. Figure 4-6 also shows the post-blast result.

4.2.4 Vertical Opening VO-4A (August 8, 1988)

This opening was a long slump into a room with a substantial opening into the workings at the south end. The total slump was 40 feet long by 25 feet wide. The depression along the cave-in was 8 to 14 feet. Near the north end the depression was so shallow that it could not be blasted. Figure 4-7 is a diagram of this vertical opening.

A blast was designed with holes along each side of the sinkhole to close the slope into the room below and to fill in the 40 foot long caved area. No holes were drilled directly over the open room in order to avoid a major collapse of the workings with a large depression being formed as a result. One hole hit the void and was plugged and backfilled to 18½ feet.

The blast was shot at 12:58 CDT on August 8. The shot filled most of the area quite well. At the north end the slump was completely filled. Trending to the south the area was left as a depression.

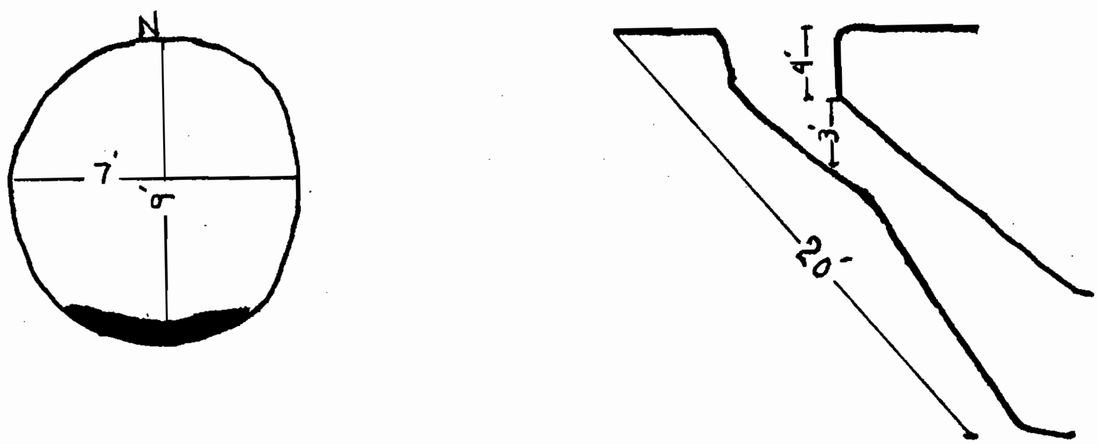


FIGURE 4-6: POST-BLASTING RESULT AND DIAGRAMS OF PREBLASTING DIMENSIONS OF VO-3

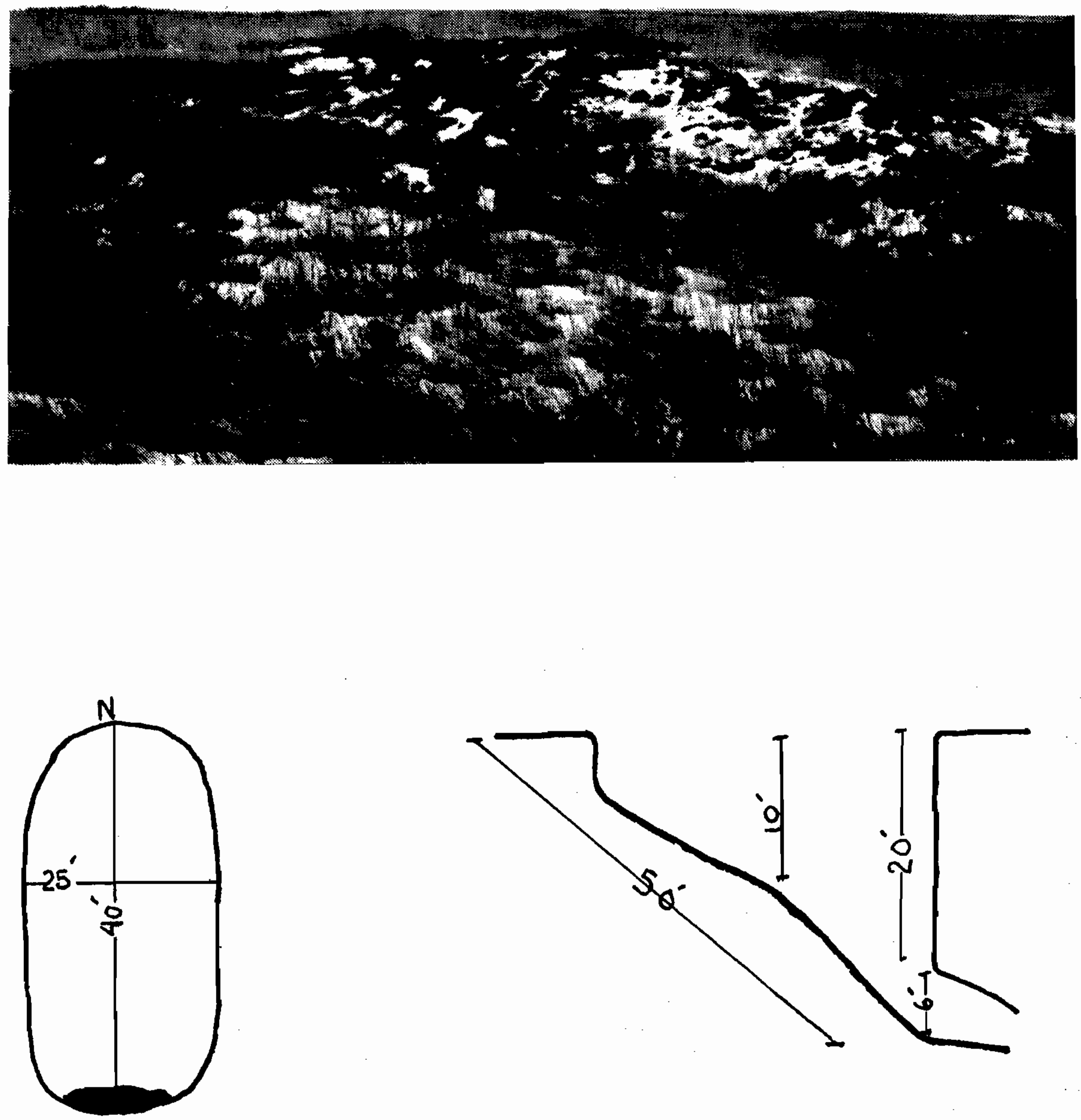


FIGURE 4-7: POST-BLASTING RESULT AND DIAGRAMS OF PREBLASTING DIMENSIONS OF VO-4A

This blast was not totally successful. At the south end, blasted overburden flowing away into the room resulted in a rathole about 15 feet deep. The hole was not open to the workings but left a poor result as far as reclaiming to the surrounding topography.

4.2.5 Vertical Opening VO-4A Shot Number 2 (August 17, 1988)

A second shot was performed on VO-4A to complete the filling of the sinkhole. Three holes were placed south of the vertical opening. These were 8 feet back from the hole and 10 feet apart. At the time of shooting the rathole was 12 feet deep. The holes were drilled 18 feet deep and loaded with two decks.

The blast was shot at 5:25 p.m. CDT on August 17. This blast was successful. It filled the remaining void very well. Although the holes were drilled over the workings beneath there was no undesired collapse. The final profile of VO-4A after two shots is shown in figure 4-7.

4.2.6 Vertical Opening VO-4B (August 17, 1988)

This opening was an oval shape. It consisted primarily of a steeply dipping void sloping north into the room beneath. The horizontal projection of the slope was about 25 feet. The slope distance was 60 feet. Figure 4-8 is a sketch of this sinkhole.

A total of eight holes were staked and drilled. The holes were placed along the north half of the oval and extended north along the slope on both the east and west side. The holes were placed for a distance of twenty feet along each side of the slope. The holes were drilled deeper as the depth of the void became greater. One hole was void at 18 feet. This blasthole was plugged and filled to 15 feet.

This shot was detonated at 5:01 p.m. CDT on August 17. The blast performed very well. The void was completely filled. There was some swelling of the ground on the west side. Topsoil remained near the surface. The blast was considered fully successful. Figure 4-8 shows a photo of the reclaimed feature.

4.2.7 Vertical Opening VO-5 (August 16, 1988)

This feature consisted of a hole with an open slope at both the north and south ends. The north ramp was the more pronounced and extended a significant distance beyond the sinkhole. The center area of the vertical opening was already filled with rubble. Blast design concentrated on the two ends. Figure 4-9 is a sketch of the sinkhole.

The blast pattern included five holes around the south end of the opening. Eight holes were drilled around the north end,

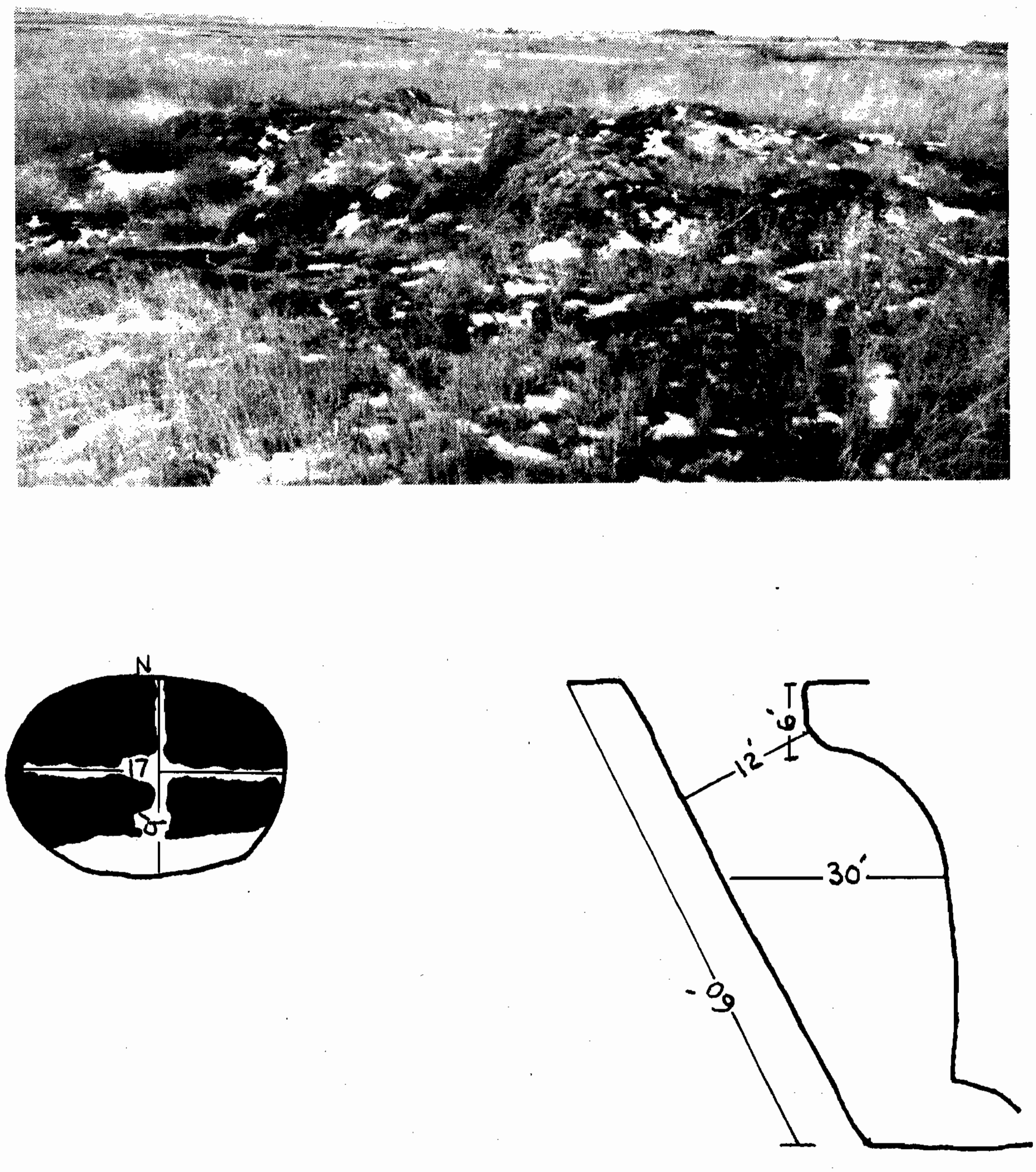


FIGURE 4-8: POST-BLASTING RESULT AND PREBLASTING DIAGRAMS OF VO-4B

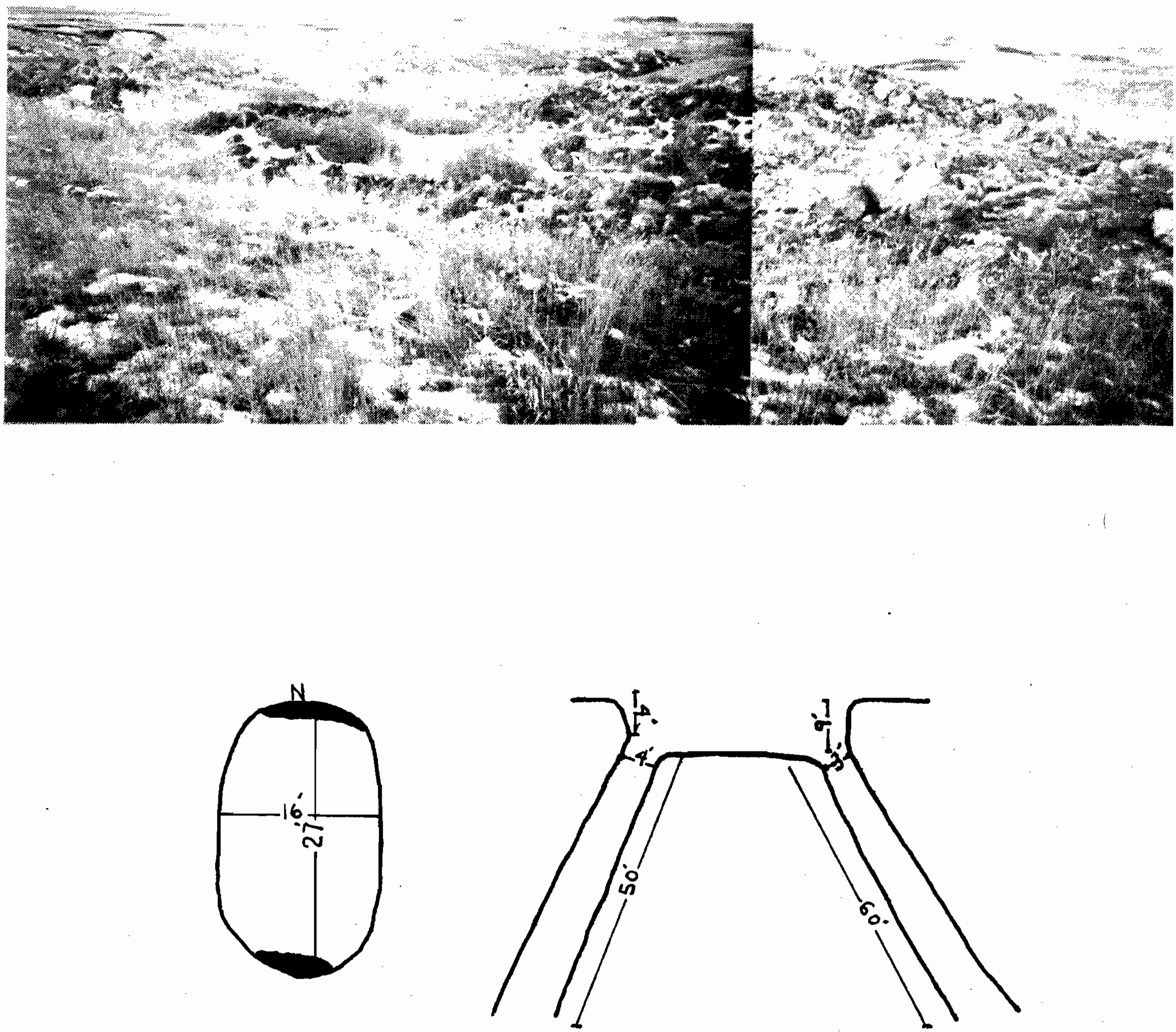


FIGURE 4-9: POST-BLASTING RESULT AND PREBLASTING DIAGRAMS OF VO-5

some of which extended north along the caved ramp. These holes intersected a void. These were plugged and backfilled as required. All thirteen holes were loaded using a combination of PowerAN 7500 and ANFO.

The blast was detonated on August 16 at 4:15 p.m. CDT. It was initiated from the south end with the north end firing last. All holes were delayed 42ms on surface; all decks were delayed by 50ms. The shot was successful with both ends being closed. There were some depressions but also some material which swelled. It would be a simple matter to rearrange the overburden to provide a level result. Topsoil remained near the top where it would assist revegetation. Figure 4-9 includes a photograph of the blasted sinkhole.

4.2.8 <u>Vertical Opening VO-6 (August 17, 1988)</u>

This vertical opening was oval in shape with a narrow opening to the workings at the south end. The center of the depression was shallow with the deepest part at the south end where the ramp occurred. One could not see all the way to the workings.

Six holes were drilled in a semicircular fashion around the south end of the sinkhole. These holes were 10 feet apart and carried 8 to 10 feet of burden at the crest. Drilling depths ranged from 14 to 20 feet. One hole struck a void at 20 feet was plugged and backfilled to 18 feet.

The blast was detonated on August 17. This shot filled the sinkhole fully. The slope at the south end was sealed. Because much of the opening was shallow, mounding of the blasted material occurred. The swell of the surface was approximately 5 feet. Figure 4-10 is a photograph of the blasted vertical opening.

4.2.9 <u>Vertical Openings VO-7N and VO-8N (August 19, 1988)</u>

Vertical openings VO-7 and VO-8 were long depressions resulting from major caving into the room below. Because of the size these were split into north and south sections. The north section of both sinkholes were shot together as were the south ends of both voids.

Both of these sinkholes had major open ramps to the workings beneath. The holes were side by side on adjacent rooms. The caved slopes extended 40 to 50 feet beyond the rim of the depression. The blast was designed to fill the northern portion of the long depressions and to close the open ramps at the end of each depression. Figure 4-11 is a sketch of VO-7. Also included is a photograph of the open ramp. Figure 4-12 is a sketch of VO-8 and includes a view of the open ramp at the north end.

FIGURE 4-10: POST-BLASTING RESULT OF VERTICAL OPENING VO-6

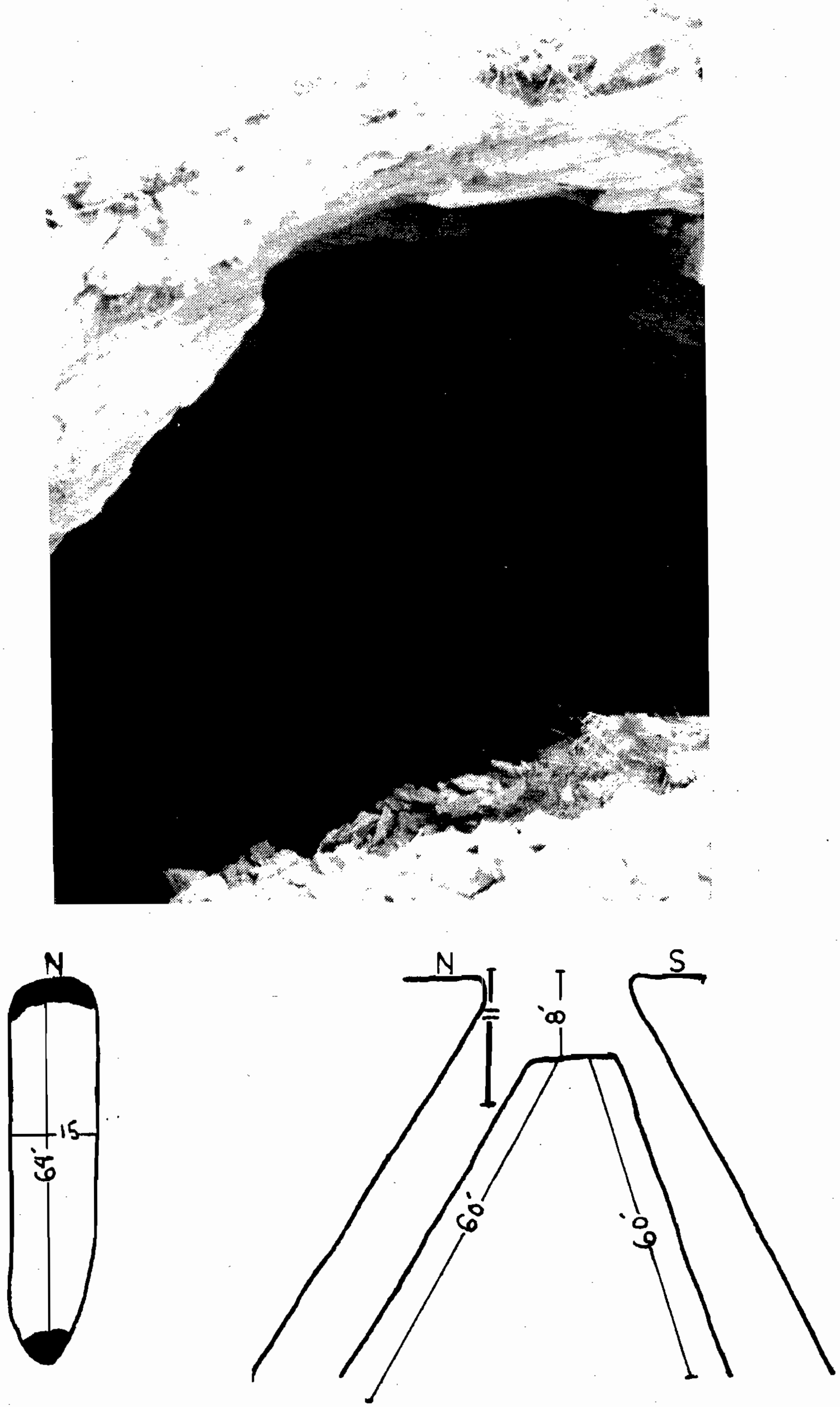
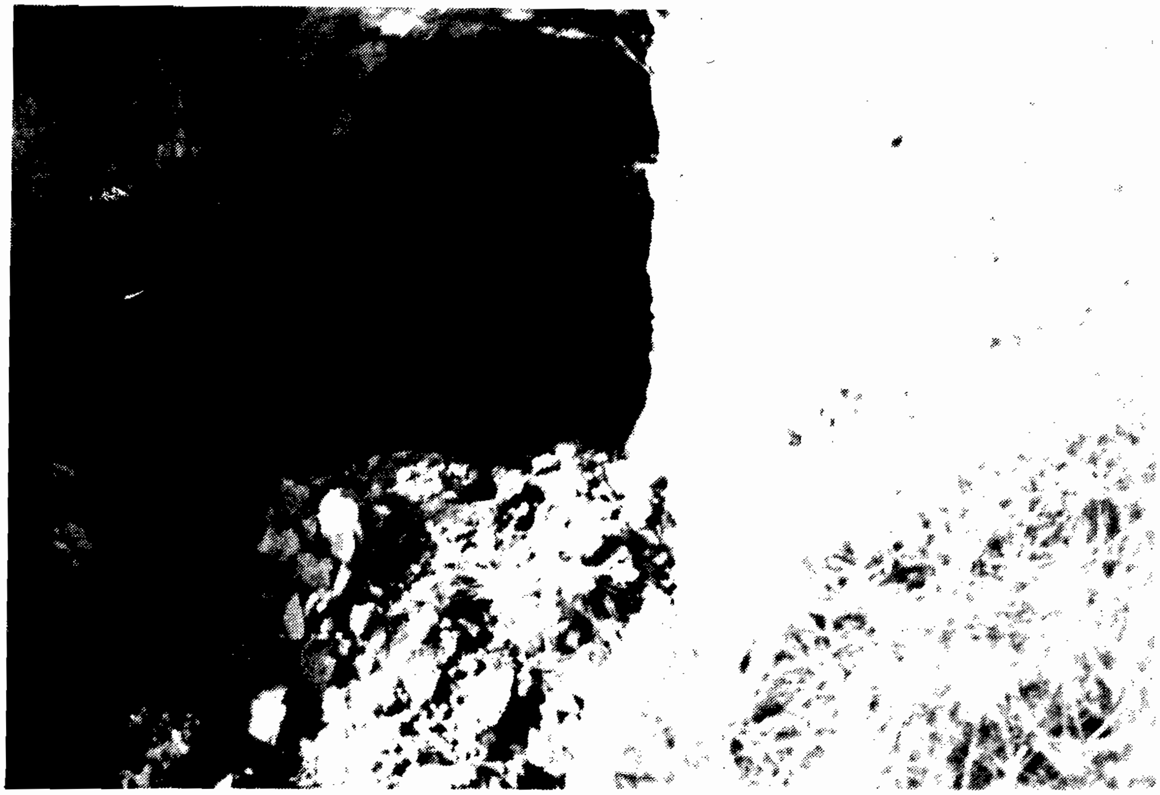


FIGURE 4-11: PREBLAST PHOTOGRAPH AND DIAGRAMS OF VO-7



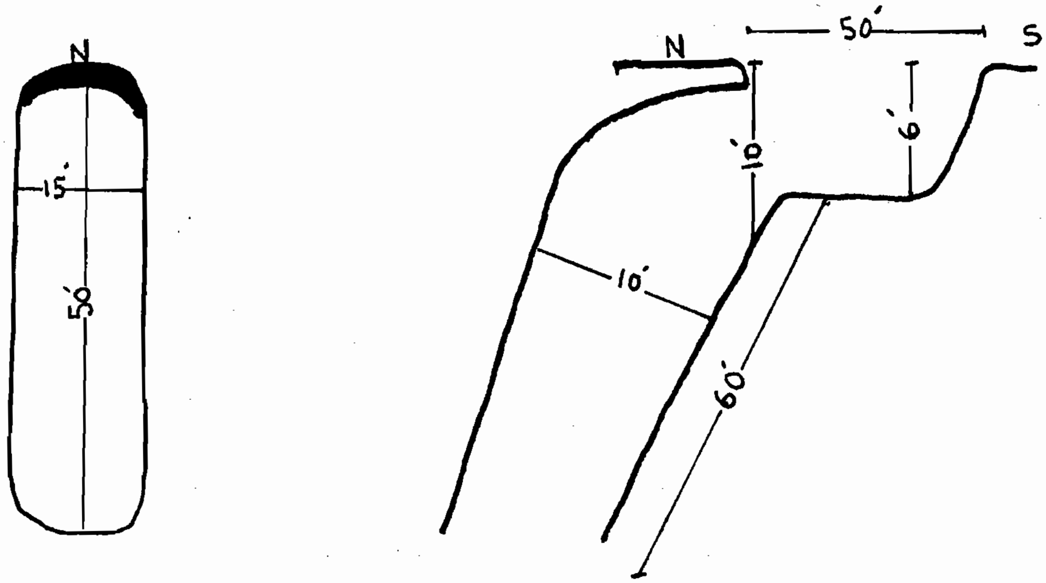


FIGURE 4-12: PREBLASTING PHOTOGRAPH OF AND DIAGRAM OF VO-8

The blast consisted of 29 holes. These were staked out and drilled along the sides of each sinkhole. Blastholes were drilled beyond the north rim of the openings in order to throw overburden into the sloping ramps thereby closing them. Two rows of holes were drilled between the sinkholes but to the north of the surface expression of these features. This was again to obtain material to cast into the ramps. It was not possible to continue these inside rows down along the depressions because there was not room to get the drill in and it was considered unsafe in any event.

Hole depths varied. the most shallow were along the sides of the depression. Deeper holes were drilled to the north as the ramps sloped to the workings below. Hole depths ranged from 15 to 35 feet deep.

A maximum of three explosive decks were used. The blast contained 2,200 pounds of ANFO and 660 pounds of PowerAN. Number 4, 6 and 8 delays were used down-the-hole and 42ms delays were used on surface.

Both the north and south ends of VO-7 and VO-8 were drilled off before any blasting. The reason was that there was concern about being able to get the drill on the shot after blasting and about the safety of doing so. Therefore, all blastholes were drilled around these sinkholes prior to shooting the north end.

The blast was shot on August 19, 1988. The open sloping tunnels at the north end were closed. The depressions along the rooms were closed for about two-thirds of the length. The north end of each room ratholed because of the large volume of voids but there was no opening to the workings remaining in either case. A second, small shot was expected to close these up.

As stated all of the sinkholes were drilled off prior to any blasting. After the north end blast one hole was lost on the west side next VO-7 and 2 were lost on the east side next VO-8. It was believed that one of the east side holes could be redrilled.

4.2.10 Vertical Opening VO-7 and VO-8 North; Shot Two (August 25, 1988)

The previous shot had left a rathole at the north end of each sinkhole. The hole at the end of VO-7N was 40 feet wide and 15 feet deep. At the north end of VO-8N the hole had dimensions of 26 feet wide by 14 feet deep. A second shot was planned to complete the filling of these holes.

A total of nine blastholes were staked out for the drillers. Each hole was to be drilled 15 feet deep. There was no point of drilling deeper because the ratholes did not exceed

FIGURE 4-14: VIEW OF OPEN VOID AT THE SOUTH END OF VERTICAL OPENING VO-7

FIGURE 4-15: FINAL RESULT AT THE SOUTH END OF VO-7 AFTER TWO BLASTS

the east side, were not drilled due to disturbance from the first blast. It was felt that the remaining four holes would adequately fill the small rathole.

The holes were drilled 20 feet deep. However, some sluffing of material into the blastholes resulted in 17 to 19 feet being loaded. All ANFO was used and period 4 and 5 delays were placed in the decks. As usual 42ms delays were used on surface.

The blast was shot on August 25, 1988. The result was successful. The hole was largely filled but a small depression did remain. There was no further subsidence in the vicinity of the room. Figure 4-15 shows the final result. Fragmented rock can be seen resulting from a lodge of rock occuring in this area.

4.2.13 Vertical Opening VO-9 (August 18, 1988)

This opening had a small hole at surface that was obscured by tall grass. Beneath surface it rapidly increased to a diameter of 30 feet. There was a large opening to the north. The hole was also open to the south. Figure 4-16 is a drawing of the sinkhole. The figure also shows a view of the void before it was blasted.

A blast was designed for this sinkhole that had eight holes. During drilling two further holes were added to try and gain more overburden to fill the large underground cavity. Because the void was drastically undercut it was necessary to watch carefully where the drill was placed. It was not possible to drill holes closer than 12 feet to the vertical opening. The holes were placed eight feet apart.

The blastholes were 19 to 25 feet deep. These were loaded with PowerAN (lower deck) and ANFO (upper deck). The delay periods used were numbers 4 and 6. The surface delays were NTD 42ms.

The blast was initiated on August 18, 1988. It filled in the area but was only partially successful. At the north end where the void joined the room, a depression 11 feet deep was formed. At the south end a depression was formed that was 22 feet deep and was open to the room striking south. the remainder of the void was closed but the configuration was one of a depression relative to the surrounding area. It was clear that a second blast was required for this vertical opening.

4.2.14 Vertical Opening VO-9; Shot Two (August 24, 1988)

A second blast was designed that arrayed blastholes around the north and south sides of the large circular depression left from the first shot. A total of six holes were drilled. Two were at the north end where a smaller, closed rathole had resulted and four were at the south end around the larger, open hole.

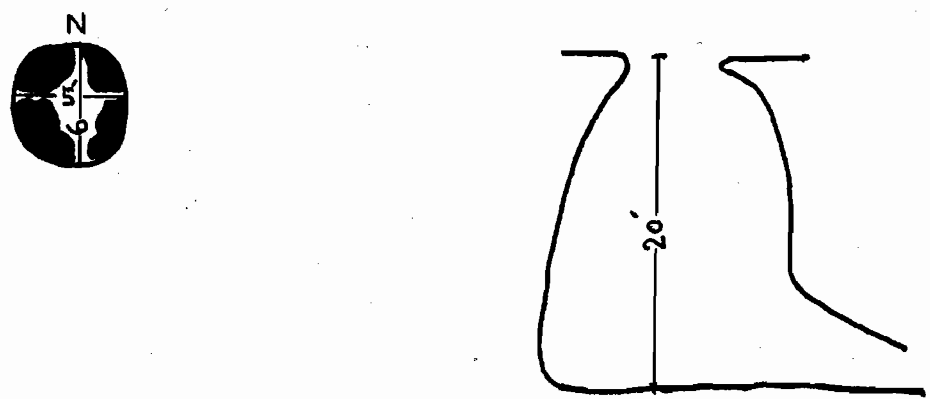


FIGURE 4-16: PREBLASTING PHOTOGRAPH AND DIAGRAMS OF VO-9

Most of the blast holes were 15 feet deep. Two, however. located at the south end were 20 feet deep. The holes were loaded with ANFO. Four holes had one explosive deck and two had two explosive decks.

The blast was shot on August 24. It was successful in closing the holes at both ends. The blasted profile overall still showed a depression, but this very dangerous vertical opening had been sealed. There remained a closed rathole of minor dimensions at both ends. This opening would require the most subsequent regrading of all those shot if a final result were desired that approximated the surrounding topography. Figure 4-17 is a photograph of the final result for VO-9.

4.2.15 Vertical Opening VO-10 (August 16, 1988)

This sinkhole was circular with a shallow depression and a narrow ramp striking to the north. The sloping ramp was perhaps 2 to 3 feet high. It was not possible to see all the way to the room. Exploration drilling indicated voids both north and south of the vertical opening. Figure 4-18 is an illustration of this sinkhole.

Five blastholes were laid out around the north end of the sinkhole. There was a 20-foot spacing between holes 3 and 4 to avoid blasting over the room beneath. Two holes intersected a void and had to be plugged. One of these voids was at 11 feet below surface and required reduced stemming at the collar to 7 feet 10 inches. The holes were from 11 to 20 feet deep. Given the shallow holes and single decks in all but one hole ANFO was used for all deck loads.

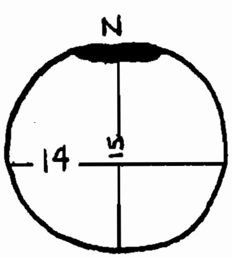
The shot was initiated at 11:15 a.m. on August 16 from the first hole on the west side of the shot. The blast was fully successful. The narrow slope was closed. There was a slight depression in the center, but also some swelled material around the perimeter which could be pushed into the center for final leveling if desired. Figure 4-18 also provides a photographic view of this sinkhole after blasting.

4.2.16 Vertical Opening VO-11 (August 30, 1988)

Sinkhole VO-11 was the first to be blasted in Beulah test area 3. It was a large oval opening with a steep ramp at the south end that was very open to the underlying workings. The depression was shallow at the north and rapidly dropping to a large cavity at the south end. Figure 4-19 is a drawing of the vertical opening and also shows a photograph of the sinkhole before blasting. Figure 4-20 is a view of the preblasted void showing the large opening to the room below before shooting.

The void to be filled was large. There was estimated to be in excess of 1,000 cubic feet of void. Based on this estimate plus the experience shooting VO-9 it was determined that two

FIGURE 4-17: FINAL RESULT OF VERTICAL OPENING VO-9 AFTER TWO BLASTS



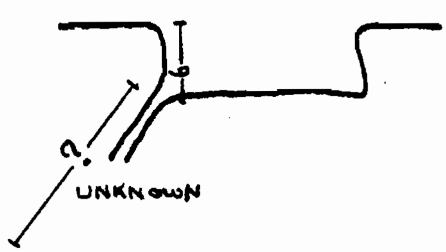


FIGURE 4-18: POST-BLASTING RESULT AND PREBLASTING DIAGRAMS OF VO-10

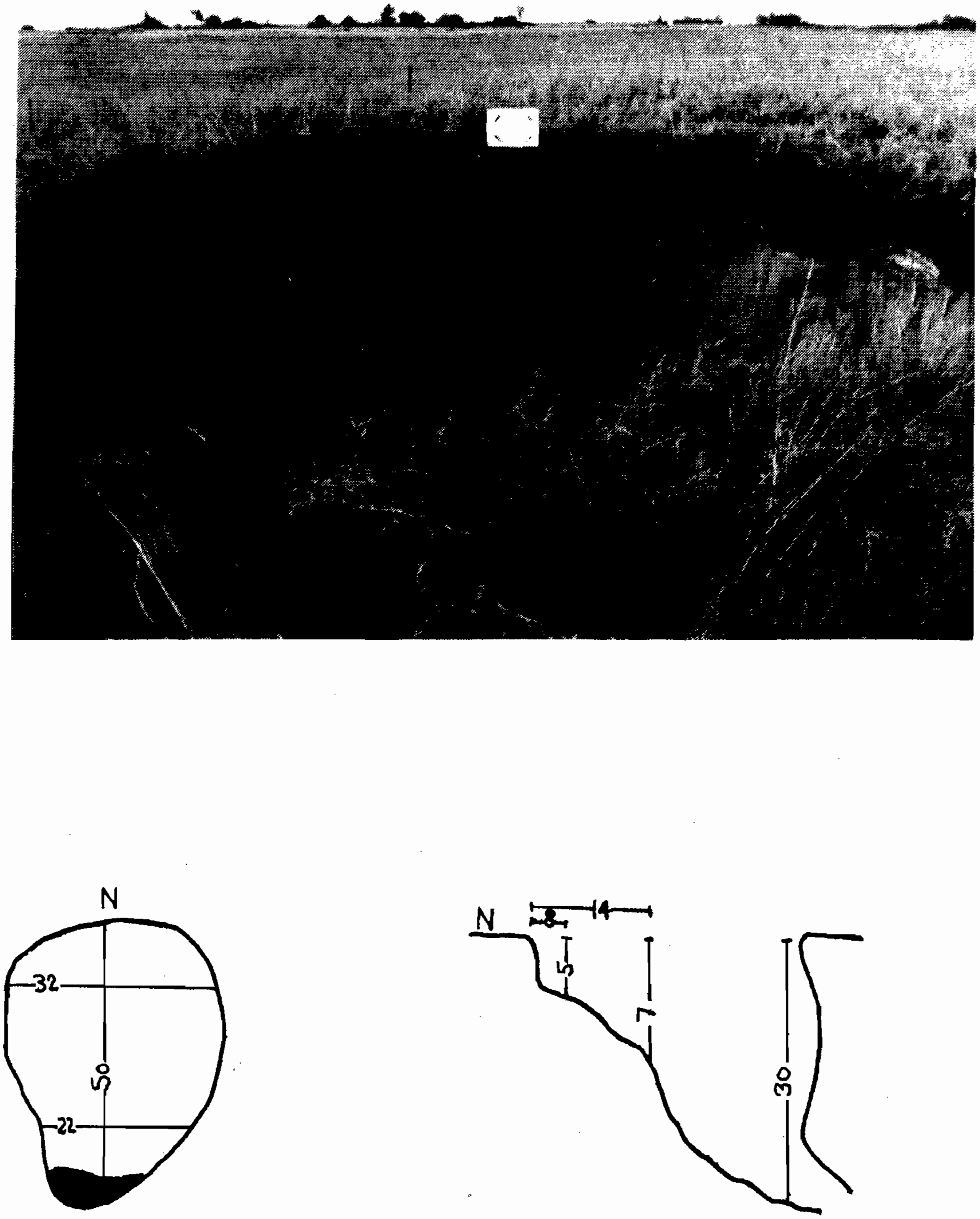


FIGURE 4-19: PREBLAST PHOTOGRAPH AND DIAGRAMS OF VO-11

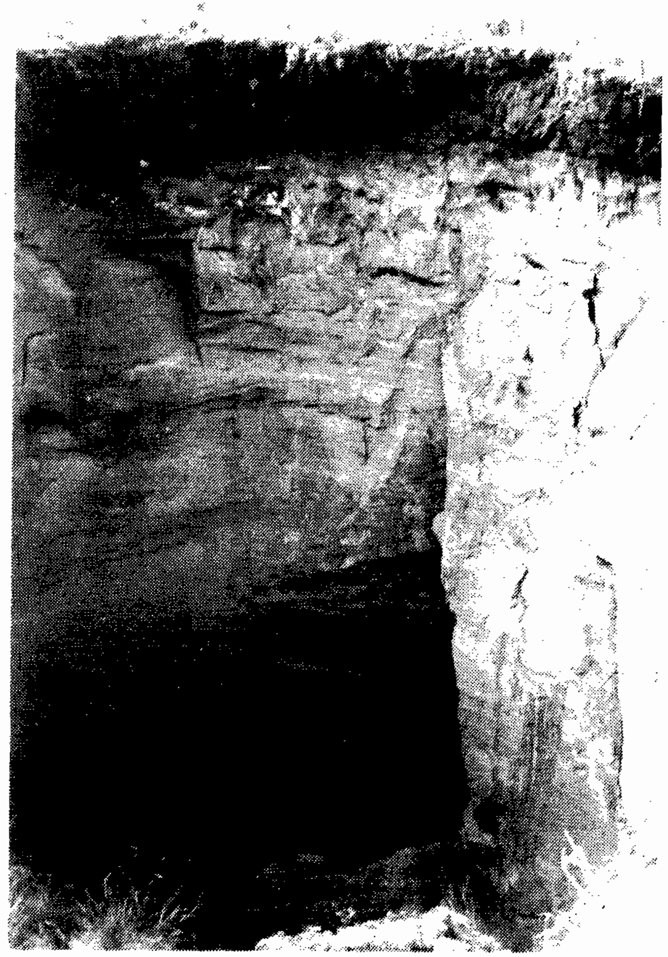


FIGURE 4-20: PHOTOGRAPH OF VO-11 BEFORE BLASTING SHOWING THE LARGE VOID TO THE SOUTH

FIGURE 4-21: POST-BLASTING PROFILE OF VERTICAL OPENING VO-11

rows of holes should be drilled around the south two-thirds of this void. The first row was drilled seven feet behind the rim of the vertical opening to give a good casting effect into the hole. The second row was drilled seven feet behind the first to further cast material toward the void. The estimated volume to be produced, at a 1.2 swell factor, was 24,800 cubic feet. This was believed to be adequate to fill the void, close off the open ramp and leave a final profile near that of the surrounding topography.

A blast of twenty-two holes was laid out. The spacing between holes was 10 feet on the front row. The spacing was 12 to 13 feet on the second row to account for the larger arc of the semicircle. The holes on the second row were staggered between the holes of the first row. The holes ranged in depth from 15 feet to 30 feet. The deepest holes were over the room. Three holes struck the room beneath and had to be plugged and backfilled. From one to three decks were used in each hole. Periods 4, 6 and 7 delays were used in the decks and 42ms delays were used on surface. When there were three decks PowerAN was used in the bottom deck.

The blast was detonated at 1:44 p.m. on August 30. The shot was successful. The open void was closed. The depression was filled. The after-blast profile was one in which the north end of the sinkhole was swelled which declined to about the normal topographic elevation to the south with a shallow depression at the south end. Figure 4-21 is a photograph of the after-blast profile.

4.2.17 Vertical Opening VO-12 (August 31, 1988)

This was also a large opening. It had a major void into the room below striking east. Exploration work also indicated a void trending west. It appeared that there might be an opening to the room in this direction also but there was not enough visibility to confirm this. However, production hole 14 at the northwest end of the shot hit a void at 8 feet below surface, further indicating the possibility of an opening back into the works to the west. Figure 4-22 is a drawing and photograph of the sinkhole. Figure 4-23 shows the large void trending east.

A total of fourteen holes were drilled around the vertical opening. The blastholes around the east end had to be placed well back from the rim for drill safety and to get a hole of sufficient depth to load. The distances ranged from 15 to 21 feet. Where possible the holes were placed 8 to 10 feet from the void.

The large void at the east end led to the placement of a second row of three holes behind the front row. These holes were 7 feet back from the first row and were spaced 12 feet apart. The hole spacings on the front row were 10 feet.

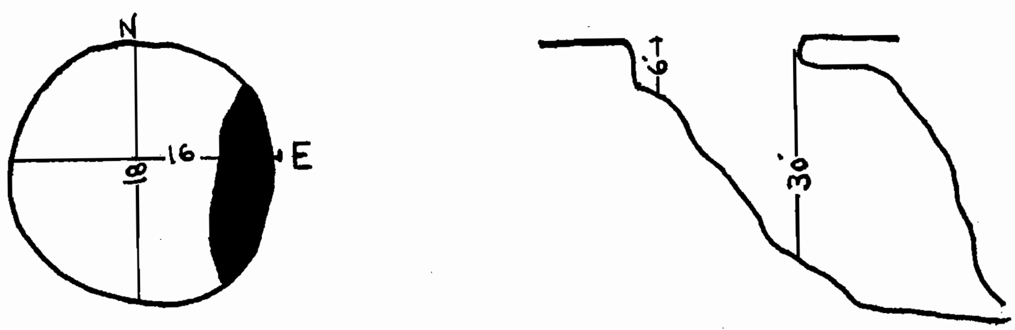


FIGURE 4-22: PREBLAST PHOTOGRAPH AND DIAGRAMS OF VO-12

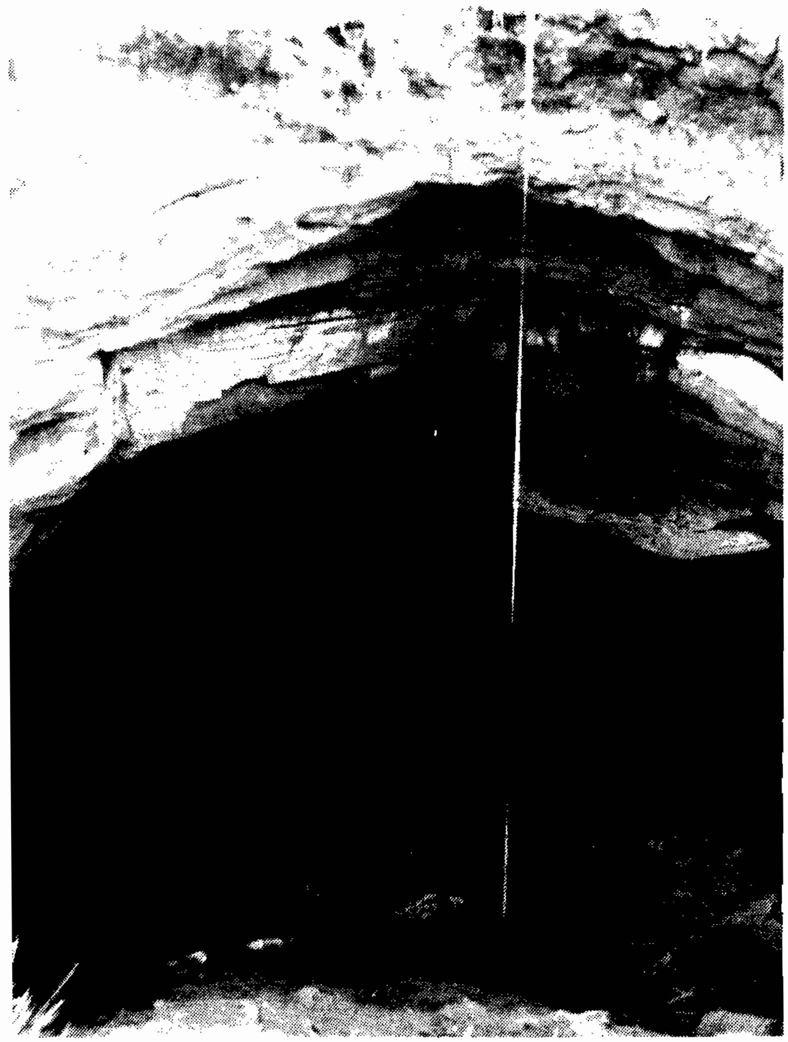


FIGURE 4-23: VIEW OF VO-12 SHOWING THE LARGE, SLOPING VOID STRIKING EAST

Four holes encountered the room below and were plugged. Loaded hole depths ranged from 15 to 25 feet. A maximum of three decks were loaded but this occurred in only two holes. Otherwise two decks were placed. Hole 14 was not loaded because a void was struck at 8 feet below surface. Delays were similar to that described in previous shots.

This blast was initiated from the northeast end on August 31. The blast was very successful. The filled vertical opening was at or near original ground elevation. Much topsoil and vegetation was left on or near the top of the blasted overburden. If further reclamation were done it would consist of minimal regrading and seeding. The blast results can be seen in figure 4-24.

4.2.18 Vertical Opening VO-13 (August 31, 1988)

This vertical opening was a long depression where a room had caved from the coal cropline toward the east. It had a large sloping ramp to the workings at the east end. The long depression was about 65 feet long, 20 feet wide and 10 feet deep. Figure 4-25 shows part of the depression and the open ramp. Figure 4-26 is a close-up view of the sloping void.

A total of 21 holes were drilled around this sinkhole. The blastholes extended 40 feet east of the rim of the opening. One hole was drilled directly over the room below also 40 feet east of the rim. The remaining holes were drilled down each side of the depression to fill this up to ground level. The holes ranged from 15 to 35 feet deep. As many as three explosive decks were used. When a hole had three decks PowerAN was used in the bottom. One hole hit the void and was plugged. Period 4, 6 and 7 delays were used down-the-hole and 42ms NTD delays were used on surface.

The blast was initiated from the center of the north row. A few deck overlaps occurred but the weight per deck was low so this was not a problem. The blast was initiated on August 31.

The results were quite good. The long depression was filled. The open ramp was sealed. There was a shallow rathole at the east end. However, there was swelled material directly west of this so that final reclamation would simply involve rearranging the blasted material.

FIGURE 4-24: POST-BLASTING RESULT FOR VERTICAL OPENING VO-12

FIGURE 4-25: VIEW OF VO-13 SHOWING A PORTION OF THE LONG DEPRESSION AND THE OPEN VOID AT THE EAST END

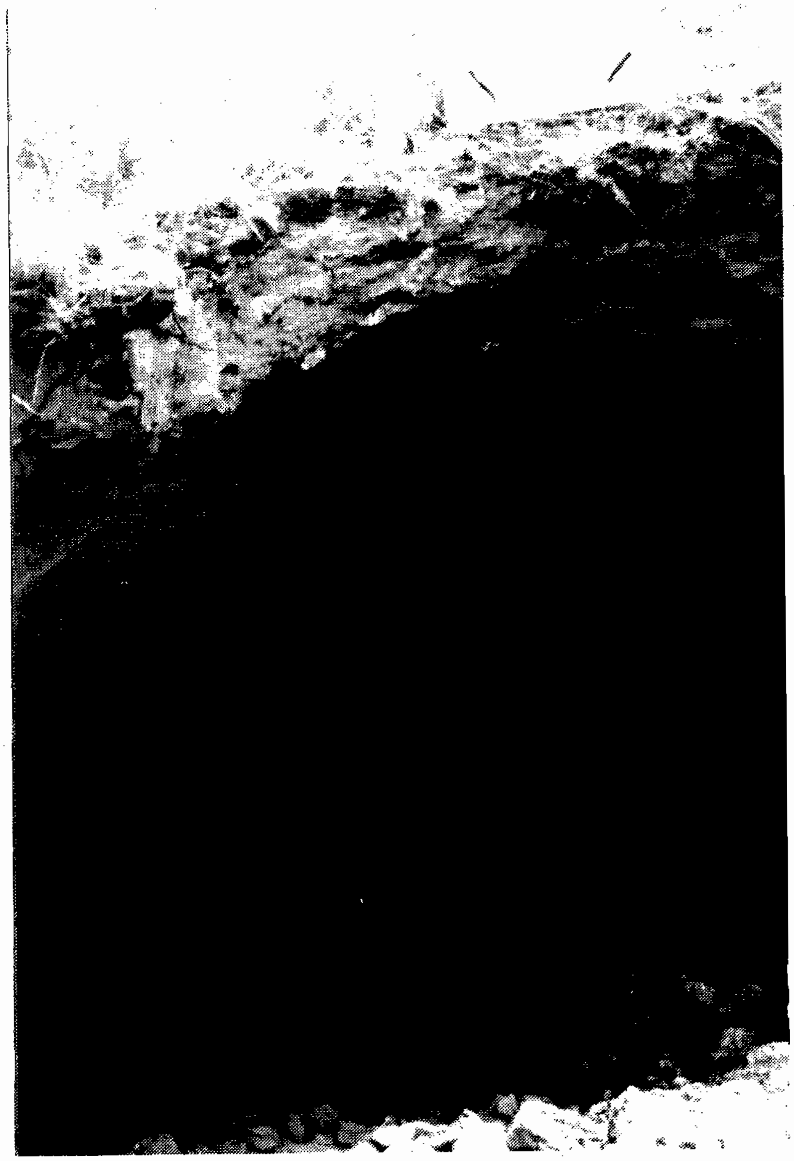


FIGURE 4-26: CLOSE UP VIEW OF THE SLOPING VOID AT THE EAST END OF VO-13

CHAPTER 5: FIELD PROCEDURES AND TECHNIQUES USED AT THE WHITE SITE

5.1 BLAST DESIGN AND IMPLEMENTATION

Blasting at the White Site involved the collapse of old underground coal mine workings. A principle difference with the Beulah Site was that at the White location the approach was to displace material down into the works whereas at Beulah one wished to cast material laterally into the central void.

The White test area afforded the opportunity to test several modifications to the general method. One modification was to vary the distribution of powder in the blastholes in an attempt to collapse the mine workings with the least amount of surface disturbance. A second change was to pattern blast rather than trying to pinpoint the rooms and blast these individually. The purpose was to determine if exploration costs might be reduced in this way and also to determine the technical feasibility of such an approach.

5.1.1 Preblast Design

A major preblasting requirement at this site was exploration and the development of an accurate mine map. The exploration work needed has been described in detail in Chapter 3. With the resulting underground mine map available individual blasts could be designed.

Several important factors had to be considered in the design of each blast. These included the number of decks of explosive and the location of these, the collar stemming heights, the spacing between holes and the layout of the blast holes.

5.1.1.1 Explosive Quantity and Location

In previous work explosive decks were placed to break all the material from the workings to the surface (4). The approach yielded good results. Initial blasts at the White test site were planned in the same manner. In this technique decks of explosive were placed in the hole to crater into the room successively. The top deck location was such that good surface displacement was developed and material throughout the column was able to move toward the opening well. There was little chance of a void being developed at a higher location in the overburden.

While this approach provides little chance for transferring the void to a higher location, it also results in quite significant surface disturbance. This means that if further reclamation, including regrading, is performed it will require more work and a greater cost will be incurred. Topsoil may need to be removed and later respread to achieve good growth.

It is better if the surface disturbance can be kept to a minimum. Ideal is the situation where there is no disturbance. To minimize the disruption on the surface the explosive charge at the top of the hole must be buried deeper. If the scaled depth of burial of the charge is high surface expression of the breakage will dramatically decrease.

The White testing area was an ideal site to study the ability to limit surface disturbance. The workings were generally small in dimension and often the void height was limited due to roof collapse. Therefore, even though the workings were shallow, swell in the blasted overburden could be expected to help fill the void. If all the overburden were broken surface swell could be expected. Consequently, it might be possible to break adequate material to fill the void while placing the charges far enough from surface to minimize disruption. Blasts using different explosive quantities and locations were designed and shot at the White test site.

5.1.1.2 Collar Stemming Heights

At the White Site considerable work was done to generate data concerning the amount of stemming to be used for the top deck of explosive. As stated the desire was to completely fill the voids but to limit surface disruption as much as possible. Initially, stemming heights were used that were the same as those at Beulah. The nine feet of collar stemming had given good results. The overburden was fully broken but there was no uncontrolled flyrock. Surface disruption was, however, considerable. For ANFO in a three-foot deck and six-inch hole diameter the scaled depth of burial of the top deck was 3.3 $ft/lb^{1/3}$.

This stemming height was used for the first two blasts. In the third blast some holes were loaded to a 12-foot collar. The ANFO load had a 3-foot rise and the scaled depth of burial was 4.3 $ft/lb^{1/3}$. In subsequent blasts other stemming heights were tried. For example, 16 feet was used yielding an SDOB of 5.6 $ft/lb^{1/3}$. Also 20 feet was used with a SDOB of 6.8 $ft/lb^{1/3}$.

Two individual test holes were also detonated. The first had 26.5 feet of stemming and a scaled depth of burial of 7.8 $ft/lb^{1/3}$. The second had 8 feet of stemming but only 20 pounds of ANFO giving a 3.3 $ft/lb^{1/3}$ SDOB.

Therefore, a good opportunity was afforded to study the stemming height versus the ability to fill the voids and also minimize surface disturbance. The results are discussed in a later chapter.

5.1.1.3 Spacing Between Holes

The design of each blast also required that spacing between the blastholes be determined. The spacings used were a combination of design principles and the need to fit the holes into the configuration of the mined room.

The rooms at the White test area were narrow. Therefore, when shooting an individual room a single row of holes was adequate to do the job. The rooms varied from 7 to 12 feet wide and for a three-foot deck of ANFO this gave a scaled radius of 1.2 to 1.9 $ft/lb^{1/3}$. This was well within the ability of the charge to break the overburden. Intersections between rooms and entries were often large. In these cases additional holes could be needed to ensure breakage.

Often a spacing between holes of 1.5 times the depth of burial works well in cratering. At this site the depth of burial was about 6.5 feet. Therefore, a spacing between holes of 10 feet appeared appropriate. This was the typically used value. Some field adjusting was necessary to fit the holes into the geometry of the irregular rooms.

Later in the field research pattern blasts were designed at the White Site. For these shots different burdens and spacing were used to determine what was best. Patterns of 10 x 10 feet, 12 x 12 feet and 15 x 15 feet were attempted. The results of each test blast were recorded.

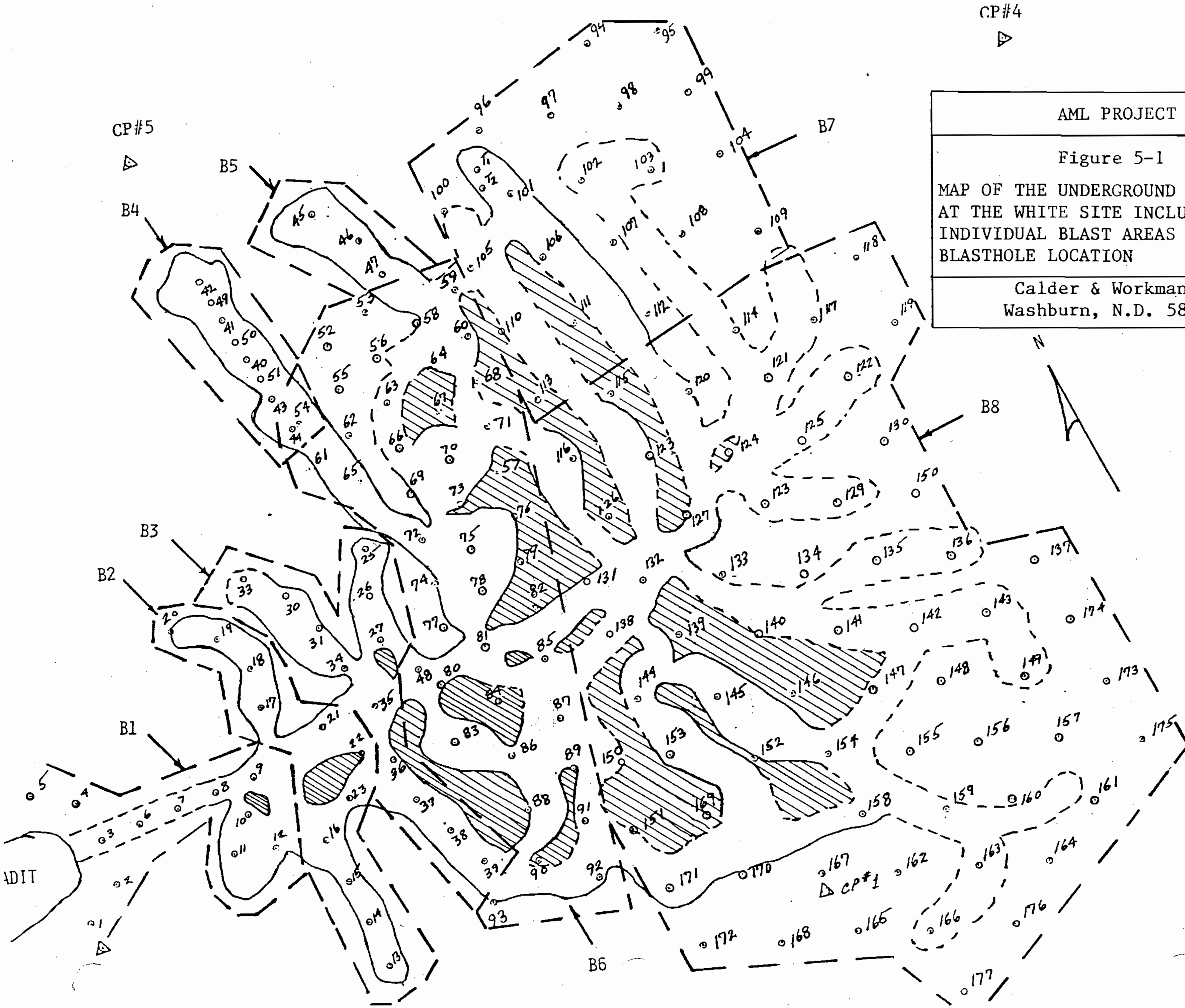
5.1.1.4 Blast Layout

Once the design requirements had been determined the blast could be laid out in the field. Planning the blast included using the map of the workings that had been developed to locate the holes. This map is shown in figure 5-1. The map also shows the location of each blast and the hole locations and numbers for the blastholes shot.

On this map B1 through B5 were blasts intended to collapse individual rooms. Blasts B6 through B8 were pattern blasts designed to collapse the entire area.

The map shows clearly the very irregular direction and dimension of these underground workings. This fact meant that holes could not always be laid out exactly as planned. However, when pattern blasts were employed, and one was not concerned about whether each hole was in a void, then the designed layout could be closely adhered to.

Another factor affecting blast layout was cracking around the previous shot. If such disturbance was extensive, then it could be difficult to drill some of the holes in the desired



AML PROJECT

Figure 5-1

MAP OF THE UNDERGROUND WORKINGS AT THE WHITE SITE INCLUDING THE INDIVIDUAL BLAST AREAS AND THE BLASTHOLE LOCATION

Calder & Workman Inc.
Washburn, N.D. 58577

locations. This was an important factor at the White Site because of the haphazard size and arrangement of the pillars. Surface disturbance could occur a significant distance behind the shot.

In the field the blastholes were staked out using 3-foot long lathe board. The hole numbers were marked on the stakes as was the hole depth. The drillers then drilled the holes at the planned locations. The drill crew kept logs of each hole as it was drilled. Figure 5-2 is a photograph of a blast area. The hole stakes can be seen. The blastholes have been drilled and the top of each lathe painted orange for greater high-speed camera visibility when the shot was detonated.

5.1.2 Blast Related Work

Once the blast had been designed and laid out it was drilled off with a 6-inch truck mounted rotary drill. In some cases the drill crew had difficulty collaring the hole because of backbreak from the previous shot. In order to successfully complete the hole, steps had to be taken to stabilize the collar. This was achieved by drilling about 10 feet of the hole and then installing 6-inch PVC pipe as a casing. The remainder of the hole was then drilled. These holes were loaded through the PVC casing.

The driller logged each hole as it was drilled including the total depth and noted the void if it was struck. Once the holes were drilled each one was taped by the blasting crew. The information concerning the depth of the hole was recorded on the blast summary sheet.

At this test site one typically drilled over the void. Therefore many holes struck the room below. The depth to the floor of the room was measured and recorded. The height of the void was then calculated.

Once this data was gathered decisions were made if and where a seismic plug should be placed. Also, the depth to be loaded was recorded on the sheet. Typically four to five feet of standoff were used from the plug to the bottom of the first explosive deck to provide an adequate depth of burial for the bottom charge.

Table 5-1 is an example of a blast summary chart at the White Site. This chart was used for blast B3. It is typical of the charts recorded for blasts at this site.

One will note that in the column indicating where to plug the holes there are numbers in brackets. These are cases where the bottom of the hole had caved between the time that the hole was first measured and the time the blast was to be loaded.

FIGURE 5-2: VIEW OF A BLAST AREA SHOWING THE STAKING PROCEDURES USED TO LAYOUT THE BLAST HOLES

AML PROJECT - BLAST SUMMARY

BLAST #: B-3

LOCATION: White #3

DATE 9-15-88

HOLE DIAM 6

NO. HOLES 13

EXPLOSIVES CONSUMPTION

ANFO 9 BAGS = 450 LBS SLURRY 15 BAGS = 450 LBS

BOOSTERS 24 42 M/S DELAYS 13

DELAYS #4 13 #5 ___ #6 11 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: North 460' PPV: 0.06 IN/SEC

CAMERA USED: Yes TARGET DELAY #: N/A FPS: 400 A2: 180' Decl. 1°

Focus: Infinity Distance: 450'

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
24	Not loaded because it hit void in newly discovered room									
25			NV	28	27		28			
26			29	28	27		32	3		
27			31	30	29		33	2		
28			Lost from Blast B-2							Too close to other holes
29			Lost from Blast B-2							
30		29	27	26	25		28½	1½		Rubble, loose materi
31		28	27	26(24)	23		28½	1½		" " "
32		27	24	23½(22)	21		27½	5½		" " "
33				30						
34		32		31(26)	25		32½	6		" " "
35		28	26	25	24		29½	3½		Plugged at 20. Plug would not go down
36			22	21	20		23	1		Lost circulation at 26'
37		15	14	13	12		19	5		
38		17	16½	15	14		20½	4		
39		21	21	20	19		25½	4½		

TABLE 5-1: EXAMPLE OF A BLAST SUMMARY SHEET USED AT THE WHITE TEST SITE

Because of the possibility of such caving each hole was measured again before loading. Typically, caving at the bottom of the hole occurs close to the hole only.

Once the seismic plug was placed the hole was backfilled, if necessary, to the correct depth for placing the bottom deck. The planned amount of the appropriate explosive was used in the bottom of the hole. Loading was performed according to the loading chart that was prepared prior to loading the blast, using information from the blast summary form.

After the bottom deck was loaded, including a one pound cast slider primer and the lowest period down the hole delay, stemming was placed according to plan. If more than one deck was loaded then the appropriate amount of deck stemming was placed, otherwise the hole was filled with stemming to the surface.

For multiple deck blastholes each deck contained a one-pound primer and an HD Nonel Primadet delay was used in every deck. The shortest delay was used in the bottom deck the longest in the top deck. Down hole delay periods were chosen to give a 50ms delay between decks within a hole.

Table 5-2 is a typical load chart for a blast at the White test area. This example is the chart used for blast B3. At this site the charts were made up by hole number rather than hole depth as was done at Beulah. Some adjustments to the loading chart had to be made in the field if the depths had changed as described above.

It is important to emphasize that using the blast summary sheet and the loading chart was essential to a successful result. These tables provided a mechanism for gathering the data needed for the planning of each blast, provided a means of written communication amongst members of the blasting team and provided adequate control of the blasting operation. Any group planning to use blasting to reclaim old works must use some form of planning and loading charts if an acceptable result is to be obtained. Loading these holes is made complicated by the multiple explosive decks required. It is important that each explosive deck be placed according to the plan for the best cratering results. Therefore, an easily followed method for loading the holes is necessary.

The bottom deck was loaded at an appropriate distance from the void. The purpose was to provide the correct depth of burial for the charge. Typically, the standoff was 5 to 6 feet. However, in cases of reduced powder loads the standoff was less. It is also true that the location of the void and the depth of hole had to be considered. If necessary, adjustments were made to the desired backfill amount. The decision was made when the loading charts were prepared.

TABLE 5-2: EXAMPLE OF A LOADING CHART USED WHEN LOADING BLASTHOLES AT THE WHITE SITE

BLAST B3 (September 15, 1988)

Hole #25	Hole #26	Hole #27	Hole #30	Hole #31	Hole #32
0	0	0	0	0	0
Stem	Stem	Stem	Stem	Stem	Stem
12	12	12	9	9	9
ANFO	ANFO	ANFO	ANFO	ANFO	ANFO
15	15	16	12	12	12
Stem	Stem	Stem	Stem	Stem	Stem
19	20	20	18	17	18
PowerAN	PowerAN	PowerAN	PowerAN	PowerAN	PowerAN
22	23	24	21	20	21
Stem	Stem	Stem	Stem	Stem	Stem
28	28	30	26	24	22

Hole #33	Hole #34	Hole #35	Hole #36	Hole #37	Hole #38	Hole #39
0	0	0	0	0	0	0
Stem	Stem	Stem	Stem	Stem	Stem	Stem
12	9	9	9	9	9	9
ANFO	ANFO	ANFO	ANFO	ANFO	ANFO	ANFO
16	12	12	12	12	13	12
Stem	Stem	Stem	Stem	Stem	Stem	Stem
20	18	15	15	13	15	15
PowerAN	PowerAN	PowerAN	PowerAN			PowerAN
24	21	19	17			17
Stem	Stem	Stem	Stem			Stem
30	26	20	21			20

Cast Primers:	24	ANFO:	9 bags	450 lbs.
Delays #4:	13	PowerAN:	15 bags	450 lbs.
#6:	11			
42 ms NTD:	13			

Once the bottom deck was loaded stemming was placed above the explosive. Drill cuttings were used for this purpose. If the hole contained multiple decks the stemming quantity used was as planned on the loading chart. The quantities used were typically from 4 to 6 feet. The stemming amount was designed to be as close to the depth of burial needed for an optimum result as possible while also accounting for the hole geometry.

Successive decks of explosive were placed as required. Once all the decks had been loaded the remainder of the blasthole was filled with stemming. As discussed above stemming heights varied in order to test various configurations and the effect on surface displacement.

During the loading of each hole a blasting tape was used to insure the correct rise of explosive and deck stemming. When PowerAN was loaded the hole was taped after each bag was added. When ANFO was poured into the hole or deck stemming was being added the hole was bobbed with the blasting tape to insure the correct rise.

The constant checking of the rise in the hole is very important. A key requirement for success is having the explosive decks correctly located. The only way to insure loading accuracy is to bob the decks accordingly. Figure 5-4 illustrates the loading and taping of a blasthole.

As stated each explosive deck had a down the hole delay. the lowest delay was in the bottom deck. A period 4 HD Primadet was used (100ms). Subsequent decks were delayed with higher periods which were usually period 6 (150ms) and period 8 (200ms) Primadets. Therefore, there was 50 milliseconds of delay between each deck in a hole. The bottom deck would break into the works, followed successively by the detonation and downward movement of overburden from higher elevations. Figure 5-5 is an illustration of a loaded hole at the White Site.

Once the holes were loaded and stemmed 42 millisecond noiseless trunkline delays were connected to the primacord downline of each hole. The combination of down hole and surface delays meant that each deck detonated separately and no closer than 8 milliseconds apart. The sequence of detonation was on a diagonal from bottom to top as described in Chapter 4. Therefore, additional freedom for displacement was obtained.

For the pattern shots, B6 through B8, the timing was somewhat different. In these cases multiple holes were shot per surface delay. The reasons were that individual delay between holes for relief did not appear to be as critical. Also, the 42ms delays inventory was depleting and reorder was a four to five week delay. Further, shooting multiple holes on a delay provided an opportunity to study the additive effects on vibration of multiple holes shot per millisecond delay. It was believed that shooting several holes simultaneously might assist the collapse.

FIGURE 5-4: THE LOADING OF ANFO INTO A BLASTHOLE WITH THE BLASTING TAPE BEING USED TO MONITOR THE COLUMN RISE

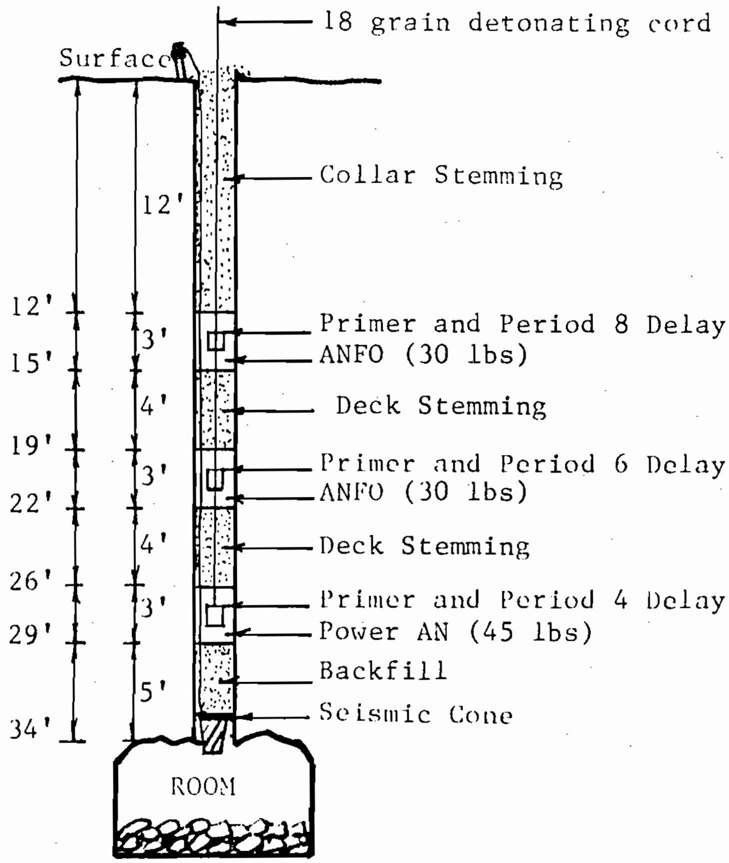


FIGURE 5-5: TYPICAL HOLE LOADING ARRANGEMENT FOR BLASTHOLES AT THE WHITE TEST SITE

Scale: 1" = 10'

While the surface tie-in was being completed the seismograph was set-up and programmed to record the blast vibration from the shot. Also, the high-speed camera was mounted in an appropriate location, usually to the north of the shot. Scaffolding was used for a better view if needed.

When everything was in readiness an electric blasting cap was attached to the lead-in delay. All surface delays, caps and primacord pigtails were buried under cuttings to reduce noise. The cap was connected to the twin lead-in line which was shunted at the other end. Access to the area was blocked off.

The blaster returned to the blasting location and the blasting siren was blown for one minute. During this time the circuit was checked with the blasting galvanometer. Once the siren had been blown the blasting machine was connected to the lead-in line. The blaster counted down from 10. The high-speed camera was activated at the count of 3 and the blast was detonated on the count of zero. After the blast the all-clear signal was blown. Figure 5-6 is an example of a blast detonated at the white Test Site.

5.1.3 After-Blasting Tasks

Once the blast had been detonated and fumes had cleared from the site the blasted area was inspected. Initial determination was made of the success of the shot relative to the goals set. Primarily this involved assessment of collapse, surface disturbance and backbreak.

The profile of the shot was observed and measured if possible. Usually, surface swell was involved because of the small voids beneath. However, when long stemming heights were employed surface disturbance was often minimal.

The amount of surface disturbance outside the shot area was measured and recorded on a mine map. This was important since less disturbance meant it was easier to drill and load the next shot. Designs were developed to reduce the disruption outside the blast area and measurement of the subsequent backbreak was an important procedure for determining success of the design.

Where possible the T.V. camera was used to look back at the area and assess the collapse of the blasted rooms. The G.E. camera from the State of North Dakota was also used to assess collapse. This instrument was able to operate in partially water-filled workings. The camera and operator were supplied by the Public Service Commission of the State of North Dakota and the cost was underwritten by the State of Montana. Drilling was also used, in some cases, to assess collapse.

FIGURE 5-6: DETONATION OF THE BLAST B8 AT THE WHITE TEST SITE

The direction and distance of the high-speed camera to the workings was measured. The vertical angle of the camera was determined with a Brunton compass. The direction and distance of the seismograph to the shot was also measured. The seismograph readings were checked as well.

5.2 DESCRIPTION OF THE BLASTS DETONATED AT THE WHITE SITE

5.2.1 General Discussion

The White Site was approximately one acre in size. Eight blasts were shot at this testing area comprising 177 production holes. In addition two test holes were shot to test particular loading possibilities.

The first two blasts, B1 and B2 were shot in a similar manner to previous work. Some difficulties arose, especially with surface disturbance beyond the blast area. Blast B2 had to be redesigned after B1 was shot for this reason. Also, there was the desire to determine the minimum possible extent of surface disturbance in order to reduce or eliminate further reclamation needs. These blasts were designed to cave in the room below.

Blast B3 incorporated some changes. This included eliminating the top deck in shorter holes, increasing stemming on the top deck in longer holes and decreasing powder quantity in the bottom deck. The goal was to reduce the surface disturbance that made drilling the next shot difficult.

Blast B4 continued the research begun in the previous blast. In this case only one deck, loaded with PowerAN, was used. Collars were 16 to 19 feet below surface. The explosive load was reduced to 45 pounds in each hole.

Blast B5 incorporated information gained from the previous shots. Two decks of explosives were employed. Explosive collars were 16 to 17 feet below surface. The purpose of the shot was to approach an optimum condition for void closure, surface disruption and ability to drill the next shot when blasting in individual rooms.

The last three blasts, B6 through B8 were pattern shots. Blast B6 was the first experiment with this approach. In this case one-half the blast was a 10 by 10 foot pattern while the remaining half was a 12 by 12 foot pattern. Blast B7 extended the research by expanding the pattern to 15 by 15 feet. The final blast, B8, was a large pattern shot that incorporated pattern and loading information from the previous two blasts.

The blast performance was studied by examining surface swell and measuring the amount of backbreak around the blast. Also several camera holes were used to observe the underground results of the blast. This was very useful in assessing the success of the cave.

5.2.2 <u>Blast B1 (September 12, 1988)</u>

This blast started at the adit and proceeded east along the main entry. It also included a widened out area just south of the entry and mined into the west side of the first room south of the main entry.

The adit was driven into the side of the hill along the west side of the property. It was mostly filled with eroded material but had a small opening above the rubble. This can be seen in figure 5-7. Five holes were drilled around the adit; the central hole being in the entry.

Four additional holes were drilled along the entry-way all of which struck the void below. Three blastholes were drilled in the wide area south of the entry.

The holes varied from 17 to 23 feet in depth. Eight holes were drilled to the void and were plugged using seismic cones at the appropriate depth. The voids below were shallow in depth and ranged from a minimum of 2½ feet to a maximum of 6½ feet. The holes around the adit were spaced 10 feet apart. The holes along the entry were 9 feet apart. The three blastholes in the widened area were placed in a triangular layout and all were spaced 9 feet from the others.

The blastholes were loaded with two decks of explosive, which was reduced to one deck for shallow holes. PowerAN was used in lower decks and ANFO was used in the remainder. Period 4 down-the-hole delays were used in all holes. When a second deck was employed period 6 delays were used. A 42 millisecond delay was used between each blasthole on surface. The blast was initiated from the south side of the adit.

Blast B1 was initiated on September 12, 1988. The shot successfully collapsed the voids. Also the adit was closed with the hole into the workings gone. This can be seen in figure 5-8. Along the entry there was considerable disturbance . The broadened out area collapsed leaving a depression with heaped material around the perimeter. There were deep cracks in the surface surrounding the blast area. Cracking and surface disruption east of the shot caused changes to be made to blast B2. In particular holes could not be drilled as close to the perimeter of B1 as originally planned. Blastholes for B2 had to be placed as much as 20 feet behind the last holes of B1. Therefore, the blast had done the job but created considerable surface disturbance and problems for the succeeding shot.

FIGURE 5-7: ADIT PROVIDING ENTRY TO THE UNDERGROUND WORKINGS SHOWING THE SMALL OPENING NEAR THE TOP OF THE ADIT

FIGURE 5-8: VIEW OF THE ADIT AFTER BLASTING

FIGURE 5-9: VIEW OF THE AREA BLASTED IN SHOT B1
LOOKING WEST TOWARD THE ADIT

Figure 5-9 is a photograph taken from east of the shot looking back toward the adit.

5.2.3 Blast B2 (September 13, 1988)

This blast was designed to collapse the rooms on both the north and south side of the entry and the large intersection between the rooms and the entry. Four holes were located along each room and three were in the intersection.

The blastholes all struck a void except number 20 which can be seen from the map to be right at the north end of the room. Hole depths ranged from 22½ to 26 feet. The height of the voids ranged from 3 to 5 feet. The holes were spaced 10 feet apart.

The backbreak from shot B1 resulted in changes to the original design. Some holes had to be eliminated. Holes 21, 22 and 23 were moved further east. Holes 16 through 23 were drilled in severely disturbed overburden from B1.

The disruption from the previous shot made drilling the blastholes difficult. The driller had difficulty collaring the holes in broken material. To solve the problem 6-inch PVC pipe was used to case the upper portion of the holes. Then drilling could proceed successfully. Explosive loading was also done through the plastic pipe.

The designed explosive loading was used. Two explosive decks were placed in each hole. PowerAN was used in the bottom deck and ANFO was loaded in the upper deck. Nine feet of stemming was used in each hole. The bottom decks were delayed with a period 4 delay; the top deck used a period 6 Primadet. On surface 42ms delays were used between holes.

The shot was initiated from the south end on September 13, 1988. The results were similar to B1. Large areas of surface swell occurred and there was considerable surface cracking in evidence. This blast generated surface disturbance 30 feet east of the centerline of the shot. This meant there was backbreak 12 to 14 feet beyond the blast area. The T.V. camera was used to examine the underground results. It was observed that all areas had collapsed except in the vicinity of the entry. This was the area where the original holes were moved. Here the roof was intact but had major cracks at least one-half way across the tunnel. This area collapsed with the subsequent blast.

5.2.4 Blast B3 (September 15, 1988)

Blast B3 was designed to collapse two rooms in a Y shape to the north, the central entry and a portion of a long room running south and then east. This shot, like the previous two, was designed to collapse the specific rooms below. Based on the results of the first two blasts changes were made to the design for B3.

The blastholes were drilled on a nominal 10-foot spacing as before. However, because of the surface disruption from the prior blast some hole spacings were greater. There was a 12-foot spacing between holes 33 and 30 and the spacing between 35 and 36 was 13 feet. Also, the spacing from 35 to 27 was 14 feet due to the small pillar between them.

The holes were 14 to 30 feet deep. The voids ranged from $1\frac{1}{2}$ to 6 feet in height. Holes 30 to 36 had to be moved 3 to 4 feet east because of the backbreak from the previous blast. Holes 28 and 29 could not be drilled for the same reason. For the holes where the ground was fractured from the previous shot the PVC pipe had to be used for successful drilling. The explosives were subsequently loaded through the pipe as well. A total of 13 holes were drilled.

Changes were made to the explosives loading for this blast. The explosive load in the bottom deck load was reduced by one foot relative to the previous shots. For holes in the northeast limb of the Y the explosive collar was increased to 12 feet to determine if this would decrease the disturbance. Short holes were loaded with only one deck.

Where two decks were used PowerAN was loaded. A period 4 down hole delay was used in the lower deck. The upper deck was loaded with ANFO and a period 6 delay was used. When only one deck was loaded ANFO was the explosive and a period 4 delay was used.

The blast was connected on surface with 42 millisecond delays. It was detonated on September 15, 1988. The result of the blast was that the entire area collapsed very well. The surface disturbance was significantly reduced. The northwest room of the Y was swelled more than the other areas. This was the section where 9-foot collars were used. A slight slump was noted on the south room. Where a 12-foot collar had been used there was little surface disturbance.

At the intersection of the three rooms an area 50 feet in diameter swelled up about 5 feet in height. The swell appeared due to shallow voids and a small pillar in the entry. Overall this blast succeeded in reducing surface disturbance. The disruption around the shot was reduced to a maximum of 10 feet from 20 feet or more seen on previous shots. The collapse of the voids was subsequently further confirmed using the T.V. camera.

5.2.5 Blast B4 (September 16, 1988)

The blast designated B4 was a smaller shot located along the first long room striking north. It was a shot designed to collapse the portion of this room in which holes were drilled.

Continuing the studies begun with blast B3 changes were made to the explosive loading. For B4 only one deck of explosive was loaded. These decks consisted of a three foot length of explosive. Three holes contained 19 feet of stemming. Two shorter holes contained 16 feet of stemming.

Five blastholes were drilled ten feet apart for this shot. The holes ranged from 24 to 27 feet deep. All holes struck the void. The void depths ranged from 1 to 3 feet high. Each hole was plugged 1 to 2 feet above the void. Backfill was then placed so that the bottom of the explosive was five feet above the room.

The explosive deck was three feet in height. Each hole was loaded with one deck of PowerAN. In three holes that were 27 feet deep 19 feet of stemming was used. The remaining two holes were 24 feet deep and 16 feet of stemming was put in each of these blastholes.

A period 4 (100 ms) delay was placed in each blasthole, in the one pound cast primer. A 42 millisecond Noiseless Trunkline Delay was connected to the primacord pigtail at the collar of each hole. The blast was initiated using one electric blasting cap.

The shot was detonated on September 16, 1988. The blast produced almost no visible surface disturbance. However, the voids did not entirely collapse. Instead, at one hole, the void had moved upward by 6 feet and reduced to a height of one foot. Therefore, the problem of surface disturbance had been successfully dealt with but at the expense of not totally collapsing the void. Since it is essential that the void be closed it was concluded that, unless the holes are quite shallow, two explosive decks should be used while using more stemming than was employed for the first two shots.

The area was subsequently reblasted to insure full collapse of the voids. Holes 49, 50, 51 and 54, as shown on the map, were employed for this purpose. The follow-up blasting was successful in fully collapsing the area. It showed that a blasted area could be redrilled and further collapsed, if necessary.

5.2.6 <u>Blast B5 (September 17, 1988)</u>

This was a small blast designed to cave the north end of the second long room to the north. The blast design was based on the information gained from the previous shots.

Three holes were drilled ten feet apart. These blastholes intersected the room beneath at depths from 32½ to 33½ feet. The voids below were 2½ to 3 feet high. Each hole had to be plugged about one foot above the hole bottom. Stemming was added above the plug so that the bottom charge was four feet above the opening.

Based on the results of B4 the design for B5 incorporated two decks of explosive. The bottom deck was PowerAN and a 4-foot powder rise was planned. Five feet of stemming was then placed followed by an upper 3-foot deck of ANFO. Two blastholes were then filled with 16 feet of stemming; the third had 17 feet.

Each bottom explosive deck was primed with a one pound primer containing a period 4 Primadet Each upper deck also contained a one pound slider primer with a period 6 down-the-hole delay. The surface tie-in was made with 42 millisecond NTD delays.

The blast was initiated from the North end on September 17, 1988. The results were very good. The voids were completely collapsed. There was only minor surface disturbance consisting of a one foot high hump along the centerline of the holes. Cracking of the surface was confined to hairline cracks extending five feet from the production holes. This blast design provided the type of result which was considered ideal when caving workings of this type. Figure 5-10 shows the results of this shot.

5.2.7 Blast B6 (September 19, 1988)

This blast was the first to study the use of a pattern blast to cave in the old works. In this case blastholes were drilled on a regular pattern without regard for whether the hole was over the void or in the pillar. The blast dimensions were 140 feet north to south and 35 feet west to east. The purpose of the shot was to attempt to collapse the entire area. If pattern blasts were successful a prime advantage could be considerably reduced exploration costs.

Holes were drilled on a 10 foot by 10 foot pattern for the north half of the shot area and on a 12 foot by 12 foot pattern in the south portion of the blast. Hole depths varied from 10 feet to 36 feet. Forty-six blastholes were drilled of which 28 intersected a void. These holes were plugged with a seismic cone as before.

The blastholes which intersected a void were loaded in the same manner as the holes in blast B5. The holes that did not strike a void were loaded with a 3-foot deck of PowerAN at 2 feet above the estimated floor of the room. A second deck of explosive, which was ANFO, was placed to be approximately midway between the first and second decks of the nearest blasthole that penetrated a void. The deck stemming placed between the two charges varied in length depending on the hole depth and was in the range of 4 to 9 feet.

For those holes that did intersect a void the hole was plugged and backfilled so that the bottom of the first charge

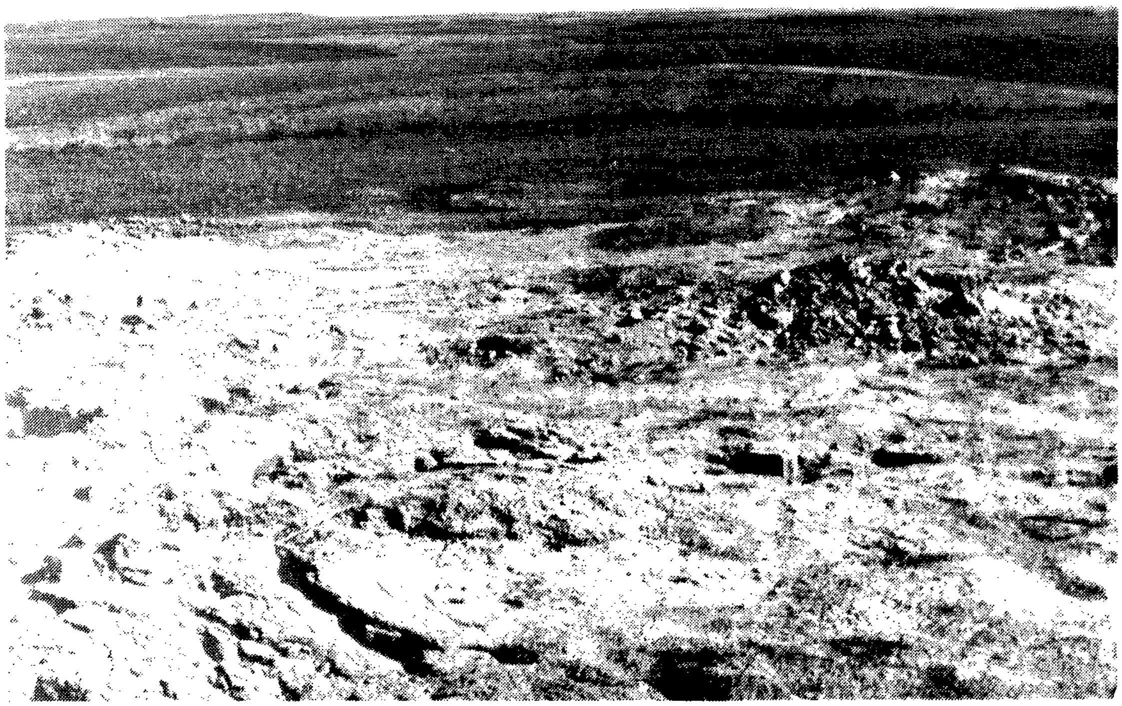


Figure 5-10: PHOTOGRAPH OF THE AFTER-BLAST PROFILE OF B5 VIEWED FROM SOUTH TO NORTH

was 5 feet above the void. The bottom deck was 3 feet long and consisted of 45 pounds of PowerAN. Five feet of stemming was then added and an upper deck of ANFO was placed. This deck was 3 feet long and contained 30 pounds of ANFO.

Stemming heights varied according to the type of hole. Blastholes that struck a void typically had 16 to 17 feet of collar stemming. Those that did not intersect a room usually had more stemming which ranged from 16 to 23 feet. Some shorter holes had less stemming in order to load one deck of explosive. In these cases stemming heights varied from 9 to 12 feet.

The bottom deck was initiated by a cast primer containing a period 4 Primadet. The top decks had a one pound primer containing a period 6 down-the-hole delay. Slider primers were used in conjunction with 15 grain detonating cord.

On surface 42 millisecond NTD delays were used to connect from hole to hole. The blast was initiated at hole 48 which was at the west perimeter of the shot and located in the central entry way. The blast progressed from this hole in an echelon fashion. A maximum of ten decks were detonated per delay. The shot was initiated by an electric blasting cap on September 19, 1988. Four holes redrilled in B4 were shot with this blast to insure collapse which had not been complete with the original shot.

The results of this blast were quite good. The room appeared to have collapsed. This was further confirmed by T.V. camera inspection of the east side of the blast. The surface disturbance was reasonable. Some areas swelled about 1 to 1½ feet while other areas showed slumps of approximately 3 feet. Cracking of the surface was limited to less than 5 feet from the holes. The 12-foot square pattern was observed to have produced equally good results as the 10-foot square pattern. Figure 5-11 is a view of the after-blast result. Blast B6 is just left of center in the photograph.

5.2.8 Blast B7 (September 22, 1988)

This shot was also a pattern blast. It was located at the northeast end of the test site. The mine rooms in this area contained water. The depth of water measured in the blastholes ranged from 1 to 8 feet. The overburden depth averaged 40 feet. The dimensions of the blast area were 63 feet west to east by 67 feet north to south.

For this shot a total of twenty blastholes were drilled. Six holes intersected the voids below and were plugged before loading. Since the 12-foot square pattern had worked well for B6 a 15-foot square pattern was attempted for this blast. The rows of blastholes were laid out diagonally across the blast area from northwest to southeast.

Figure 5-11: AFTER-BLAST VIEW OF SHOT B6 WHICH CAN BE SEEN IN THE LEFT-HALF OF THE PHOTOGRAPH

The holes varied in depth from 17 to 40 feet. Except for one hole all were more than 30 feet deep. When a void was intersected the hole was plugged and backfilled so the bottom of the explosive deck was 5 feet above the void. A 3-foot deck (45 pounds) of PowerAN was placed. A second deck was placed 4 or 5 feet above the first and a third deck was placed 4 or 5 feet above the second. Twelve feet of collar stemming was employed.

When there was no void the bottom deck was located two feet above the estimated mine floor. This deck contained the waterproof PowerAN and was 3 feet in length. Some of these holes contained water. To protect the ANFO in subsequent decks a seismic plug was placed at the top of the water and backfill was placed above the plug. The second deck of 3 feet of ANFO was then loaded an average of 9 feet above the bottom deck. A third deck consisting of 2 feet of ANFO was subsequently loaded after about 5 feet of deck stemming was shoveled into the blasthole. The collar stemming ranged from 13 to 16 feet in these blastholes.

The bottom decks were delayed with a period 4 down-hole delay. Each deck was delayed by 50 milliseconds from the one below. Therefore, period 6 and period 8 delays were used in the succeeding decks. Trojan 16LS slider primers were used in each deck. An Atlas 15 grain detonating cord was used as the downline.

Detonating cord and 42 millisecond delays were used to tie-in the surface. The result was that a maximum of four blastholes detonated simultaneously. Therefore, a maximum of 4 decks detonated together. The progression of the detonation was a diagonal in the vertical sense with bottom decks detonating followed by middle decks on the preceding row and then the top decks two rows back. This arrangement would maximize the relief available.

The surface delay arrangement opened the shot with holes 105 and 110. The detonation progressed from west to east along an open echelon. Holes 95 and 109, in the corners, were detonated last.

Upon blasting the entire area displaced vertically four to six feet and then settled back into place. There were a few areas of surface swell that ranged from 0.5 to 3.0 feet in height. Large chunks were found throughout the blasted area, caused by the surface flex, with the remainder of the area quite flat. There were no depressions resulting from the blast. Cracking of the surface did not extend more than 15 feet. The surface result was deemed quite acceptable.

Underground the result was less acceptable. When the nearest blasthole was more than 2 feet away from the void T.V. studies showed that the void still existed. Where the hole was closer to the void roof material was severely cracked and hanging and the voids were closed. The conclusion was that either charges need to be in contact with the void or greater powder loads should be used.

The primary effect of water was in the pillar holes. The distance between the bottom and middle deck had to be adjusted according to the height of water when ANFO was used in upper decks. Plugs were placed at the top of the water and backfill employed which increased the loading time for the holes and affected powder placement. An alternative approach would be to load all waterproof explosive in these holes. Explosive costs would be increased but loading times and labor cost would be reduced. Water was noted being ejected from exploration holes upon detonation. The presence of water did not seem to create any adverse affect on the blast as a whole. Figure 5-12 is a view of the blast area looking north. Figure 5-13 is a photograph looking south with B7 in the foreground and figure 5-14 is a view looking east. All the photographs show that the surface disturbance and cracking was quite modest.

5.2.9 <u>Test Holes (September 23, 1988)</u>

In an area of B7 where full collapse had not been achieved two holes were shot to test procedures for further shooting areas that did not collapse successfully initially. The first hole, T-1, contained one deck of explosive and the second one, T-2, had two decks.

For T-1 the hole was 36 feet to the void. It was plugged at 34 feet and 3 feet of stemming were added. The explosive deck was then 5 feet above the void and was 4.5 feet long. This deck contained 50 pounds of ANFO. The remainder of the hole was stemmed. When detonated the void surrounding the hole remained but was moved 15 feet closer to the surface.

Therefore, the test hole T-2 was drilled. This hole struck the void at 21 feet and was plugged at 17 feet. One foot of backfill was added followed by three feet of ANFO. Then 3 feet of deck stemming was added followed by 2 feet of ANFO. Eight feet of collar stemming completed the hole loading. When detonated T-2 fully collapsed the surrounding void. The conclusion was that two decks are necessary to collapse a redrilled area. A third might be necessary in a deeper hole than T-2. It appeared to be important to allow for energy absorption by the previously blasted overburden.

5.2.10 <u>Blast B8 (September 24, 1988)</u>

Based on the results from B6 and B7 the final blast, B8, was a pattern shot that incorporated changes to the explosive loading. The blast dimensions were 150 feet north to south by 90 feet west to east. The shot was directly south of B7 and

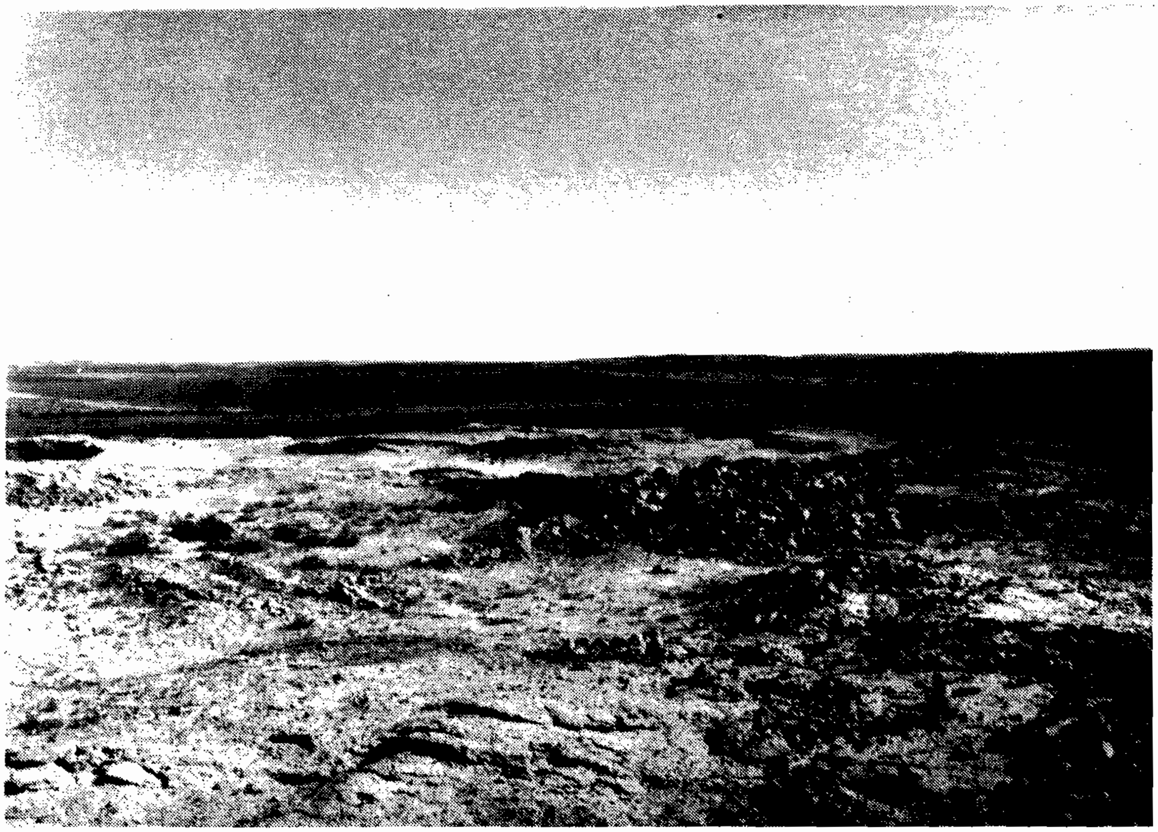


FIGURE 5-12: AFTER-BLAST PROFILE OF SHOT B7
VIEWED TO THE NORTH

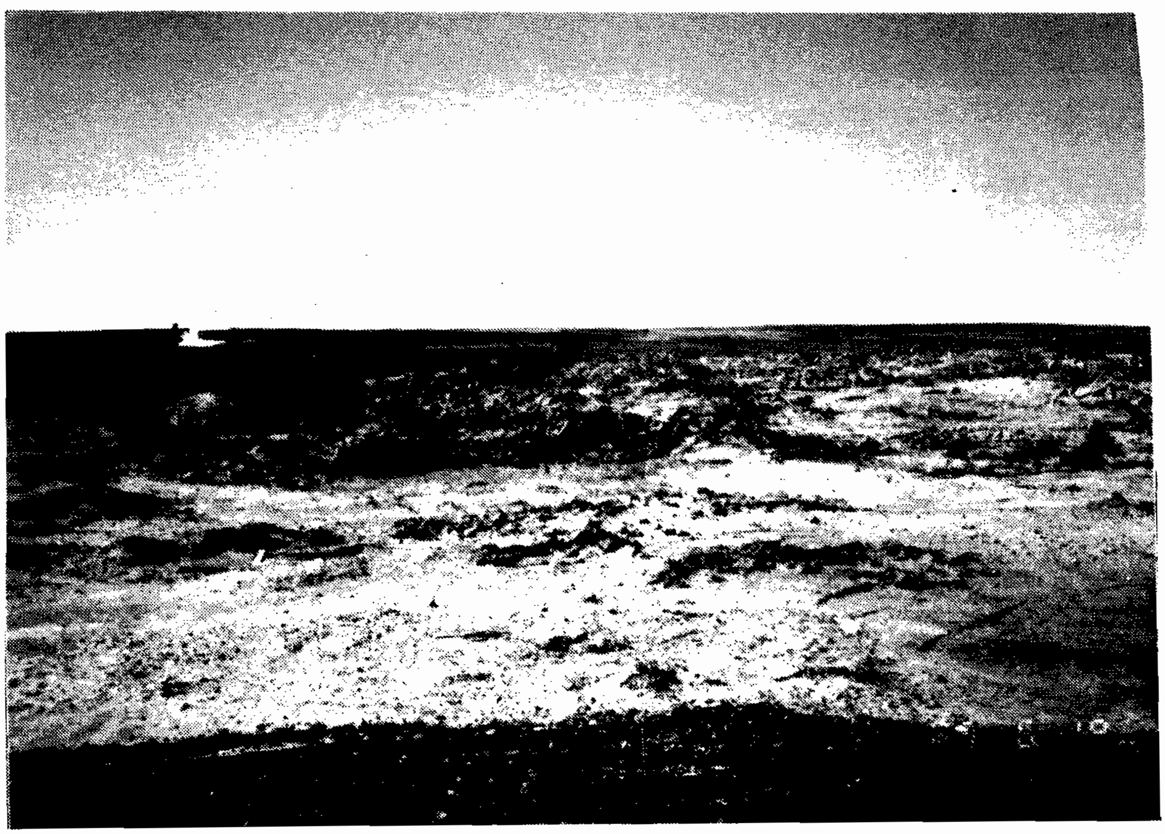


FIGURE 5-13: VIEW OF B7 AFTER BLASTING LOOKING
SOUTH WITH B7 IN THE FOREGROUND

FIGURE 5-14: VIEW OF BLAST B7 LOOKING EAST

east of B6. Two-thirds of the blast area had water in the voids.

This blast consisted of 65 holes drilled on a 15 by 15 foot pattern. Thirty-three holes penetrated the voids while 32 holes were in the pillars and perimeter areas. Observation of the blast map shows that a better distribution of holes in or next to the voids was obtained in this shot than in B7. The hole depths were quite variable ranging from 20 to 49 feet deep. Many holes were drilled a total of 41 feet which appeared to represent the floor elevation of the mine workings.

Holes that intersected the void were plugged and backfilled as required. The bottom deck had a 4 to 5 foot standoff to the old works. A second deck, containing ANFO was placed 5 feet above the first and a third deck was located 5 feet above the second. Some adjustments were made to the deck stemming, as required, to account for variations in hole depths.

When the hole was drilled into a pillar, or around the perimeter, the bottom deck was placed about 2 feet above the floor and loaded with 60 pounds of PowerAN which was an increase of 15 pounds over the B7 loading. The second deck contained 30 pounds (3 feet) of ANFO. This deck was located 5 to 8 feet above the bottom deck. The third deck was increased to 30 pounds of ANFO. It was located approximately 5 feet above the second deck.

The collar stemming was typically ten feet. This blast was at the perimeter of the test area so cracking and surface disturbance were not a hinderance to further blasting work. There was the thought, based on the results of B7 that more explosive was needed and the total overburden needed to be disrupted. However, if increased surface disturbance resulted more post-blasting reclamation might be needed.

Water was found in this area. Plugs were used above the water where these were required. PowerAN was used in the bottom deck where water was encountered. For the one third of the blast that was dry ANFO was used in all the decks. A total of 7,830 pounds of explosive was used of which 4950 pounds was ANFO and 2880 pounds was PowerAN.

A combination of down-the-hole and surface delays were used to control the progression of the shot. The down-the-hole sequence was the same as previous shots with a period 4 delay used in the bottom deck followed by a period 6 and then period 8. The surface delays used were the 42 millisecond NTD units.

The blast was initiated at the northwest end between holes 115 and 116. The delay system resulted in a shot that progressed on a diagonal from northwest to southeast. A maximum of four holes were detonated per delay period.

The result of shot B8 was that the blast completely collapsed the entire area. The surface cracking and disturbance did not progress more than 10 to 15 feet from the holes. Even though the collars were close to the surface the disruption was minor and consisted mostly of small humps. Figure 5-15 is an after-blasting view of the shot looking south. Blast B7 is in the immediate foreground followed by B8. It can be seen that the surface disturbance is modest. Figure 5-16 is a photograph looking east that gives a more detailed view of the surface disturbance. Figure 5-17 is a view to the south also illustrating the degree of disruption of the surface. In all cases this is seen to be quite modest and easily leveled.

Having shot two blasts in workings containing water one observed that the water had little impact on the success of the collapse. However, it is important to be sure that waterproof explosives are used in wet holes or that ANFO be protected against water attack. The primary inconvenience caused by the water is that the loading time is increased (especially if seismic cones are placed as described above) and there is a greater consumption of a more costly waterproof explosive.

5.2.11 <u>Summary</u>

The research program at the White Test Site included the collapsing of individual rooms and the use of pattern shots to collapse entire areas. Methods of minimizing the surface disturbance and cracking, resulting from the shot were studied. An appreciation was gained for the amount of exploration required to blast individual, irregular, poorly documented rooms and entries. This could be compared with that needed for pattern blasting.

The effects of mine workings containing water was studied. In general this did not pose a great problem. One does however need to be aware of proper techniques when loading wet areas.

Figure 5-18 is a view of the blasted area looking from north to south. The adit is at the right side of the photograph. The approximate locations of the blasts are indicated on the photograph.

The picture shows that the surface disturbance was primarily swell. This related to the small void size beneath. It can be observed that the disruption is modest. Topsoil was removed from the area before blasting. It would be a simple matter to level the area, replace the topsoil and seed the acre of land involved. Figure 5-19 is also an overall view of the blasted area taken from the east side and viewing to the west. The surface of the blasted area can be compared with the unblasted topography in the foreground. The swelling of the surface did not exceed five feet.

FIGURE 5-15: VIEW OF BLAST B8 LOOKING SOUTH WITH A PORTION OF B7 IN THE FOREGROUND

FIGURE 5-16: VIEW OF SURFACE EXPRESSION OF BLAST B8 VIEWED TO THE EAST

FIGURE 5-17: VIEW OF SURFACE DISRUPTION FROM BLAST B8 LOOKING SOUTH

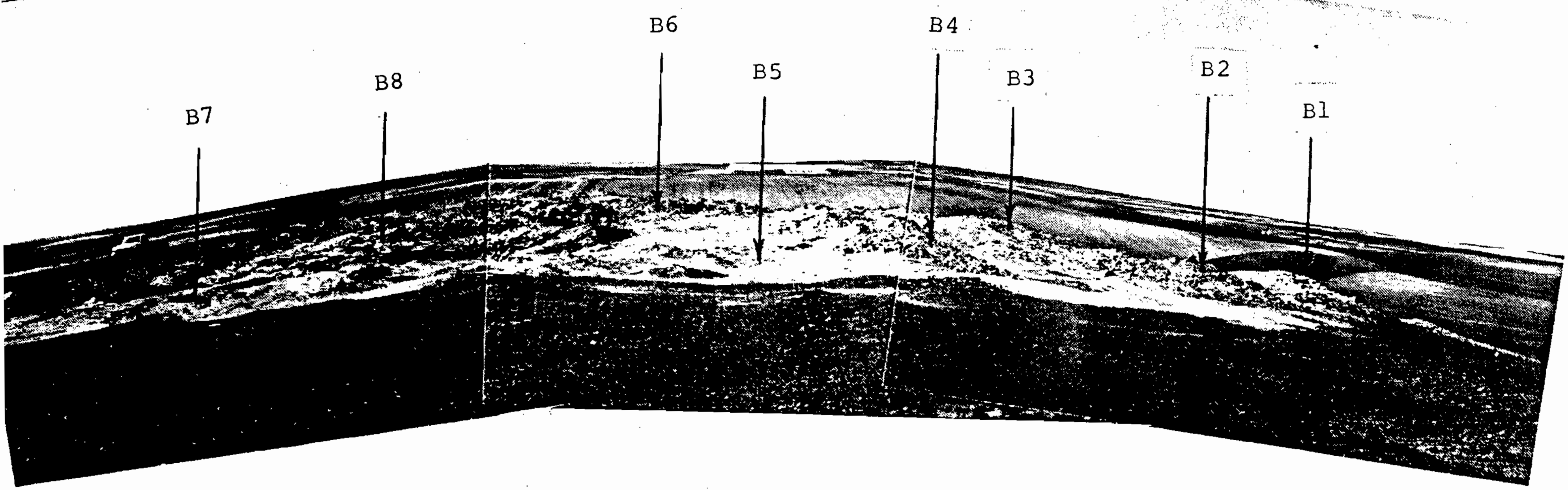


FIGURE 5-18: VIEW OF THE WHITE TEST SITE AFTER BLASTING WAS COMPLETED SHOWING THE APPROXIMATE LOCATION OF EACH BLAST AND THE DEGREE OF SURFACE DISTURBANCE

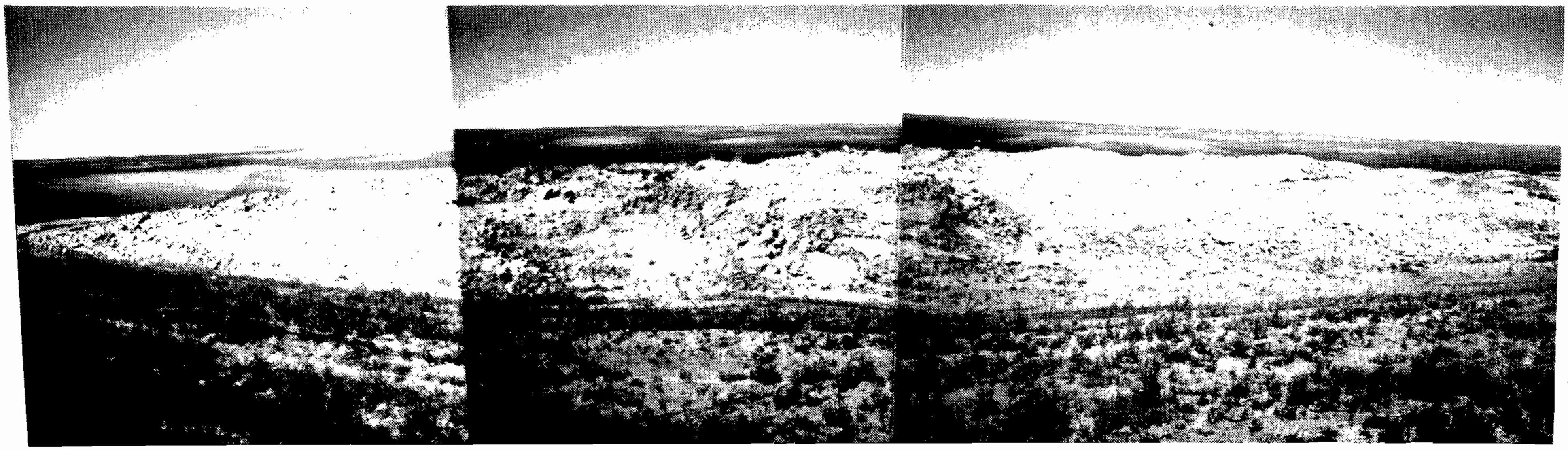


FIGURE 5-19: EAST TO WEST VIEW OF THE BLASTED AREA ILLUSTRATING THE FINAL SURFACE PROFILE

CHAPTER 6: HIGH-SPEED CAMERA STUDIES AT THE BEULAH AND WHITE TEST SITES

6.1 INTRODUCTION

The high-speed camera is an important tool for the analysis of blasting performance. Blasting events proceed much too rapidly for evaluation by the unaided eye or by normal filming and videotaping methods.

A high-speed camera was used during this research to monitor many of the blasts The equipment used was a Red Lake Locam capable of framing speeds up to 500 frames per second. The unit was battery powered and included a D'Agineau zoom lens.

There were several purposes for use of the camera in this study. One was to be able to examine the overall blasting event. Others were to study the accuracy of the surface delays used, to be able to study any unusual occurrences of collapse and to monitor any bursting, uncontrolled top movement, misfires and so forth.

The primary problem with the use of the high-speed camera for this work is that it is difficult to position the unit to get a good view of the blast. In mines, by contrast, the camera can be placed on the bench above to gain elevation or across the pit from the blast.

To reduce the problem the camera was often mounted on five or ten feet of scaffolding to gain elevation. This was helpful but did not eliminate all visibility restrictions. At the White site the unit was usually located on top of the topsoil stockpile north of the mine area and on scaffolding.

Fifteen films were taken between the two sites. Eight blasts were filmed at the Beulah Site. One of these was spoiled due to a camera malfunction. At the White testing area seven blasts were filmed. Of these one was over exposed during development and one was not analyzed because there was little or no surface manifestation of the shot. A total of twelve films were studied. Table 6-1 is a listing of the films taken.

6.2 FIELD SETUP

Prior to each shot monitored the high-speed camera was set up in an appropriate location. Selection of a suitable place for the camera included finding a spot with good visibility. Also one wanted the camera located so that the blast was progressing toward the Locam, otherwise the shot would become obscured by holes detonation early in the event.

TABLE 6-1: SUMMARY OF ANALYSIS OF HIGH-SPEED CAMERA FILMS TAKEN AT THE WHITE AND BEULAH TEST SITES

Film Supplied: 15
Processed: 12
Non-Processed: 3

Blast No.	Film Speed (ms/frame)	Comments
B1	2.5	Analyzed
B2	---	Film over-exposed, no picture
B3	2.4	Analyzed Film partially over-exposed
B4	---	No massive surface movement, just some gas ejections. Holes and delay numbers are hard to identify.
B6	2.5	Analyzed
B7	2.5	Analyzed
B8	4.0	Analyzed
VO-6	2.0	Analyzed
VO-7&8 North	2.0	Analyzed
VO-7&8 #2	---	No start
VO-7&8 South	2.0	Analyzed
VO-9	2.0	Analyzed
VO-11	2.0	Analyzed
VO-12	2.9	Analyzed
VO-13	4.0	Analyzed

The scaffold was setup at the chosen location unless the visibility was suitable at ground surface. The camera was mounted on the tripod and battery power was connected. Light meter readings were taken and appropriate lens and framing speed settings were made.

For each shot certain data had to be recorded for use in analysis. This included the azimuth and distance to the blast, frames per second, f-stop, zoom setting, focus, light meter reading and vertical angle of the camera. Azimuth and vertical angle were measured with a Brunton compass.

The Locam does not reach the selected framing speed immediately. It is important that the camera run for a short period before detonating the blast. For this work it was possible to fire the shots from the same point as the camera setup. The blaster counted down from ten and the camera was manually activated on the count of three. Therefore, the camera ran for about three seconds before the shot, allowing adequate time to achieve full framing speed.

6.3 FILM RESULTS AT THE BEULAH TEST SITE

Blasts were filmed at Beulah beginning with vertical opening VO-6. The surface delay system was analyzed on these films. Also, the films were observed for any variations to the expected firing rotation. The latter could be analyzed in a qualitative way but not quantitatively. Therefore the primary quantitative data was the surface delay time analysis.

6.3.1 Qualitative Analysis

Each film was examined for the general result of the blast. The nature of vertical openings makes it difficult to see what happens in the opening during the shot. However, the film could be checked for the rotation, surface movement and so forth.

The film for VO-6 was taken from a position 450 feet northeast of the vertical opening. The film showed that the holes detonated as planned, beginning with hole 1 on the east side and ending with hole 6 on the west side. Ground disturbance was significant.

When analyzing the record of VO-7 and 8 North it was found that one or two holes were out of frame. Also a few holes were obstructed by fumes. The film showed considerable surface disruption with substantial surface heave. Early in the blasting event (350 ms.) considerable disruption was noted north of the voids. This appeared to be the ramp to the workings collapsing. The phenomenon occurred before holes between the vertical opening had fired. However, there was no post-blasting observation of misfires and no unusual airblast.

There was no uncontrolled flyrock. The camera position was to the east south east of the shot.

When vertical opening VO-7 and 8 South were detonated the camera was located 300 feet Northeast of the blast. This film was taken at 500 frames per second. The result showed much displacement of the ground around the vertical openings. There was no uncontrolled flyrock. The detonation of the holes appeared correct.

Vertical opening VO-9 was filmed from a position 311 feet to the southwest at a setting of 500 frames per second. Examination of the film showed that all holes fired. The rotation was correct. The surface displaced considerably upon detonation but there was no flyrock. The film indicated that a significant area around the shot had settled.

Blast VO-11, in test area 3, was filmed from a position 300 feet southeast of the sinkhole at 500 frames per second. All holes were visible in the record except 21 and 22 which were out of frame. The record showed that all holes fired and that the rotation was correct. Surface disturbance was similar to previous results. There was no uncontrolled flyrock.

Blast VO-12 was filmed with the camera placed 300 feet southwest of the opening. The framing speed was 350 frames per second. All blastholes were initiated in the correct rotation. The surface displaced sufficiently to be fragmented but the blast was sufficiently controlled to avoid any flyrock.

The final blast was filmed from a location directly west of the shot. The blast was detonated late in the afternoon and the framing speed was 250 frames per second. On this film the zero time was difficult to define so accumulated delay times were not meaningful. Seven holes were obscured and could not be analyzed. The primary problem was with holes firing to the east from the initiation point midway along the north side row (hole 16). These were obscured by detonations to the west of the initiation point. For those holes that could be seen all fired as expected. Material could be seen being cast into the central depression There was no flyrock.

For all the shots filmed several things were noted. There was no indication of flames ejecting upward from the detonation. This indicated that there was no bursting of detonation gases through the surface that would indicate too little stemming. Associated with this was the fact that there was no flyrock ejected from the blasting area for any of the blasts filmed. This would be of particular importance if blasting near residences or commercial buildings. The films did show that the loading plan for the holes caused substantial vertical surface displacement which led to the overburden being fragmented throughout the column. Thus, it was confirmed that a nine-foot stemming height was near optimum for this work. Although difficult to see, because the camera could not be positioned to look directly into the sinkholes, there was the indication that the material was well displaced toward the opening.

6.3.2 Analysis of Delay Times

For each shot filmed at the Beulah site the firing times of the 42 millisecond delays was determined. Some delays were not analyzed because these were obscured by prior detonations or were out of frame. The delay elements, primacord pigtails and a short length of the Nonel tube of the succeeding delay were buried to avoid airblast. However, if the flash of the delay itself could not be seen the outgoing detonation of the tube from the next delay was clearly visible. Even if as much as 5 feet of the tube were buried this would not introduce an error of more than 0.83 milliseconds, assuming the Nonel tube detonated at 6000 feet per second. This represents less than one frame on the film.

Tables 6-2 to 6-8 show the results of the delay time analysis for each blast filmed at Beulah. These tables list the hole numbers, frame count, total delay time and the delay interval between successive holes. At the bottom of each table the sample mean, sample standard deviation, maximum and minimum are also shown. Comments concerning the analysis are provided where appropriate.

Examination of these tables indicates that the mean times are not much different than the nominal delay times. The standard deviation of the sample does indicate a significant spread in the delay times however. This is also indicated when studying the minimum and maximum values.

Many of the 42ms delays used at Beulah had been in storage for a year. Often delays show additional scatter with aging and this may have contributed to the variation seen in these film studies.

In the film of blast VO-9 one delay appeared to fire after a delay of 82ms. This seems an unusually large variation from the nominal time. It is possible that the flash was obscured and identified late. However, the possibility does not fully explain the long delay because over the 40 milliseconds beyond the nominal time the detonation could travel 240 feet, but the lead length of the delays used was only 30 feet. Even if the flash was detected after 25 feet of Nonel tube, of the next delay, had detonated this would only introduce an error of 4.2 ms. If this delay is ignored then the mean time is 43.4 ms and the standard deviation of the sample is 7.5, still a significant spread but appreciably better than the case when all delays are considered.

TABLE 6-2: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST VO-6 AT BEULAH

Blast VO-6
Film Rate2.0 ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
1	0	0	0
2	21	42	42
3	36	72	30
4	60	114	42
5	83	160	46
6	112	218	58
		Mean	43.6
		Maximum	58.0
		Minimum	30.0
		St. Dev.	9.0

TABLE 6-3: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLASTS VO-7 & 8 NORTH AT BEULAH

Blast VO-7&8-N
Film Rate 2ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
7-14	0	0	0	
7-15	21	42	42	
7-16	42	84	42	
7-17	67	134	50	
7-18	86	172	38	
7-19	110	220	48	
7-20	133	266	46	
7-21	164	328	62	
7-22	195	390	62	
8-11	213	426	36	
7-23	236	472	46	
8-12	256	512	40	
7-24	277	554	42	
8-13	303	606	52	
7-25	319	638	32	
8-14	342	684	46	
7-26	362	724	40	
8-15				Unable to define clearly
7-27				" " " " " " "
8-16				" " " " " " "
8-10	426	852		
8-9	449	898	46	
8-8	472	944	46	
8-7	496	992	48	
8-6	511	1022	30	
8-5	533	1066	44	
8-4				Unable to identify
8-3				" " " " "
		Mean	44.7	
		Max	62.0	
		Min	30.0	
		St. Dev.	7.8	

TABLE 6-4: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLASTS VO-7 & 8 SOUTH AT BEULAH

Blast VO-7&8-S
Film Rate 2.0ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
12	0	0	0
11	15	30	30
10	33	66	36
9	51	102	36
8	69	138	36
7	89	178	40
6	117	234	56
5	132	264	30
4	149	298	34
3	167	340	42
2	191	386	46
1	218	436	50
17	236	472	36
18	246	492	20
19	262	524	32
20	274	560	36
21	298	608	48
22	317	648	40
		Mean	38.1
		Max	56.0
		Min	20.0
		St. Dev.	8.3

TABLE 6-5: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST VO-9 AT BEULAH

Blast VO-9
Film Rate 2ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
1	0	0	0
2	20	40	40
3	61	122	82
4	84	168	46
5	108	216	48
6	128	256	40
7	144	288	32
8	165	330	42
9	193	386	56
		Mean	48.3
		Max	82.0
		Min	32.0
		St. Dev.	14.3

TABLE 6-6: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST VO-11 AT BEULAH

Blast VO-11
Film Rate 2ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
1	0	0	0	
2	21	42	42	
3	42	84	42	
4	69	138	54	
5	84	168	30	
6	114	228	60	
7	131	262	34	
8	146	292	30	
9	169	338	46	
10	193	386	48	
11	213	426	40	
12	234	468	42	
13	257	514	46	
14	279	558	44	
15	298	596	38	
16	320	640	44	
17	338	676	36	
18	356	712	36	
19	376	752	40	
20	395	790	38	
21				Hole out of frame
22				Hole out of frame
		Mean	41.6	
		Max	60.0	
		Min	30.0	
		St. Dev.	7.3	

TABLE 6-7: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST VO-12 AT BEULAH

Blast VO-12
Film Rate 2.9ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
1	0	0	0
2	12	34.8	34.8
3	27	78.3	43.6
4	42	121.8	43.5
5	55	159.6	37.7
6	68	197.2	37.7
7	81	234.9	37.7
8	93	269.7	34.8
9	108	313.2	43.5
10	122	353.8	40.6
11	136	394.4	40.6
12	149	432.1	37.7
13	163	472.7	40.6
		Mean	39.4
		Max	43.6
		Min	34.8
		St. Dev.	3.0

TABLE 6-8: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST VO-13 AT BEULAH

Blast VO-13
Film Rate 4ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
16	0	0	0	Zero was difficult to define
17				Therefore total delay would
15	8	32	32	Be incorrect
18	11	44	44	
14				
19	21	84	40	
13				
20	32	128	44	
12	40	160		
21	42	168	40	
11	41			
10	57	228	60	
9	69	276	48	
8				
7	87	348		
6	96	384	36	
5	106	424	40	
4	114	456	32	
3				
2	131	524		
1	143	572	48	
		Mean	42.2	
		Max	60.0	
		Min	32.0	
		St. Dev.	7.7	

The film of blast VO-7 and 8 north indicated that holes 7-27, 8-15 and 8-16 could not be identified clearly and therefore were not analyzed. Something occurred somewhat earlier in the blast that obscured these holes. It appeared that the ramp down to the workings at the north end of sinkhole VO-7 collapsed as the holes along the west side of the opening detonated accompanied by much dust and disruption. There was no observation of flame from holes on the east side of the opening which would have indicated loss of burden as a result. Nor was there any post-blasting observations of misfires. However, this result does suggest that it is wise to watch the duration of total delay times around such ramps.

Examination of the tables shows that in all cases the initiation of the blastholes was in the expected order. However, the variation in time could lead to out-of-rotation firing of the deck charges when the down-the-hole delays are taken into account.

From the timing analysis of the films taken at Beulah one concludes that the average firing times are fairly close to the nominal time. However, one can also see that there is significant scatter in the individual firing times which could alter the firing sequence of the individual explosive decks.

6.4 WHITE TEST SITE

6.4.1 Qualitative Analysis

The qualitative analysis of the films was performed to look for unusual events, surface movement and any misfires or unexpected firing times. This information was recorded as seen.

The film of blast B1 indicated that all holes were initiated as expected. There were no indications of any misfires occurring. The film showed substantial surface movement throughout the area. The detonation of the last two holes was partially to completely obscured by previous detonations. It was observed that there were strong white gas ejections occurring through unloaded holes about 50 ms after surrounding loaded holes fired. These unloaded holes were from the exploration program to identify the underground workings.

The film of blast B3 allowed good visibility for most holes detonated. There was considerable surface movement associated with this blast. The surface disruption was somewhat uneven because some holes had a 12-foot stemming height, rather than 9 feet as was the case for the rest of the shot. There was no indication of any misfires or premature initiations. The camera was located 450 feet north of this blast.

The film for B4 showed that there were no massive surface movements during the detonation. There were only some gas ejections resulting from this blast. The shot was loaded with only one deck of powder and had long stemming heights in an attempt to reduce the surface disturbance generated in previous shots. The film confirmed that there was virtually no surface movement resulting from this blast. However, subsurface investigation had shown that caving was not complete and voids had been moved closer to the surface. The holes and delay numbers were hard to identify on this shot so a delay time analysis was not completed.

The film of B6 had good visibility and a full delay time analysis was possible as well as an overall view of the shot. The analysis showed that all delays detonated in the planned sequence. There was no evidence of any misfires. The surface showed significant heave during the blast. The camera location was 270 feet northeast of the shot and the film was taken at 400 frames per second.

Blast B7 was a pattern blast with several holes detonated per delay period. The film was taken from 270 feet northeast of the shot at 400 frames per second. The film showed that the holes were initiated in the designed sequence. There was limited ejection of stemming or gases from the blastholes. It was observed that a good overall heave development was obtained. There was a strong gas ejection near blasthole 103 at 174 ms which was 71.5 ms after that hole had been initiated. There was no indication of any misfires.

Blast B8 was a large pattern blast directly south of B7. The film was taken 270 feet north of the shot at 250 frames per second. This shot detonated from north to south so it was difficult to observe the overall action of the shot. A complete delay time analysis of all holes was not possible. Surface movement of some holes was not clear. Near delay 9 red flames from explosive detonation ejecting upward were noted. Stemming heights were not unusually low in this area, but the bursting may have resulted from structural weakness in the ground. Although somewhat obscured the surface movement seemed reasonably uniform throughout the blast.

6.4.2 Analysis of Surface Delay Times

The accuracy of the surface delay times at the White Site was analyzed. Some delays were not analyzed because the event was obscured by dust and fumes. However, every 42 millisecond delay that could be viewed was studied. A total of 37 delays were analyzed at this site

The results of the study of delay times for each blast filmed (except B4) are presented in tables 6-9 through 6-13. These tables provide the same data as was previously presented for the Beulah Site.

TABLE 6-9: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST B1 AT THE WHITE TEST SITE

Blast White-B1
Film Rate 2.5ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
1	0	0.0	0.0	
2	11	27.5	27.5	
3	22	55.0	27.5	
4,6	37	92.5	37.5	
5,7	50	125.0	32.5	
8	66	165.0	40.0	
9	79	197.5	32.5	
10	92	230.0	32.5	
11				Partially covered by last delay
12	121	302.5	72.5	Obscured by dust and fumes
		Mean	32.9	
		Max	40.0	
		Min	27.5	
		St. Dev.	4.3	

TABLE 6-10: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCUALTED FROM THE HIGH-SPEED FILM OF BLAST B3 AT THE WHITE TEST SITE

Blast White-B3
Film Rate 2.4ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
39	0	0.0	0.0	
38	19	45.6	45.6	
37	35	84.0	38.4	
36	53	127.2	43.2	
35	69	165.6	38.4	
34				Unable to identify
32	134	321.6		Northwest room
31	150	360.0	38.4	" " "
30	166	398.4	38.4	" " "
33	183	439.2	40.8	" " "
27	103	247.2		Northeast room
26	118	283.2	36.0	" " "
25	134	321.6	38.4	" " "
		Mean	39.7	
		Max	45.6	
		Min	36.0	
		St. Dev.	2.8	

TABLE 6-11: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST B6 AT THE WHITE TEST SITE

Blast: White-B6
Film Rate:2.5ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
48	0	0.0	0.0
77,80	15	37.5	37.5
74,81,85,84,83 49,50,51,54,61	31	77.5	40.0
72,78,82,87,86 65,69,75,79,89	39	97.5	20.0
88 62,66,73,76,91	46	115.0	17.5
90,93	57	142.5	27.5
55,63,70,57,92	71	177.5	35.0
52,56,67,71	84	210.0	32.5
53,64,68	94	235.0	25.0
58,60	105	262.5	27.5
59	118	295.0	32.5
		Mean	29.5
		Max	40.0
		Min	17.5
		St. Dev.	7.0

TABLE 6-12: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST B7 AT THE WHITE TEST SITE

Blast: White-B7
Film Rate: 2.5ms/frame

Hole Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.
105,110	0	0.0	0.0
100,101,106,113	16	40.0	40.0
96,102,107,111	26	65.0	25.0
97,98,103,112	41	102.5	37.5
94,99,104,108	59	147.5	45.0
95,109	74	185.0	37.5
		Mean	37.0
		Max	45.0
		Min	25.0
		St. Dev.	6.6

TABLE 6-13: SURFACE DELAY TIMES FOR FORTY-TWO MILLISECOND SURFACE DELAYS AS CALCULATED FROM THE HIGH-SPEED FILM OF BLAST B8 AT THE WHITE TEST SITE

Blast White-B8
Film Rate 4ms/frame

Delay Numbers	Number of Frames	Total Delay ms.	Delay Interval ms.	Comments
1	0	0	0	
2	10	40	40	
3	19	76	36	
4	30	120	44	
5	40	160	40	
6	51	204	44	
7				Not found from film
8	69	276		Remaining delay times are
9	79	316	40	obscured by dust and fumes
		Mean	40.7	
		Max	44.0	
		Min	36.0	
		St. Dev.	2.7	

A review of these tables shows that the average times are somewhat lower than the nominal firing times of the 42 ms delays. The standard deviation of the samples do indicate some significant scatter in the delay times. The blast that suffered most from this was B6. Scatter was less than at Beulah however.

In table 6-9 it can be seen that the last delay analyzed showed a large variation from the 42 ms nominal time. However, there was difficulty viewing this hole because of the camera location relative to holes 10, 11 and 12. It is believed that the time seen here in fact represents the initiation of hole 12 and is the combined delay time from hole 10 to 11 and then to 12. When this delay is removed the mean is 32.9 ms and the standard deviation is 4.7. The maximum time in the sample is then 40 ms.

In table 6-11 the timing results are seen to be widely scattered. These delays detonated at times consistently less than the nominal 42 ms time expected. The variation is the greatest seen in any shot detonated at either test site. There is no clear reason for the variations seen.

From the tables one sees that in all cases the sequence of the shots was in the expected order. There were no cases of premature detonation. Again there was sufficient variation in the initiation times to lead to some overlapping of individual decks.

Review of the analysis indicates that the average firing times were lower than the nominal time. However, the standard deviation shows that significant scatter occurred which was further confirmed by the sample minimum and maximum times.

6.5 SUMMARY OF DELAY TIME ANALYSIS

The statistics for all of the delays at Beulah and the White test areas have been calculated. Also the mean and standard deviation has been computed for the total sample of both sites. The results are reported in table 6-14.

At the Beulah Site the calculations included the one delay with a large deviation from nominal (82 ms) since there was no reason related to the filming for this value to be suspect. The 72 ms reading obtained in blast B1 at the White Site was discarded since the value could not be read with certainty from the film and may have pertained to the succeeding hole.

From table 6-14 it can be seen that the average firing time for the delays was very close to the nominal times. The table also shows that the standard deviation of the sample is indicative of significant scatter in the delays. This is further borne out by the sample maximum and minimum times. One reason for the variation may be the shelf life of one year experienced by many of these delays.

TABLE 6-14: OVERALL STATISTICS FOR THE ACTUAL FIRING TIMES OF FORTY-TWO MILLISECOND SURFACE DELAYS AT THE BEULAH AND WHITE TEST SITES

Item	Beulah Site	White Site	Combined Sites
Number of Delays Analyzed	93	37	130
Sample Mean	42.1	35.4	40.2
Standard Deviation	8.8	6.8	8.8
Sample Maximum	82	45.6	82
Sample Minimum	20	17.5	17.5

The statistics for the White test area indicate delays that consistently shot at a somewhat lower time than nominal. This can also be seen by reviewing the individual tables presented earlier. The standard deviation of the sample indicates that these delays exhibited less scatter around the mean than did the surface delays used at Beulah. This fact is further attested to by the sample maximum and minimum times.

The combined sample shows delays that had a mean time 1.8 millisecond less than the nominal times which is about 3.5 percent below the stated time. Again the sample standard deviation indicates significant scatter in the delays which was often 20 per cent of the stated time. The total variation from minimum to maximum times was 64.5 ms or 1.5 times the nominal delay time.

The scatter in these 42 millisecond delays seems large compared to other 42 ms delays we have studied (4, 14). The only reason, for the Beulah site at least, could be the age of the delays. The delays for the White Site were a different order and the age is uncertain.

6.6 EFFECT OF DELAY VARIATION ON THE FIRING SEQUENCE

The question that arises is how much the delay time variation affects the firing rotation of the blast compared to the designed sequence. To determine the effect two blasts, for which quite complete data was obtained, were analyzed for both the Beulah and White Test Sites. The results are presented below.

6.6.1 Actual Firing Times for Two Blasts at Beulah

Table 6-15 shows the hole number, delay interval and actual deck firing times for blast VO-7 and 8 south.

TABLE 6-15: ACTUAL DECK FIRING TIMES FOR BLAST VO-7 AND 8 SOUTH AT BEULAH USING THE CALCULATED ACTUAL SURFACE DELAY INTERVALS

Hole Number	Delay Interval ms.	Deck Firing Times ms.
12	0	100
11	30	130, 180
10	36	166, 216
9	36	202, 252
8	36	238, 288, 313
7	40	278, 328, 353
6	56	334, 384, 409*
5	30	364, 414*,439*
4	34	398, 448
3	42	440*, 490*
2	46	486*, 536*
1	50	536*
17	36	572, 622*
18	20	592, 642
19	32	624*, 674
20	36	660, 710* 735
21	48	708*, 758, 783
22	40	748, 798, 823

* Decks firing less than 8 ms. apart

In this case there were six cases where the time between decks detonating was less than 8 milliseconds. In one case two decks fired simultaneously. In no case did a subsequent deck detonate before a prior deck. The firing times do assume the nominal down hole delay times since these delay elements were not instrumented. However, the analysis gives a good idea of what happens due to variations in delay times.

The firing of two decks in less than 8 milliseconds was not a major problem in this case because housing was a considerable distance away relative to the light deck charges employed. If blasting were being conducted closer to built up areas this could be more of a problem as blast vibration levels become more critical.

The primary disadvantage in this instance from decks firing close together is the decrease in freedom to move material toward the void. However, post-blasting observation did not yield any clear indication that variations in deck firing times created a problem with filling the void.

Table 6-16 shows actual firing times for explosive decks in blast VO-11. As before the nominal times are assumed for the down-the-hole delays. Therefore, the analysis is not exact but provides an idea of how delay times scatter could affect the results.

TABLE 6-16: FIRING TIMES FOR EXPLOSIVE DECKS IN BLAST VO-11 BASED ON THE CALCULATION OF ACTUAL DETONATION TIMES OF THE SURFACE DELAYS

Hole Number	Delay Interval ms.	Actual Firing Times ms.
1	0	175
2	42	217
3	42	259
4	54	238, 288
5	30	268, 318
6	60	328, 378
7	34	362, 412
8	30	392, 442*
9	46	438*, 488*
10	48	486*, 536
11	40	526, 576
12	42	568, 618*, 643
13	46	614*, 664*
14	44	658*,708,733*
15	38	696, 746*,771*
16	44	740*, 790
17	36	776*, 826
18	36	812, 862
19	40	852, 902
20	38	965
21	--	---
22	--	---

* Decks firing less than 8 ms. apart

The tabulated data shows seven cases where decks fired in close proximity. In this instance the smallest spread was 2 milliseconds. In all cases the decks fired in the planned sequence. However, the close timing would be a concern regarding vibration if blasting close to housing.

An examination of the tie-in for blast VO-11 shows that the decks that did fire close together would not have had an important impact on fragmentation or material movement provided the down-the-hole delays detonated close to the nominal firing time. Observation of the blast gave no indication that it had been choked by inadequate relief. This blast consisted of two rows so out-of-sequence detonations could have created severe problems if back row holes had fired before front row holes.

Table 6-17 shows the firing times for blast B1 at the White Site. These calculations also assume nominal times for the down-the-hole delays.

TABLE 6-17: FIRING TIMES FOR EXPLOSIVE DECKS IN BLAST B1 BASED ON THE CALCULATION OF ACTUAL DETONATION TIMES OF THE SURFACE DELAYS

Delay Number	Hole Number	Delay Interval ms.	Deck Firing Times ms.
1	1	0	100, 150*
2	2	27.5	127.5, 177.5
3	3	27.5	155*, 205
4	4, 6	37.5	192.5, 242.5
5	5, 7	32.5	225, 275
6	8	40.0	315, 365
7	9	32.5	297.5, 347.5
8	10	32.5	330, 380
9	12	72.5	Interval Time Uncertain

*Decks firing less than 8 ms apart

In this case there is only one instance where decks detonated less than 8 ms apart. Thus, while there was significant scatter in the surface delays it had little impact on the effectiveness of the blast.

Table 6-18 lists the firing times for individual explosive decks in blast B6. This was a pattern blast and several holes were initiated per delay period. Therefore, overlapping firing times could have a greater impact on vibration and blasting results.

TABLE 6-18: FIRING TIMES FOR EXPLOSIVE DECKS IN BLAST B6 BASED ON THE CALCULATION OF ACTUAL DETONATION TIMES OF THE SURFACE DELAYS

Delay Number	Hole Numbers	Delay Interval, ms.	Deck Firing Times, ms.
1	48	0.0	100, 150
2	77, 80	37.5	137.5, 187.5
3	74, 81, 85, 84, 83	40.0	177.5, 227.5
4	49, 50, 51, 54, 61	----	----
	72, 78, 82, 87, 86	20.0	197.5, 247.5*
5	65, 69, 75, 79, 89, 88	17.5	215, 265
6	62, 66, 73, 76, 91, 90, 93	27.5	242.5*, 292.5
7	55, 63, 70, 57, 92	35.0	277.5, 327.5
8	52, 56, 67, 71	32.5	310, 360*
9	53, 64, 68	25.0	335, 385
10	58, 60	27.5	362.5*, 412.5
11	59	32.5	395, 445

* Decks firing less than 8 ms. apart

In this blast there were two cases of holes that detonated in close proximity to one another. Since the tie-in of the pattern blast consisted of multiple holes per delay this meant several decks firing close together. For the case of delay numbers 4 and 6 seventeen decks would have detonated 5 milliseconds apart. For delay numbers 8 and 10 six decks would have fired 2.5 ms. apart.

At the White test area the nearest residences were quite far away. Therefore, the overlapping of even 17 decks was not a particular concern. This could become quite critical, however, when blasting in closer proximity to built-up areas. Therefore, testing of both surface and down-the-hole delays may be prudent if close in-pattern blasting is contemplated. In addition these circumstances would warrant the use of more surface delays to reduce the weight of powder shot per delay period and minimize the potential vibration problems if some overlaps did occur.

From an examination of the blast tie-in it does not appear that the close firing times noted would have had a severe impact on blasting results. The results of the shot did not indicate a major problem resulting from the situation. The fact that the detonations are directed downward and not to a usual free face would make the situation less critical.

In summary it can be said that the delay time spread would have led to cases of decks detonating in close proximity to one another. The primary concern would be that blast vibration effects could be amplified in cases where blasting was conducted close to residential areas. The impact on blast results would be less critical in most cases. For the examples presented above there was no reversal of firing sequence. However if an adverse spread of 10 milliseconds or even less occurred in the down-the-hole delays such reversals could have occurred. To completely characterize the deck firing times would require a full analysis of the in-hole delays as well as surface products. In general, however, scatter time in the surface delays did not pose a great problem in this research.

CHAPTER 7: VIBRATION ANALYSIS AND CONTROL

7.1 INTRODUCTION

Nearly all of the blasting events in this research were monitored using a blasting seismograph. The blasts for vertical openings VO-1 and VO-2 at Beulah were not monitored because the equipment had not yet arrived. Blasts VO-4A and VO-10 at Beulah were monitored but data was not obtained due to equipment malfunction.

At the White Site in Daniels County, Montana all eight blasts were monitored and data was obtained. The test blasts, consisting of two single hole shots were not measured.

In total twenty-one blasting events were recorded. The data obtained has been analyzed specifically by property, in total and in conjunction with previously gathered information.

There were two reasons for monitoring the blasts with a seismograph:

1. To insure that there was no damage to any nearby structures and to monitor the blast induced vibration and airblast effects relative to known human response criteria.

2. To gather data which would be useful to those using blasting for AML reclamation in the future.

The first reason was, obviously, to insure that the current field research did not lead to any adverse blasting effects. Also, one wanted to keep the blasting below human response levels whenever possible to avoid complaints and monitoring the shots was therefore important.

The second reason was equally important. Often there has been a reluctance to employ blasting in AML work because of concerns about blast vibration and airblast. These concerns may well be overstated, provided proper blasting principles are applied. The blasting approach tested in this research, utilizing deck loaded blastholes, provides suitable blasting technology and a means of reducing blast vibration to low levels.

The blasting seismograph used was an SSU-1000D supplied by Berger (Geosonics) and manufactured by Nomis. The unit was programmable and had the facility to record events on a $3\frac{1}{2}$-inch floppy disk. The disk feature was not used. The unit also printed out each event on a stripchart and provided a summary of important data. Figure 7-1 is a photograph of the unit with the vibration monitor in the foreground and the airblast microphone pointed toward the blast. Figure 7-2 is a typical vibration record obtained from the equipment.

The SSU-1000D utilizes three perpendicularly oriented normalized electrodynamic transducers to measure longitudinal, transverse and vertical vibration. The frequency range is 2 to 256 Hertz and the measuring range is to 4.0 ins/sec.

The sound monitor is on the F (flat) weighting scale. The measuring range is 110 to 140 dB or .0009 to .0296 psi. The frequency range is 2 to 256 Hertz (-3dB at 2Hz) on 5 second recording time. The accuracy is ±0.7dB at reference point (127 dB peak, 250 Hz continuous sine wave input).

On a 5 second recording time the unit samples at a rate of 1024 samples per second per channel. The operating temperature range is 35° to 95° F. However, the unit will work from 0° to 120°F with reduced legibility of the print out.

7.2 DISCUSSION OF VIBRATION PRINCIPLES

7.2.1 Blast Generated Ground Vibrations

Once the zone in which the shock front from the detonation produces failure of the rock mass has been passed, the strain pulse attenuates into an oscillatory wave in which ground particles move along cyclically repeating orbits. From this stage on the energy radiating from the explosion will produce particle motion in the ground which are within the rock's elastic limit. The energy propagates as elastic waves and these are the basis of blast vibrations. The energy travels in the form of kinetic energy of particle motion and potential energy of particle displacement in the wave motion.

If the particle motions are excessive then there will be damage to buildings. Therefore, the first goal is to design blasts that generate vibrations below that at which damage may occur. Often, a second goal is to design blasts that create even less vibration in order to reduce human response to the shots.

In the late 1960's considerable research was performed in the field of blasting generated vibration. In 1971 the Bureau of Mines published a report which reviewed their data and the data of others (9). The results of this report have been widely used in blast vibration regulation and control. The USBM found that the best descriptor of potential damage was the peak

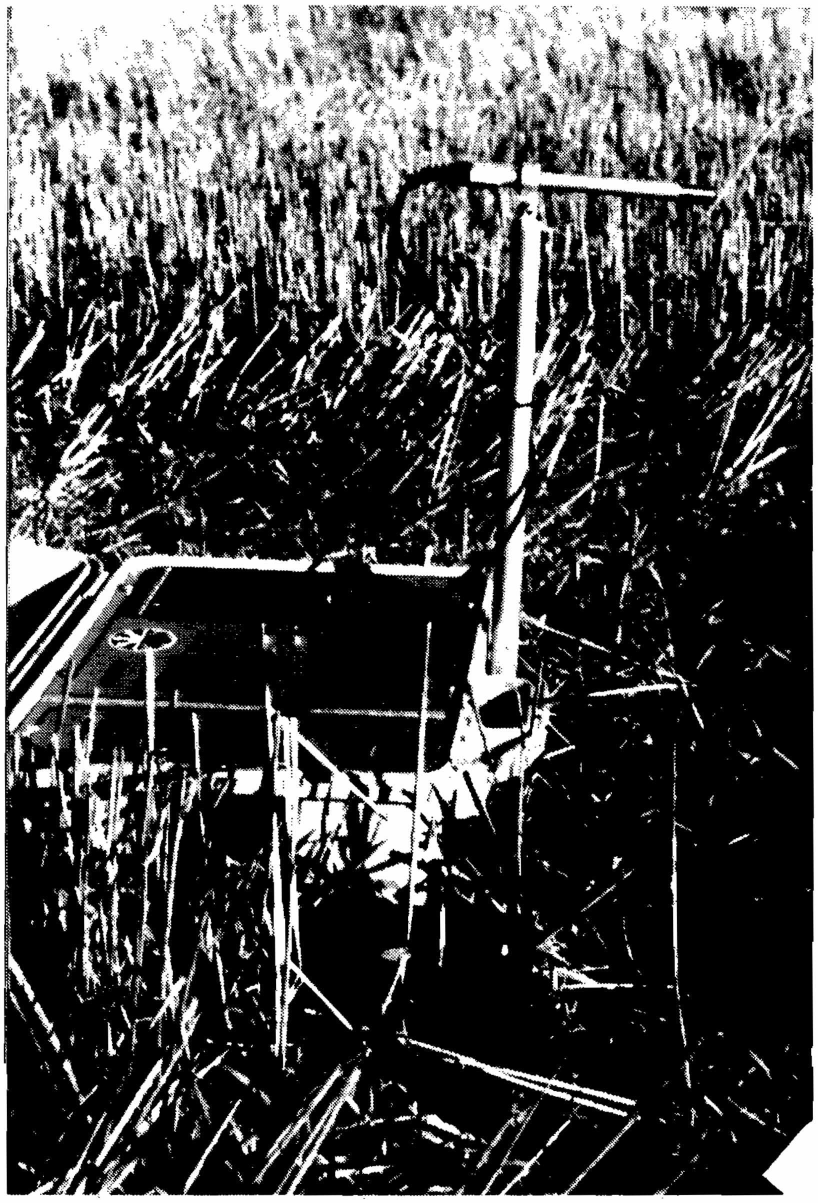
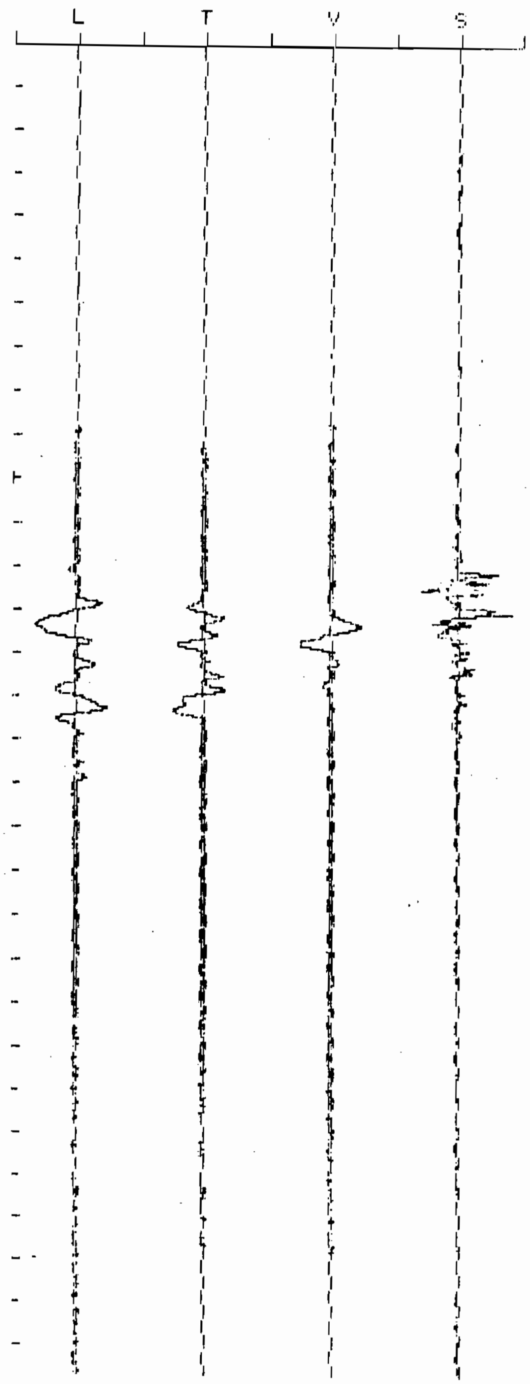


FIGURE 7-1: VIEW OF SSU-1000DK SEISMOGRAPH USED TO MONITOR THE TEST BLASTS

** SAFEGUARD SEISMIC UNIT 1000D **
SEP 12/88 12:41:53
INSTRUMENT # SSU 1609-DK
EVENT # 032
COPY # 2
CLIENT:BAUER CALDER WORKMAN
OPERATION:AML BLASTING WHITE#3 SITE
SSU LOCATION:NORTH OF SITE
DISTANCE TO BLAST (ft): 350
OPERATOR:LYALL WORKMAN
SEIS TRIG LVL (iPs): .02
SOUND TRIG LEVEL (dB): 125
RECORD TIME (sec): 5
PRINT (Y/N) Y
DISK EVENTS (Y/N) N

GRAPHICAL RECORD
****
TIME = .333 sec/in
SOUND= 0.0028 Psi/div
L,T,V= .25 iPs/div



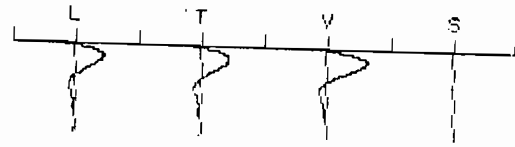
SEP 12/88 12:41:53

	L	T	V
PPV(in/s)	0.16	0.12	0.13
PDK(in)	0.00324	0.00217	0.00166
PPA(g)	0.041	0.062	0.041
Frq(hz)	7.6	10.0	11.6

PPPV(in/s) 0.20
PEAK SOUND 118 dB 0.00237 Psi
** A/D CALIBRATED OK **

CALIBRATION GRAPH

TIME = .384 sec/in
SOUND= 0.028 Psi/div
L,T,V= 1 iPs/div



SHAKETABLE CALIBRATED ON JUL 29/88
BY:PHILIP R BERGER & ASSOCIATES,INC.
BOX 779, WARRENDALE, PA. 15095
TEL:412-776-3600

SCANNING

FIGURE 7-2: EXAMPLE OF BLAST VIBRATION AND AIRBLAST RECORD PRODUCED BY THE SSU-1000 DK SEISMOGRAPH particle velocity, that is the velocity at which a particle in the ground mass oscillated about its point of rest. It was concluded that a peak particle velocity in any one of three mutually orthogonal modes of greater than 2.0 inches per second represented the onset of plaster cracking in residences and the goal became to keep all blasting vibrations, at the point of interest, below 2.0 inches per second.

The blast vibrations, as measured by the particle velocity, could be related to a scaled distance factor. The scaled distance was the distance from the blast to the measuring point divided by the square root of explosive weight. The explosive weight was the total weight firing on an individual delay period.

Having obtained the peak particle velocity and calculated the scaled distance logarithmic graphs of particle velocity versus scaled distance could be prepared. Such charts invariably showed considerable data scatter so regression techniques were usually employed to determine a best fit line representing the data. This line would have one half the data above and below the line. However, one does not wish to perform designs which have excessive vibration fifty percent of the time. Therefore, an upper limit line could be drawn on the graph, parallel to the best fit line but above all the data upon which the chart was based. Then, for a given scaled distance all vibration readings were expected to fall below the level represented by the upper limit line.

As an example figure 7-3 is a graph of typical vibration levels. This graph consists of data from numerous mines and quarries. The maximum weight per delay represented is 60,000 pounds and the maximum weight of explosive in a blast is 700,000 pounds. The upper limit line represents some 1,500 data points, some of which are shown on the graph.

The graph shows the scatter that typically occurs. It also shows the upper limit line used for design. The approach to vibration control then has typically been to select a peak particle velocity that is not to be exceeded and draw a line horizontally across to the upper limit line. From this point of intersection a line is drawn vertically down to the horizontal axis which gives the scaled distance, $d/W^{1/2}$, required for the chosen maximum allowable particle velocity. Once the distance from the blast to the point of interest has been established the permissible explosive weight per delay can be calculated. The blast is then designed to meet the required criteria. The process is shown on figure 7-3.

The chosen peak particle velocity may be the result of regulations governing blast vibration or a more conservative value may have been selected if human response is a concern. If AML blasting is being conducted near to residences the latter is

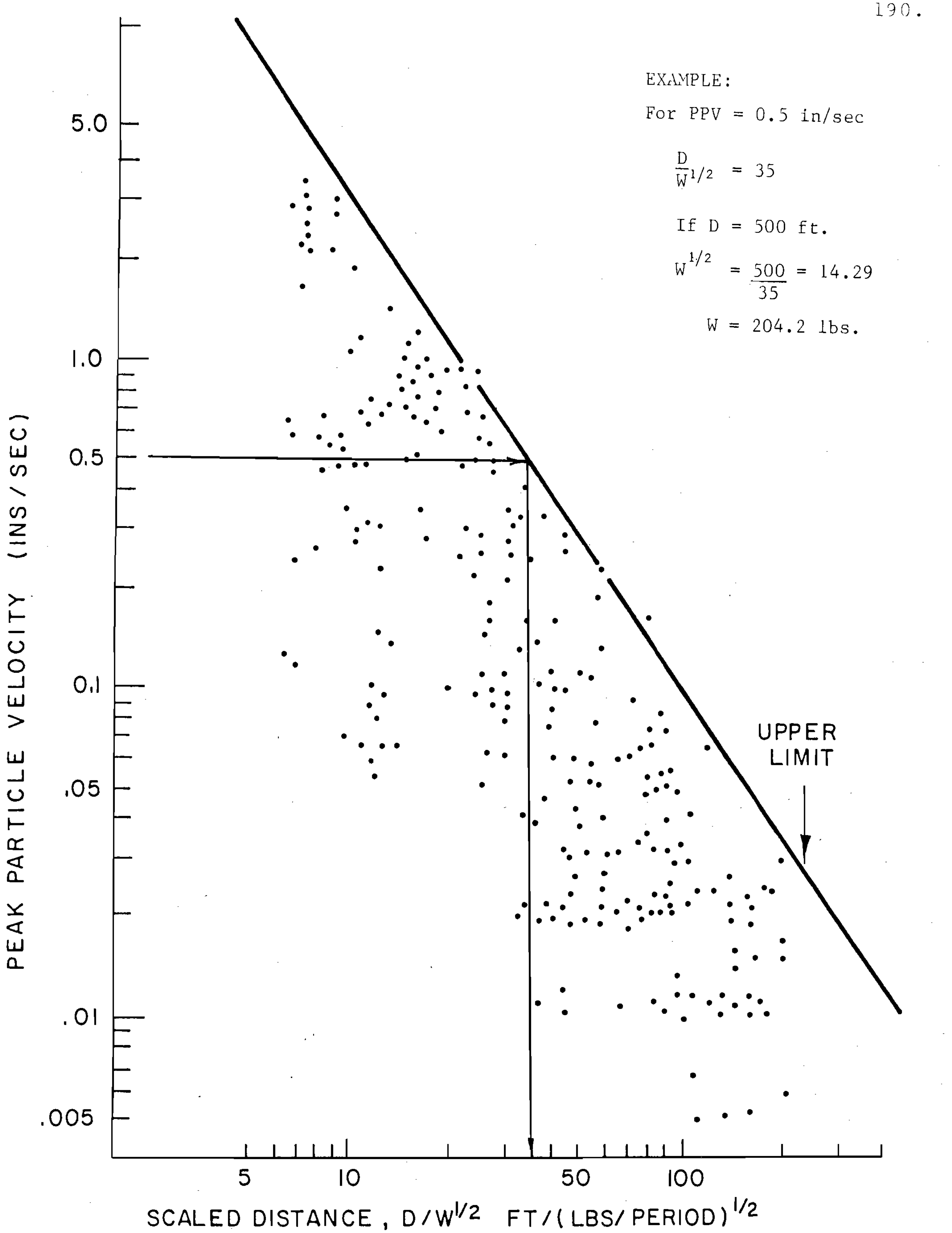


FIGURE 7-3: TYPICAL VIBRATION DATA FROM MULTIPERIOD DELAY BLASTS IN OPEN PITS AND ROCK STRIP MINES.

likely to be the governing criteria. People respond to blasting events at particle velocities considerably less than those that are usually likely to damage structures.

In the late 1970's the Bureau of Mines performed additional extensive research on blasting vibrations. The results of this study were published by Siskind et al in 1980 (10). The role of frequency, which had been largely ignored in previous work was now recognized.

It was found that the response of a building to blast induced vibration could be quite different depending on the frequency of the oscillations. In general, it was found that blasts with frequencies near to the natural response frequency of the structure created a greater response than did those blasts generating vibrations of a frequency significantly different than the natural frequency of the residence.

The frequencies of structural corners of residences have been found to typically vary from 4 to 12 Hz. Two story houses have the lowest frequencies while the natural frequencies of 1 and 1½-story structures are usually higher and similar.

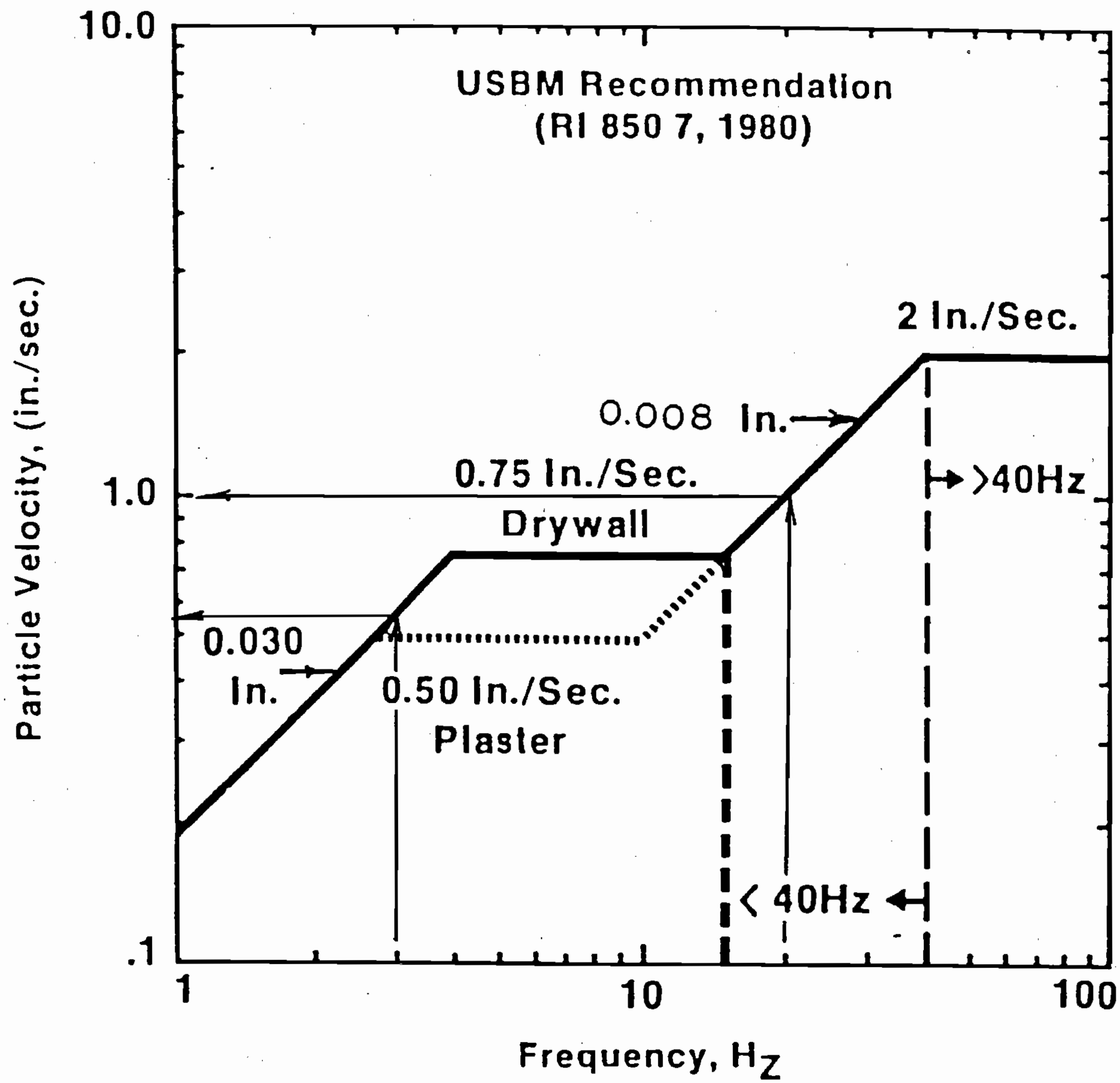
Midwall natural frequencies are generally higher. These tend to range from 11 to 25 Hz. Midwall motions are normal to the wall surface whereas corner motions are a shearing action.

This study, which correlated the roles of peak particle velocity and frequency led the Bureau of Mines to make new recommendations for avoiding damage due to blasting. It was recommended that for blast vibration frequencies below 40 Hz that the peak particle velocity be kept below 0.75 ins/sec. If the houses in the vicinity of the blast are older and have plaster on lathe construction then the recommended maximum particle velocity was 0.50 in/sec.

For those cases where the frequency of the vibrations is greater than 40 Hz a maximum particle velocity of 2.0 in/sec is considered safe. In all cases the particle velocity is still considered to best characterize the damage potential.

It has been concluded, therefore, that the particle velocity is the best descriptor for regulating the damage potential from blasting. The Bureau of Mines, in the 1980 publication, recommended that if one did not wish to monitor each blast with a seismograph then the scaled distance should be kept above 70 $ft/lb^{\frac{1}{2}}$ which would lead to vibration levels commonly between 0.08 and 0.15 in/sec.

The Bureau of Mines also provided an alternative blasting level criteria using both the measured structure amplifications and the damage summaries they had recorded. This criteria is shown in figure 7-4. This is a smoother set of criteria but



EXAMPLE:

1. $V_o = D_o 2\pi f$

   $D_o = .008$ ins.

   $f = 20$ Hz

   $V_o = .008 \times 2 \times \pi \times 20$

   $= 1.00$ in/sec

2. $V_o = D_o 2\pi f$

   $D_o = 0.030$ ins

   $f = 3$ Hz

   $V_o = .030 \times 2 \times \pi \times 3$

   $= 0.565$ in/sec

FIGURE 7-4: USBM RECOMMENDATION FOR SAFE BLASTING requires more measurement capability involving both velocity and displacement. From the figure one can see that above 40 Hz a particle velocity of 2.0 ins/sec is considered acceptable for all cases. Below 40 Hz the particle velocity decreases at a rate which gives a constant peak displacement of 0.008 inches. At frequencies which correspond to a particle velocity of 0.75 in/sec (drywall) or 0.50 in/sec (plaster on lathe) the constant particle velocity is again appropriate. When the frequencies are very low ($\leq$4Hz) then an ultimate peak displacement of 0.030 inches is recommended.

In 1983 the Office of Surface Mining published final regulations concerning the control of ground vibrations and airblast in coal mines. The OSM regulations provided three methods by which the regulations could be complied with. These are:

- Scaled Distance Criterion
- Limiting Particle Velocity Criterion
- Chart of Maximum Particle Velocity versus Frequency Criterion

The scaled distance method requires that the scaled distance be kept at or above a certain level. The minimum scaled distance varies with distance from the shot. This reflects the fact that at greater distances the frequencies are usually lower and more in line with the natural frequencies of residences. Therefore, the maximum allowable particle velocities are lower. The criteria are summarized in table 7-1. No monitoring by seismograph is required if the scaled distance criterion is followed.

TABLE 7-1: PERMISSIBLE SCALED DISTANCES AT DIFFERENT DISTANCES FROM THE BLAST SITE TO THE STRUCTURE

Distance From Blast (feet)	Permissible Scaled Distance ($ft/lb^{1/2}$)
0 - 300	50
301-5000	55
5001 and greater	65

The limiting particle velocity criterion requires that each shot be monitored with a seismograph. Provided the peak particle velocities do not exceed those specified the regulations are deemed to be met. The maximum particle velocity allowed decreases as the distance from the shot increases. The reason is that lower frequencies usually predominate at longer distances. As these frequencies are nearer to the natural frequencies of structures damage potential increases, so the particle velocity is decreased. The criterion is listed in table 7-2.

TABLE 7-2: MAXIMUM PARTICLE VELOCITIES PERMITTED AS A FUNCTION OF DISTANCE FROM THE BLAST SITE

Distance in Feet	Maximum Particle Velocity, in./sec.
0-300	1.25
301-5000	1.00
5001 and greater	0.75

The method avoids the restrictive blast designs that attend the conservative scaled distances of the first method. More flexibility in blast design may be possible and the shooting of larger blasts may be permissible. Over time a plot of peak particle velocity versus scaled distance should be prepared. An upper limit line should be established and this can be used for further design.

The last method allows the operator to use variable particle velocity limits based on frequency. Figure 7-5 is the particle velocity versus blast vibration frequency chart used. This chart is not the same as the USBM graph shown in figure 7-4. For Example the OSM chart shows 2.0 in/sec as acceptable above 30 Hz. Also the 0.50 in/sec line for plaster on lathe is not included.

In this method the vibration trace is analyzed to determine the predominant frequency and the particle velocity associated with that frequency. The particle velocities must be within the specified limits for the given frequency. This method requires the most sophisticated equipment and the greatest level of skill for analysis.

These criteria are primarily designed to avoid structure damage due to blasting. However, it is often human response that is the limiting factor. Vibration levels that can be felt and perceived as objectionable are usually much less than those required to cause damage to buildings. Therefore, it is often necessary to design for vibration levels that will minimize human response.

The perception of blasting by people is a subjective matter. It is difficult to determine exactly what level of blasting vibration will cause a citizen to complain. Figure 7-6 superimposes information about human response on the upper limit line shown earlier. One can see that vibration becomes perceptible at 0.10 in/sec and unpleasant at 0.60 in/sec. Somewhere in this range a significant level of complaints is likely to begin.

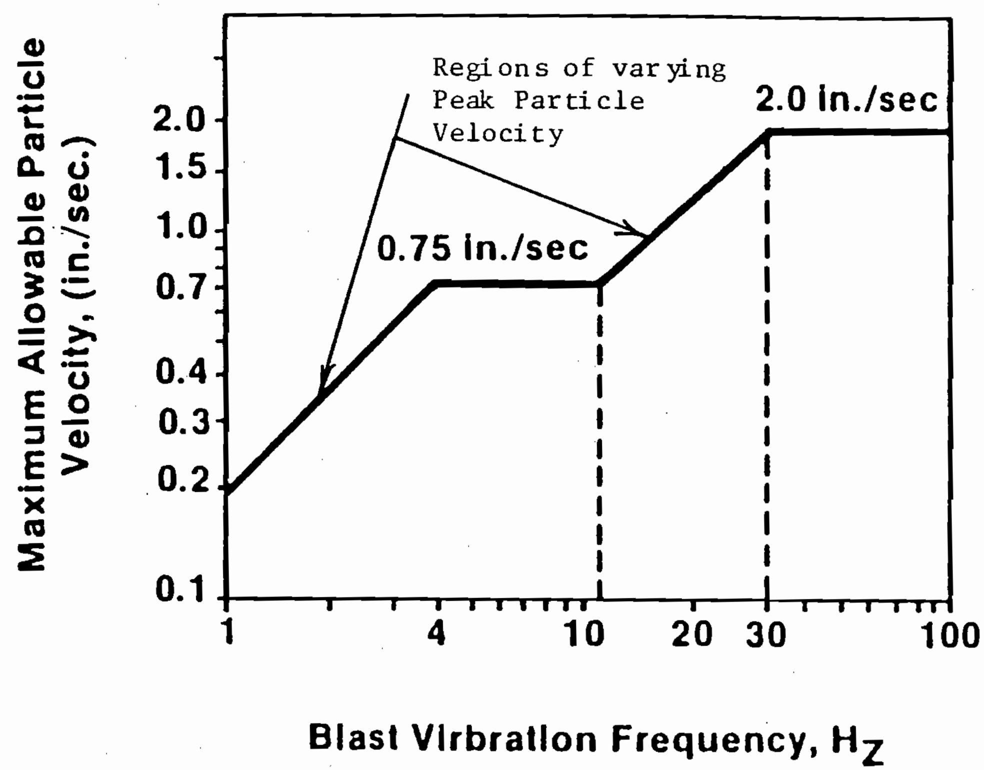


FIGURE 7-5: OSM REGULATION

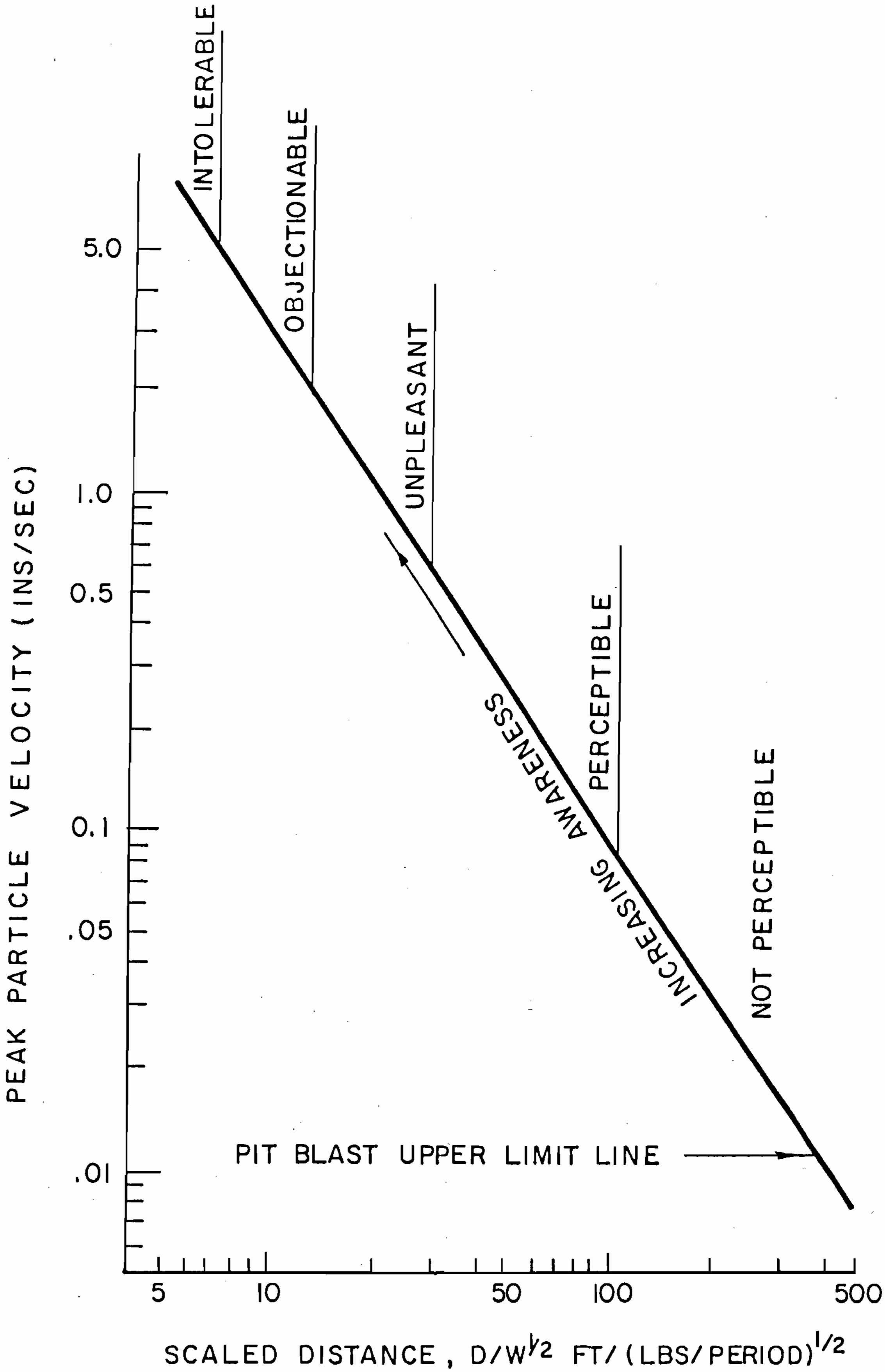


FIGURE 7-6: ANTICIPATED HUMAN RESPONSE AT DIFFERENT SCALED DISTANCES FROM PIT BLASTS.

The Bureau of Mines study (10) concluded that a vibration level of 0.50 in/sec ought to be tolerable to 95 per cent of those who perceive it as distinctly perceptible. It was further concluded that human response is dependent on duration as well as magnitude. For people at home a 0.50 in/sec level could result in a complaint rate as high as 30 per cent for a variety of reasons.

From the various studies one concludes that, if human response is an important factor, a vibration level less than 0.50 in/sec should be designed for. Where not overly restrictive for blast designs and performance a particle velocity of 0.10 in/sec ought to be aimed for. Also, the blasting event duration could be kept as short as possible. Blasts of total duration less than 1 second will reduce complaints. These criteria combined will negate almost all complaints associated with blast vibration.

There are a number of ways by which ground vibrations may be reduced. These are listed below:

1. Reduce the weight of explosive per delay period

2. Reduce the confinement on the charge:

   (a) Reduce burden and spacing
   (b) Reduce stemming (careful about airblast)
   (c) Reduce subgrade drilling
   (d) Reduce hole depth (bench height)
   (e) Provide maximum relief (two free faces)

3. Blast progression should be away from the structure

4. Optimize the millisecond delay timing

5. Use millisecond delays from the same manufacturer and lot number

6. Keep blast duration below one second

For AML blasting points 1, 2(b), 3, 5 and 6 are likely to be the most important.

7.2.2 Blast Induced Airblast

The other factor to be considered is airblast. If one is not careful the airblast generated by the shot may create more problems than the ground vibration.

Airblast is the impulsive sound generated by the blast. It can result from poor confinement of the charge, venting through fractures, the piston effect of displaced rock off the face and top of the charge and unconfined delays and detonating cord on surface.

The magnitude of the airblast experienced at a given point can be quite dependent on the weather conditions. Figure 7-7 illustrates situations which are favorable and unfavorable regarding airblast. Temperature inversions and wind in the direction of the structure are primary causes of magnified airblast.

Damage from airblast is most likely to occur in windows. Loose or poorly installed panes are the most likely to be cracked or broken. Window damage is certainly possible at a level of 0.030 lb/in^2 (142dB). Damage to doors and other structural damage generally requires greater airblast levels. In most situations the primary problem with airblast may be that people find it objectionable. This usually occurs because an individual is startled by the event or frightened by the rattling of windows and doors. The level of airblast at which considerable complaints will be generated is quite subjective. However, there is no question that, as the noise level increases above 125 dB the potential for complaints will begin to rise.

In 1980 Siskind et al (15) of the Bureau of Mines produced an extensive study of airblast and its effects. The Report of Investigation concluded that the following represented safe maximum levels.

TABLE 7-3: SAFE AIRBLAST LEVELS FOR VARIOUS MEASURING SYSTEMS

System	Safe Maximum Level dB
0.1 Hz high-pass system	134
2 Hz high-pass system	133
5 or 6 Hz high-pass system	129
C-slow (events not exceeding 2 secs. duration)	105

The best system to use is one which is linear to 2Hz. Where there are many large plate glass windows one may wish to reduce these maximum levels.

Siskind et al also provided minimum cube root scaled distances from the blast to the structure which can be used if there is no monitoring of the shots. These criteria are given in table 7-4.

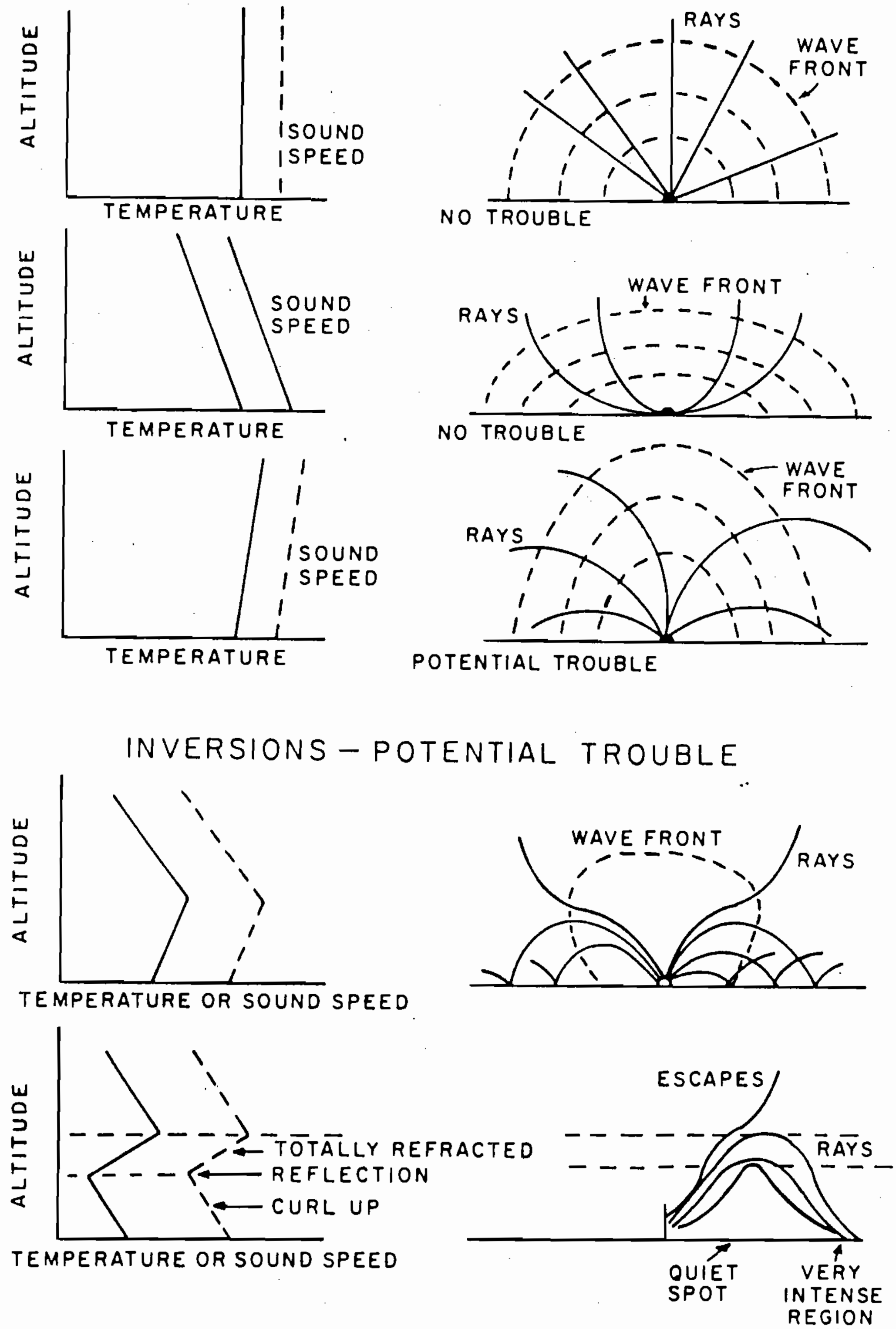


FIGURE 7-7: EFFECT OF ALTITUDE TEMPERATURE PROFILES ON AIR BLAST PROPAGATION.

TABLE 7-4: MINIMUM SCALED DISTANCES TO BE USED IN THE ABSENCE OF AIRBLAST MONITORING

Type of Blast	Minimum Scaled Distance $ft/lb^{1/3}$
Coal highwall	180
Coal parting	500
Quarries and mines	250
Construction and excavation	500
Unconfined blasting	800

These values are conservative. In most cases monitoring the blasts will lead to greater flexibility. The scaled distances generally reflect the degree of confinement with less well confined blasts having a greater scaled distance.

The following is a list of common airblast sources:

1. Detonating cord trunklines and surface delays.

2. Lack of proper stemming materials.

3. Inadequate stemming height.

4. Overdug and overloaded front row holes. Collars or burdens near the crest too small due to backbreak.

5. Delay sequence (piston effect of displacing rock mass.)

6. Atmospheric conditions such as temperature inversions or wind in the direction of concern.

7. Secondary blasting.

In AML work the most likely sources of airblast will be 1, 2, 3 and 6.

There are ways by which airblast can be minimized. These are listed below:

1. Insure adequate confinement of the charge at the collar and the front row.

2. Cover surface detonating cord and delay caps with drill cuttings when blasting close to housing.

3. Examine weather conditions. If at all possible do not blast on days when there are inversions, heavy clouds or wind in the direction of the structure.

4. Avoid overdigging the previous blast and design blasts with adequate relief to avoid excessive backbreak.

5. Use +1/2 - 3/4-inch crushed stone for stemming if it is available.

6. Cover toe shots (increases confinement).

7. Design the front row initiation carefully to avoid reinforcing the piston effect of the rock moving off the face on the air in front of the blast. Avoid row-on-row blasting.

8. Minimize secondary blasting which is almost always noisy.

9. Reduce the charge weight per delay.

In AML blasting items 1, 2, 3 and 9 will be most helpful in the minimizing the airblast associated with the shots.

7.3 TEST RESULTS

7.3.1 Beulah Test Site

At Beulah vibration and airblast readings were obtained for thirteen shots. Scaled distances ranged from 31.4 to 63.3 $ft/lb^{1/2}$. Peak particle velocities ranged from 0.02 to 0.45 ins/sec and the airblast readings varied from 108 to 128 dB. Table 7-5 lists the information obtained from each shot and the scaled distances as well.

From this table a plot of peak particle velocity versus scaled distance was prepared. A regression analysis was performed to give the best fit line for this data which is shown on the graph in figure 7-8. An upper limit line has also been drawn which encompasses all the data. An additional line has been drawn which is an upper limit line enclosing all the data except for one point.

The airblast, measured as psi, has been plotted against the cube root scaled distance. This plot is shown in figure 7-9. A best fit line has been calculated and an upper limit line is also shown. The data is quite closely grouped in this case.

7.3.2 White Test Site

At the White Test Site vibration and airblast readings were obtained for each of the eight principle shots. The information obtained is listed in table 7-6. Scaled distances varied from 15.7 to 67.1 $ft/lb^{1/2}$ and the peak particle velocities from 0.06 to 0.93 ins/sec. The airblast readings ranged from 112 to 134

TABLE 7-5: DATA OBTAINED FROM SEISMOGRAPH MONITORING OF BLASTS AT THE BEULAH TEST SITE

Blast	Date	Time	Particle Velocity L	Particle Velocity T	Particle Velocity V	Particle Velocity Peak	Frequency L	Frequency T	Frequency V	Airblast dB	Airblast psi	wt./delay lbs.	Distance ft.	Scaled Dist. ft/lbs-1/2
VO-3	8-8-88	12:15:19	0.02	0.01	0.01	0.02	170.7	170.7	170.7	108	0.00072	67.5	500	60.9
VO-5	8-16-88	15:15:32	0.32	0.02	0.45	0.45	4.7	256.0	5.0	123	0.00418	72.8	300	35.2
VO-4B	8-17-88	16:00:51	0.06	0.01	0.02	0.06	15.5	102.4	39.4	123	0.00402	81.0	322	35.8
VO-4A#2	8-17-88	16:25:34	0.03	0.01	0.01	0.03	21.3	128.0	128.0	111	0.00105	31.2	313	56.0
VO-9	8-18-88	16:18:36	0.04	0.04	0.02	0.04	32.0	27.0	256.0	126	0.00578	98.0	311	31.4
VO-7&8N	8-19-88	16:20:41	0.02	0.01	0.02	0.02	170.7	128.0	128.0	111	0.00105	84.0	500	54.6
VO-6	8-22-88	16:10:43	0.01	0.01	0.02	0.02	170.7	73.1	170.7	111	0.00105	62.4	400	50.6
VO-7&8S	8-23-88	14:50:16	0.01	0.01	0.02	0.02	128.0	128.0	170.7	108	0.00072	62.4	500	63.3
VO-9#2	8-24-88	18:13:11	0.02	0.04	0.02	0.04	170.7	13.8	170.7	117	0.0021	62.4	450	57.0
Vo7&8N&7S#2	8-25-88	11:32:20	0.01	0.02	0.01	0.02	170.7	256.0	170.7	117	0.00198	65.0	500	62.0
VO-11	8-30-88	12:44:48	0.04	0.05	0.01	0.05	16.5	17.7	57.0	117	0.00215	66.0	300	36.9
VO-12	8-31-88	12:39:51	0.02	0.01	0.01	0.02	170.7	170.7	85.3	128	0.00688	62.4	300	38.0
VO-13	8-31-88	16:53:55	0.02	0.04	0.02	0.04	170.7	13.5	170.7	113	0.00132	84.0	450	49.1

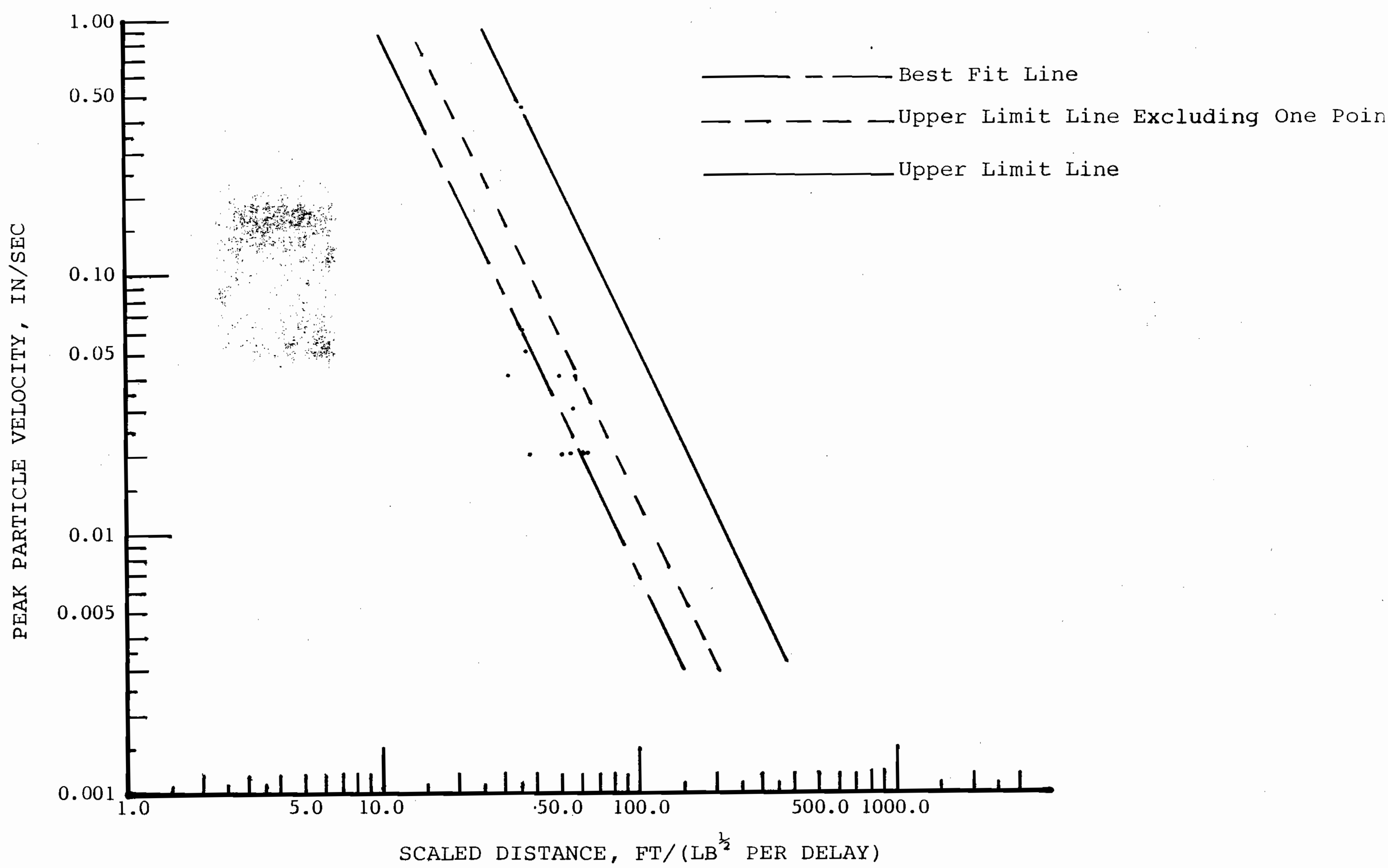


FIGURE 7-8: PEAK PARTICLE VELOCITY AS A FUNCTION OF THE SCALED DISTANCE FOR THE BLASTS AT THE BEULAH SITE

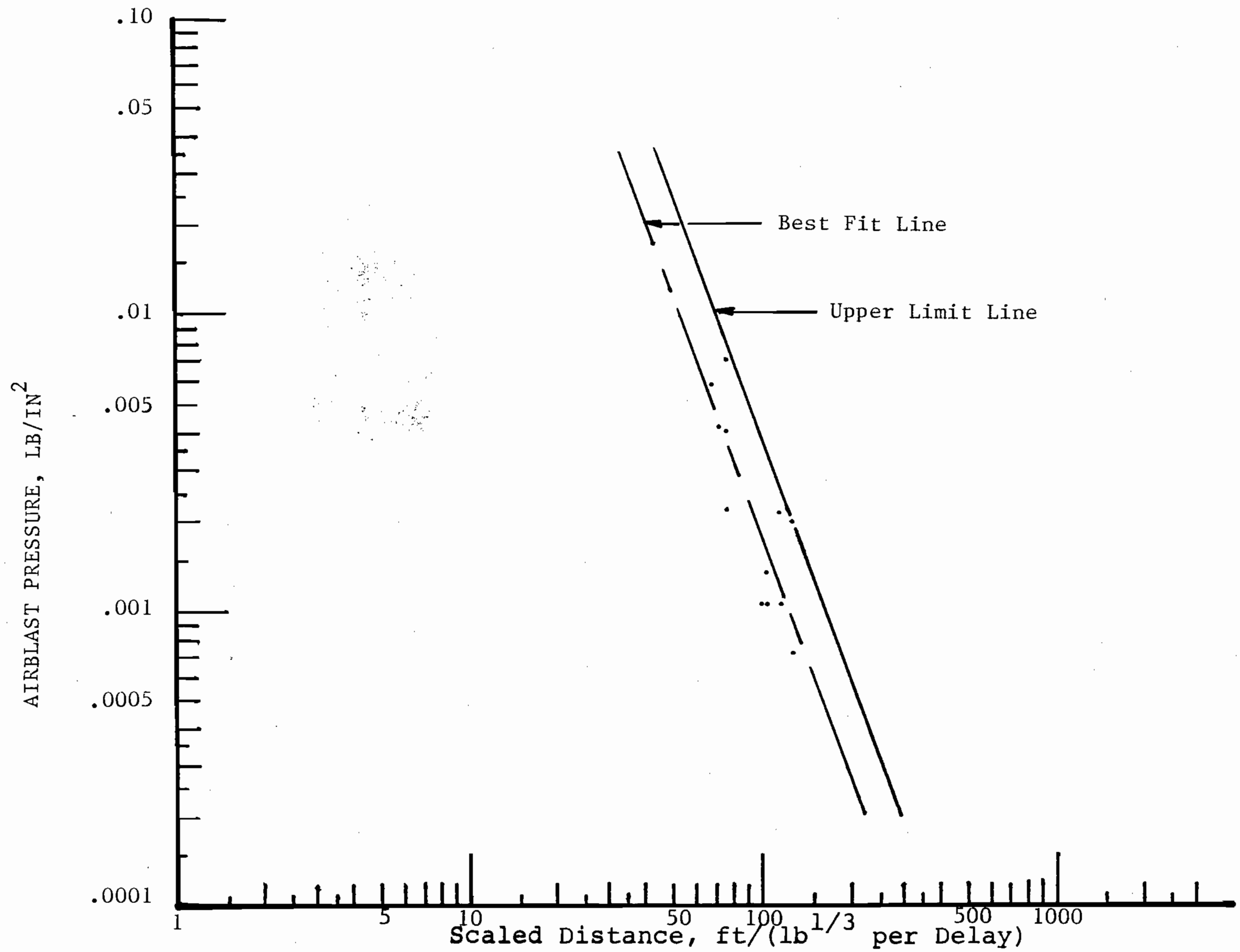


FIGURE 7-9: AIRBLAST PRESSURE AS A FUNCTION OF SCALED DISTANCE FOR BLASTS AT THE BEULAH TEST SITE

TABLE 7-6: DATA OBTAINED FROM SEISMOGRAPH MONITORING OF BLASTS AT THE WHITE TEST SITE

Blast	Date	Time	Particle Velocity L	T	V	Peak	Frequency L	T	V	Airblast dB	psi	wt./delay lbs.	Distance ft.	Scaled Dist. ft/lbs-1/2
B1	9-12-88	12:41:53	0.16	0.12	0.13	0.16	7.6	10.0	11.6	118	0.00237	124.8	350	31.3
B2	9-13-88	15:56:48	0.10	0.03	0.10	0.10	13.8	21.3	12.2	119	0.00260	70.0	350	41.8
B3	9-15-88	16:18:54	0.06	0.04	0.02	0.06	12.2	25.6	27.0	128	0.00732	84.0	350	38.2
B4	9-16-88	15:09:59	0.01	0.03	0.10	0.10	73.1	28.4	16.5	112	0.00116	45.0	450	67.1
B5	9-17-88	12:49:39	0.62	0.20	0.10	0.62	16.0	14.2	20.0	119	0.00260	61.0	270	34.6
B6	9-19-88	14:51:07	0.46	0.30	0.16	0.46	11.1	8.5	4.7	132	0.01128	294.0	270	15.7
B7	9-22-88	13:49:59	0.93	0.56	0.45	0.93	8.8	13.8	57.0	134	0.01430	208.0	270	18.7
B8	9-24-88	17:23:42	0.73	0.54	0.37	0.73	11.6	9.7	8.0	126	0.00600	266.0	270	16.6

dB. The weight per delay and the scaled distance varied more than at Beulah because of the pattern shots detonated at the White test area.

From this data a plot of peak particle velocity versus scaled distance was prepared. A best fit line was calculated for the data. An upper limit line, enclosing all the data, is also shown. The graph is shown in figure 7-10.

The airblast in decibles has been plotted as a function of the cube root scaled distance. A best fit line has been calculated and is shown on the graph in figure 7-11. An upper limit line has also been drawn on the chart.

7.3.3 Combined Data

All of the peak particle velocity data from this study has been plotted versus scaled distance on one graph. This chart is shown in figure 7-12. There are a total of 21 data points on this graph. A best fit line for the data has been calculated and shown on the chart. The upper limit line has also been drawn on this plot.

A graph of airblast overpressure versus cube root scaled distance has also been drawn and is presented in figure 7-13. This graph also has 21 data points. The best fit line has been calculated and drawn on the chart as has the upper limit line. The data is quite tightly grouped around the best fit line.

To further extend the database a combined graph of peak particle velocity versus square root scaled distance has been drawn that includes data from the previous study at Beulah. The graph is displayed in figure 7-14. There are 38 data points on this graph. The best fit line representing the data has been drawn on this chart. The upper limit line is also shown.

The same approach is taken for airblast. Thirty-four data points, including those from the previous study, have been plotted in figure 7-15. As before the best fit line has been determined. The upper limit line is also shown on this graph.

7.4 DISCUSSION OF RESULTS

7.4.1 Ground Vibration Results

7.4.1.1 Beulah Test Site

The vibration data obtained at the Beulah site and listed in table 7-5 indicates that as the peak particle velocity increases the frequency of the vibration decreases. Almost always the mode exhibiting the highest particle velocity also had the lowest principle frequency. The frequencies associated with the peak particle velocity were typically in the range of

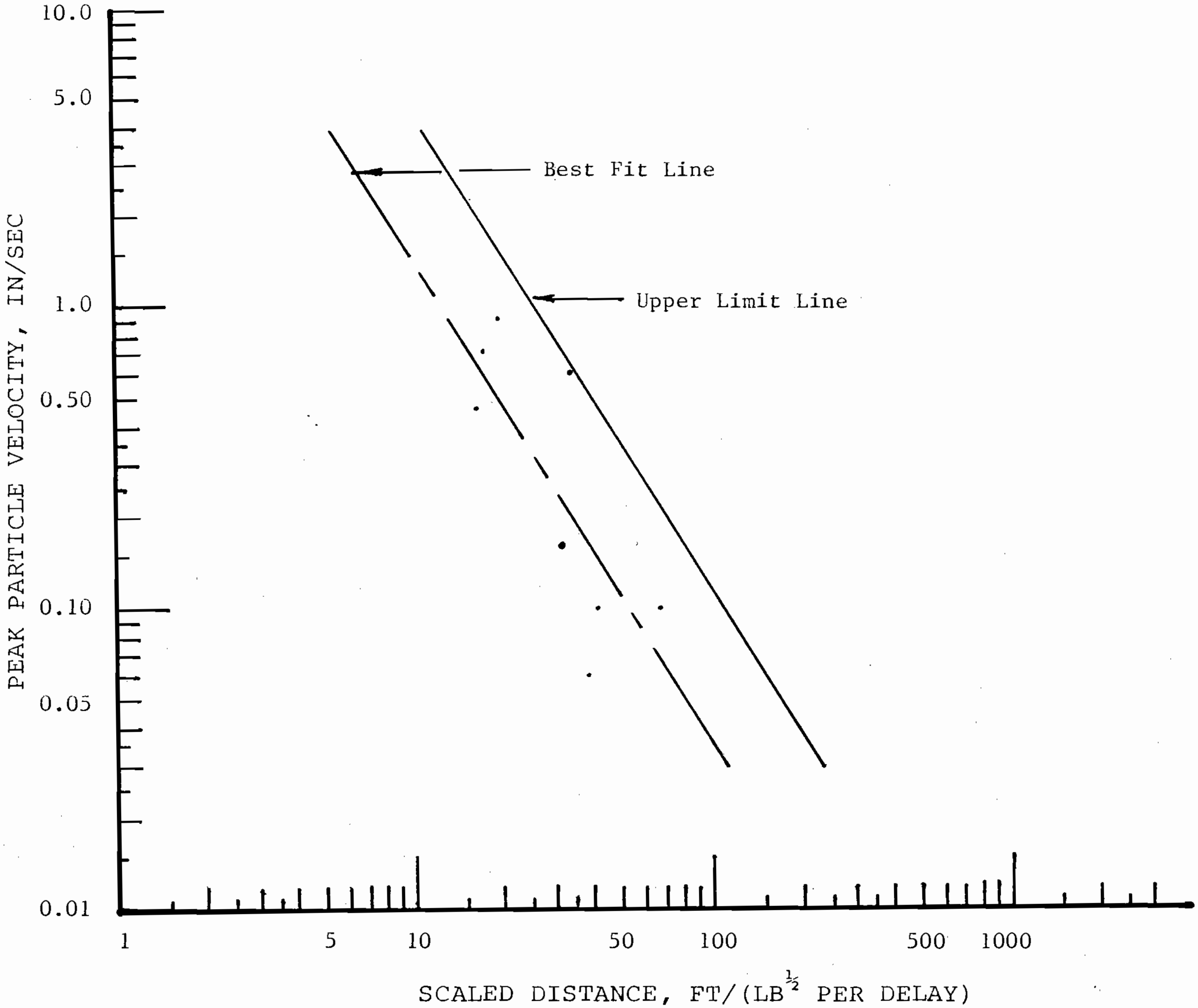


FIGURE 7-10: PEAK PARTICLE VELOCITY AS A FUNCTION OF SCALED DISTANCE FOR THE BLASTS AT THE WHITE TEST SITE

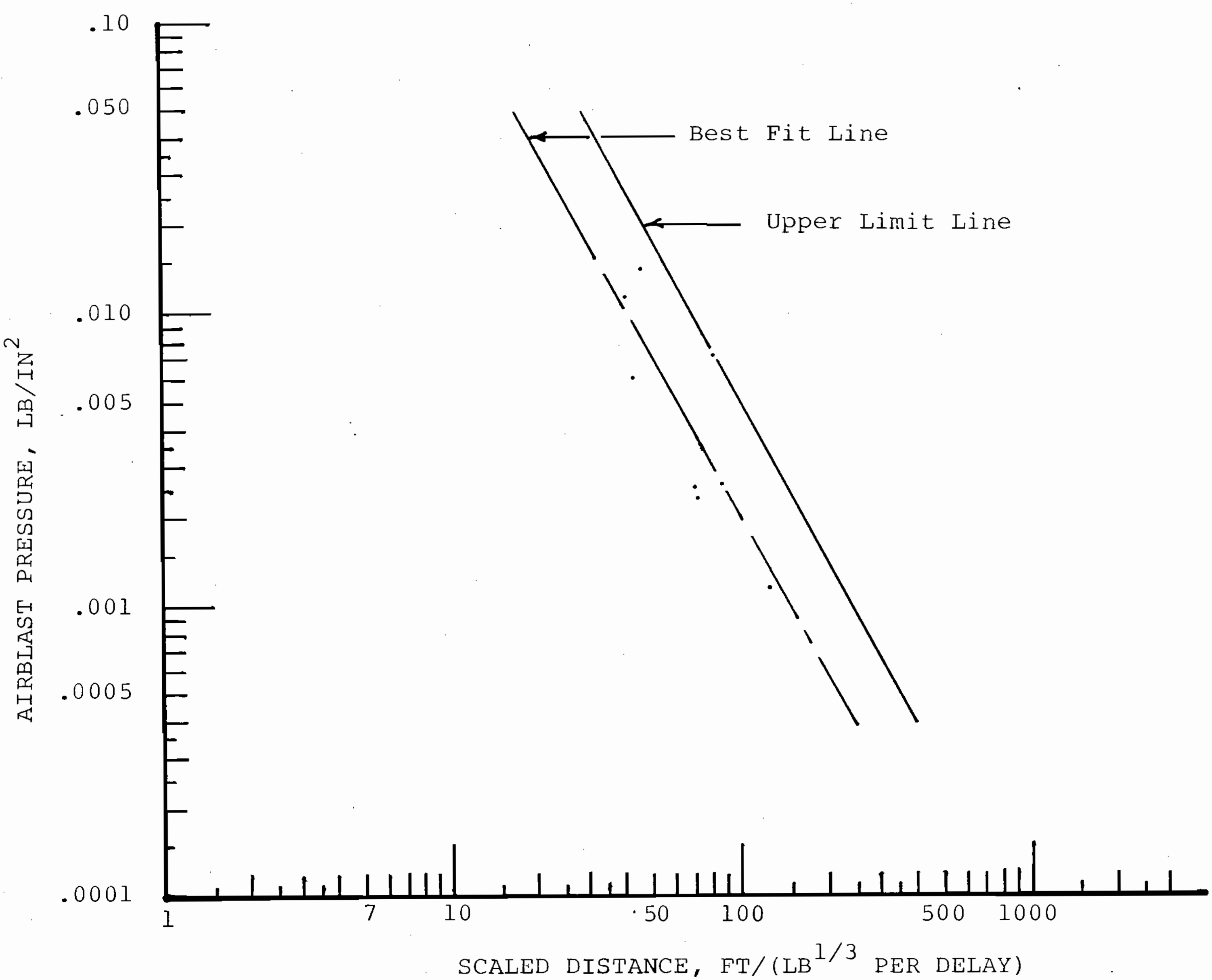


FIGURE 7-11: AIRBLAST PRESSURE AS A FUNCTION OF SCALED DISTANCE FOR BLASTS AT THE WHITE TEST SITE

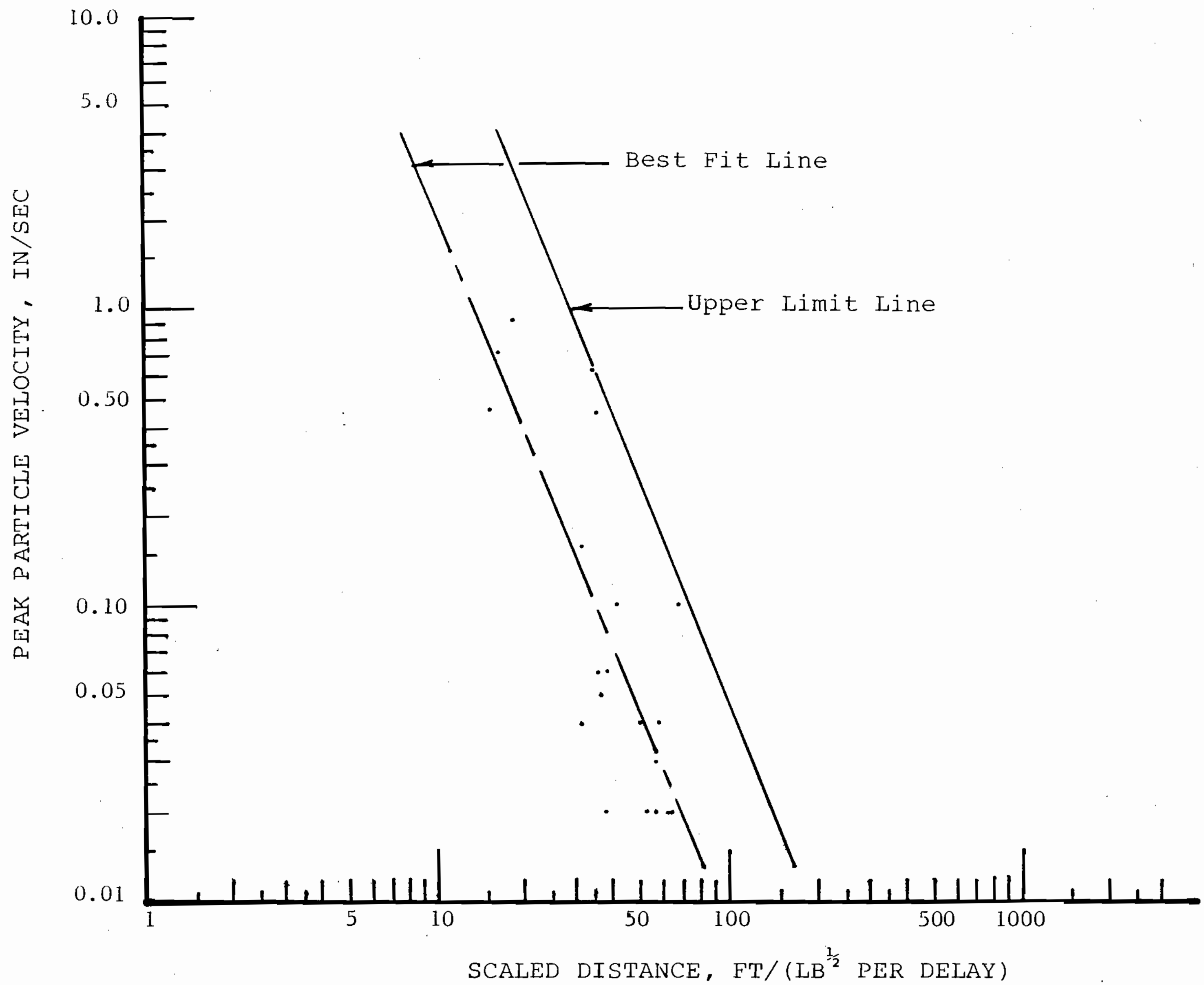


FIGURE 7-12: PEAK PARTICLE VELOCITY VERSUS SCALED DISTANCE FOR ALL DATA FROM BOTH SITES

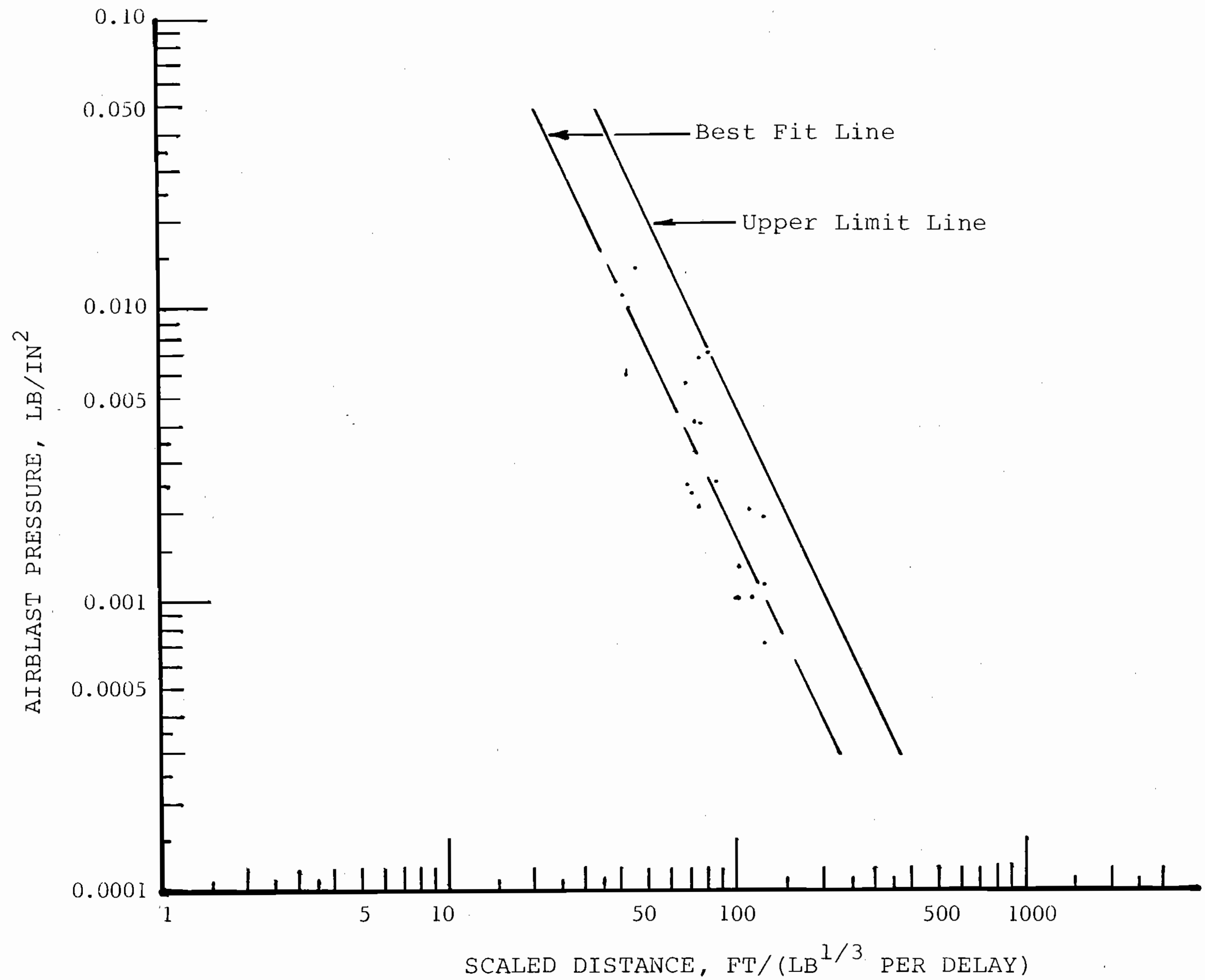


FIGURE 7-13: AIRBLAST PRESSURE VERSUS SCALED DISTANCE FOR ALL DATA FROM BLASTS AT BOTH TEST SITES

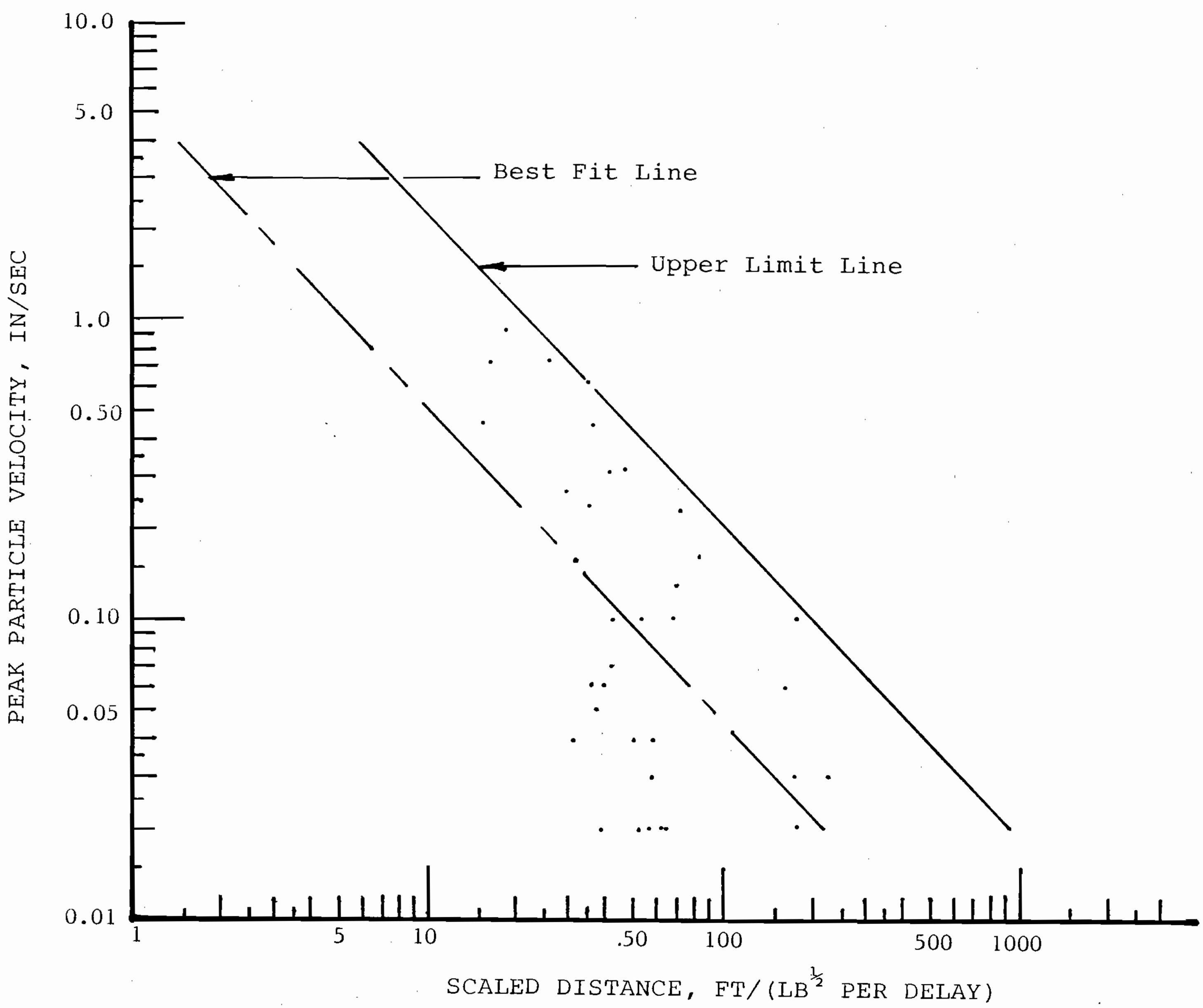


FIGURE 7-14: PEAK PARTICLE VELOCITY VERSUS SCALED DISTANCE FOR ALL DATA INCLUDING READINGS OBTAINED IN PRIOR STUDIES

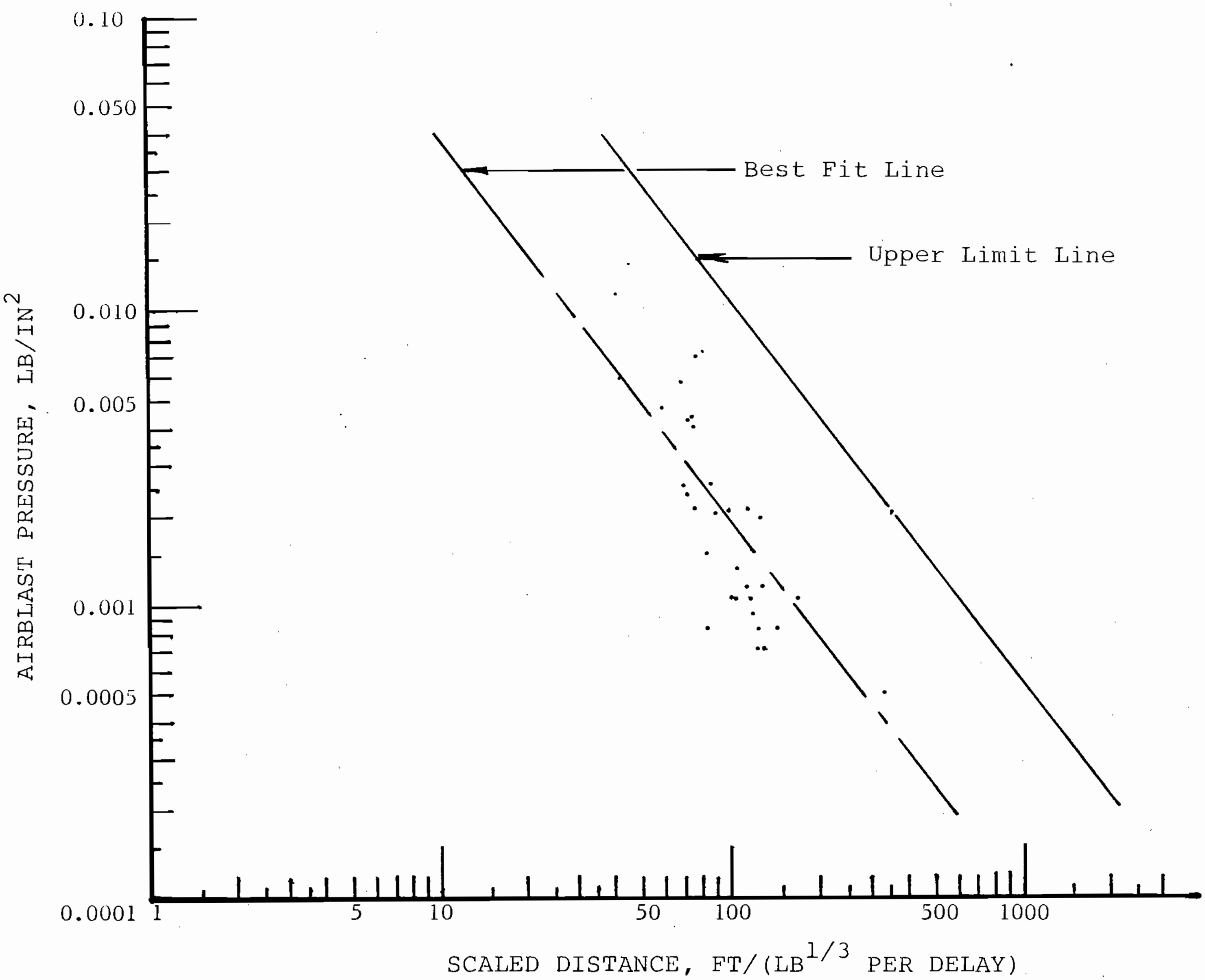


FIGURE 7-15: AIRBLAST PRESSURE AS A FUNCTION OF SCALED DISTANCE FOR ALL DATA INCLUDING INFORMATION FROM PREVIOUS STUDY

response frequencies associated with residences but at the high end of that range, which is a favorable result. When the peak particle velocity was 0.02 in/sec or less the frequencies were considerably higher than the typical response frequencies.

In general the vibration levels experienced at Beulah were modest. Figure 7-8 shows results not dissimilar from the graph of a large number of mining shots shown in figure 7-3. On the upper limit line a particle velocity of 2 in/sec occurs at a scaled distance of 17.2 $ft/lb^{1/2}$. A level of 0.75 in/sec is reached at a scaled distance of 27 ft/lb. At a scaled distance of 72 $ft/lb^{1/2}$ the particle velocity at the upper limit line is 0.1 in/sec, the level at which little or no citizen complaint would be expected.

At Beulah only one reading exceeded 0.1 in/sec. This value is somewhat anomalous compared to other results. It may have resulted from sympathetic detonations between decks or unusual scatter in the delay times. An upper limit line excluding this data point is also shown on the graph. This line would provide more flexibility in blast design. However, since there are relatively few data points on the chart using the more conservative limit line would be prudent.

At the Beulah site the underground workings were extensive. Large rooms (22 x 10 feet) were separated by 18-foot rib pillars. The extent of the workings led to the seismograph being set up over mined excavations. This may account for the high frequencies and low vibration readings experienced.

7.4.1.2 <u>White Test Site</u>

When one examines the results of the eight blasts in Montana (table 7-6) it can readily be seen that the frequencies were typically lower than at Beulah and the peak particle velocities were usually higher. The frequencies were well within the range of the natural frequencies of residences. Often these were in the range of whole structure response (4-12 Hz). The correspondence of lower frequency to the higher levels of vibration or to the peak particle velocity in any mode, in a given event, was not as evident at the White Site as at Beulah.

At the White test area measurements were obtained over a broader range of scaled distances than at Beulah because of the pattern shots. Significant levels of vibration developed at the low scaled distances. Measurements were taken at fairly close in range which did not exceed 450 feet.

From the upper limit line in figure 7-10 it can be determined that a peak particle velocity as high as 2 in/sec is possible at a scaled distance of 16.5 $ft/lb^{1/2}$. A level of 0.75 in/sec can be reached at a scaled distance 29 $ft/lb^{1/2}$. The level of 0.1 in/sec, which is often attractive to design for to eliminate human response, can occur at a scaled distance of 105 $ft/lb^{1/2}$. This may be hard to achieve in some cases, even in AML work, however, a maximum level of 0.2 in/sec, which will also eliminate most complaints, can be obtained at a scaled distance of 70 $ft/lb^{1/2}$.

At the White site particle velocities as high as 0.93 in/sec were measured 270 feet from the blast. However, at a distance of 500 feet this would have reduced to 0.62 in/sec at the upper limit line (S.D. = 34.7 $ft/lb^{1/2}$) and 0.198 in/sec at the best fit line. These levels would be quite acceptable, however the higher level could lead to some citizen concern.

It is important to realize that the particle velocities measured in the last three shots could have been further reduced. These were pattern shots and multiple decks were detonated per delay. If necessary more surface delays could be used in the same shot to reduce the charge weight detonated per delay and hence the particle velocity. Since there were no residences near to the blasting site there was no need to further delay these shots and the method selected allowed vibration levels at low scaled distances to be obtained.

At this test area the seismograph was normally located off the old workings. Also the workings were narrow and low when compared to those at Beulah. These facts may have contributed to the frequencies and particle velocities seen at the White Site. The reason the seismograph was set up off the mined area was that the site was small and one did not want to have previously blasted material between the shot and the seismograph.

7.4.1.3 <u>Combined Test Results</u>

When a plot of peak particle velocity versus scaled distance was made for all the data obtained in this project a typical chart was obtained (figure 7-12). However, the upper limit line was slightly more conservative than for either site plotted separately.

A particle velocity of 2 in/sec could be obtained at a scaled distance of 24 $ft/lb^{1/2}$ in this case. The 0.75 in/sec level could be experienced, at the upper limit line, at a scaled distance of 37.5 $ft/lb^{1/2}$. The 0.1 in/sec level could occur on the upper limit line at a scaled distance of 74 $ft/lb^{1/2}$.

When compared with the typical vibration graph from many mines previously presented in figure 7-3 the combined chart has a quite similar trend. However, the upper limit line is at a somewhat more steep slope than the previous graph. All of the data points but one on the current graph would fit under the upper limit line in figure 7-3. The remaining point, 0.62 in/sec at 34.6 $ft/lb^{1/2}$ would plot only slightly above the limit line on that chart.

The other combined plot (figure 7-4) included points from previous work done at the Beulah site in an area east of the test area 2. Including this data gave a peak particle velocity, scaled distance graph based on 38 data points. When this data was added an upper limit line with a considerably flatter slope resulted. This appears to have developed because the particle velocities in the previous work tended to be significantly higher for a given scaled distance than were obtained in the current work. There was little that was different between the two studies except that in the current case the confinement was less and the size of the shots was smaller at the Beulah site. What may be represented is the effect of confinement on blasting ground vibrations and also additive effects from larger blasts.

In this plot a level of 2 in/sec can be reached at the upper limit line at a scaled distance of 11.5 ft/lb$^{1/2}$. The 0.75 in/sec level can occur at a scaled distance of 30 ft/lb$^{1/2}$. To not exceed 0.10 in/sec a scaled distance of 200 ft/lb$^{1/2}$ is required, which would often be difficult to achieve. A level of 0.2 in/sec is attained at 100 ft/lb$^{1/2}$ which, in AML work, may often be possible. On the upper limit line a scaled distance of 70 ft/lb$^{1/2}$ would not lead to vibration levels above 0.30 in/sec which would still negate by far the majority of potential complaints.

7.4.1.4 Interpretation of Ground Vibration Results

A review of the tables and graphs prepared and presented above leads to the conclusion that particle velocities and frequencies can vary from site to site in AML blasting work. Therefore, it would seem wise to monitor each site with a seismograph to insure that no unacceptable levels of vibration are produced. As data is developed it should be plotted on a particle velocity versus scaled distance chart and an upper limit line developed.

Based on all of the data we have obtained designing blasts to a scaled distance of at least 70 ft/lb$^{1/2}$ will not only avoid any damage to residences but will also eliminate virtually all complaints due to human perception of the blasting ground vibrations. Whenever possible designing to a scaled distance of 100 ft/lb$^{1/2}$ will be an even better approach. The methods developed during this research, which use independently delayed decks of low explosive weight, will allow such scaled distances to be maintained even at relatively close distances to residences and other structures.

The vibration results obtained have also given an indication of the effect confinement has on AML blasting. Both the White site and the earlier work at the Beulah site consisted of blasting material down into the mined rooms. Since there was no vertically standing free face and the rooms below were often partially filled with debris it is reasonable to consider that these shots were heavily confined. The current work at Beulah, on the other hand involved displacing material into a central void. The goal was to have the charges near enough to the face to obtain good displacement without excessive airblast. Therefore, the combination of a central void, reasonable burdens and mined voids below led to much less confined shots when closing individual vertical openings.

For similar scaled distances the particle velocities at the White site and in earlier work at Beulah tended to be higher than those experienced when shooting individual sinkholes at Beulah during this research. Also, the frequencies of the ground motions tended to be higher when shooting in sinkholes than when collapsing rooms. Since many other factors affecting vibration were similar there is good reason to believe that the differences in vibration levels resulted from differences in confinement of the shots and from greater additive effects from the larger shots involved in blasting down the mined rooms.

The vibration results obtained at both sites are not unlike those one usually sees in mining. Comparison with the scaled distance plot in figure 7-3 confirms this. Only one data point obtained in this study would plot above the upper limit line on that graph.

The frequencies recorded are not unlike those in mining shots. Those at Beulah are perhaps somewhat higher than what might be normally seen in coal mine blasts. Those at the White site, however, are quite typical. The higher readings at Beulah are most likely due to less confinement and extensive undermining of a large area.

The frequencies are often in the range of natural response frequencies of residences. Therefore, if AML reclamation is performed near buildings it is important to develop designs that will minimize ground vibrations since vibrations in the range of the natural frequencies will be the ones most severely felt by the structures.

When the data collected is compared to current guidelines and regulations for controlling blast vibration one concludes that it is quite feasible to blast to within 500 feet of buildings when performing AML blasting. This conclusion is based on the structures being sited on unmined land. It is also based on the careful control of any fly rock. Experience gained in this research indicates that fly rock can be limited to 100 feet or less from the shot.

Table 7-7 lists the weight per delay that can be detonated at various distances from a building. This table is for scaled distances of 70 $ft/lb^{1/2}$ and 60 $ft/lb^{1/2}$.

TABLE 7-7: WEIGHT OF EXPLOSIVE PER DELAY THAT CAN BE DETONATED AT VARIOUS DISTANCES FROM A BUILDING FOR SCALED DISTANCES OF 70 FT/LB$^{1/2}$ AND 60 FT/LB$^{1/2}$

Distance feet	Weight of Explosive per Delay 70 ft/lb$^{1/2}$ pounds	60 ft/lb$^{1/2}$ pounds
250	12.8	17.4
500	51.0	69.4
600	73.5	100.0
700	100.0	136.1
800	130.6	177.8
900	165.3	225.0
1000	204.0	277.8

A typical, independently delayed deck loaded in a 6-inch blasthole is 65 pounds. Therefore, it would be possible to blast to within 600 feet of a structure while maintaining a scaled distance of 70 ft/lb$^{1/2}$. At 500 feet a scaled distance of 60 ft/lb$^{1/2}$ could be chosen for design which still allows 65 pounds to be detonated per delay. The alternative would be to modify the loads to 50 pounds per independently delayed deck. Doing so would require some changes to the pattern the blastholes are laid out on, but in many cases this is not expected to be a problem.

From the combined scaled distance plot in figure 7-12 the vibration level produced at 60 ft/lb$^{1/2}$ would not be expected to exceed 0.165 in/sec. The vibration level for a scaled distance of 70 ft/lb$^{1/2}$ would be a maximum of 0.11 in/sec. These vibration levels are quite modest. Damage to buildings ought not to be a concern. Further, it is quite unlikely that human response will lead to complaints by residents of the area.

A more conservative approach is taken if the chart that includes previous data is used (figure 7-14). In this case a scaled distance of 60 ft/lb$^{1/2}$ yields a maximum particle velocity of 0.345 in/sec. For a scaled distance of 70 ft/lb$^{1/2}$ the maximum particle velocity would be 0.295 in/sec. Again, these maximums, from the upper limit line, are low levels of vibration. Damage to buildings, except possibly old, fragile historic structures, would not be expected. Citizen complaint should also be minimal but may be more likely to occur than at the ground vibration levels cited from figure 7-12.

It is quite clear then that, from a ground vibration viewpoint, it is possible to successfully conduct AML blasting to within 500 feet of residences located on competent ground that has not been undermined. When blasting 800 to 1000 feet from residences some flexibility in the timing of pattern shots like those fired at the White test area is allowed. Beyond 1000 feet ground vibration will pose little problem in AML blasting and there will be excellent flexibility for designing the millisecond delay timing of pattern type blasts.

When blasting less than 1000 feet from residences with properly designed shots any citizen complaints are likely to result from one of two sources. One is airblast and is discussed below. The other is that persons watching the shot may be frightened or fear for damage to their homes due to the surface disruption and vertical displacement of material that they are able to observe. The only way to alleviate these fears will be to educate people, in advance of blasting, as to what can be expected. Public meetings should be held in which high-speed films and photographic slides are used to show the manifestation of typical shots. What is happening during the shot should be explained. Reasons why the physical manifestations of the blasts are not a problem should be discussed. If a good program of community education is not conducted prior to close-in blasting, then regardless of how well ground vibration and airblast are controlled considerable citizen concern, due to the physical manifestation of the blasts, is very likely.

Beyond 1000 feet the problem of physical manifestation causing fear should not be a problem. This has been the experience of the current research. During the test blasts at Beulah, however, each homeowner in the immediate vicinity was visited. It was explained that blasting operations would be conducted and why. This approach should always be taken. At the least such visits should be made to persons living within one-half mile of the AML site.

7.4.2 Airblast Results

The airblast results are typical of those observed in the industry. Only one of the twenty-one readings was above the 133dB recommended as maximum when using a 2Hz high pass system. A value of 134 dB was recorded for blast B7 at the White Site. However, the measurement was taken at 270 feet from the blast which is quite close. This was a pattern shot with more than one hole per delay and there was more detonating cord laid out on surface. The cord was buried but could be expected to lead to additional noise. The airblast from surface detonating cord is usually of higher frequency and attenuates more rapidly. Therefore, the 134 dB reading would have been expected to attenuate quickly with increasing distance from the shot.

The airblast results have been presented as plots of overpressure in pounds per square inch versus scaled distance. The scaled distance involves the cube root of the charge weight per delay which is the typical approach for plotting airblast measurements. The weights of explosive are the weight fired on an individual delay. In most cases this is the weight in one explosive deck, but for the pattern shots it involves multiple decks detonating per delay.

Such charts can be misleading since the weight per delay is only one variable affecting airblast. It would not account for uncovered delays and detonating cord on surface for example. However, for this research, all surface delays and detonating cord were well buried and therefore confined. Since these accessories were confined and the explosive weight contained was negligible relative to the deck explosive weights the graphs would not be much affected by the presence of these devices.

The results are also affected by weather as shown above. For scaled distance plots that contain large numbers of data points such effects may well be included for the upper limit line. So may variations due to differences in confinement (top and side) and in air displacements effects. Graphs generated for smaller numbers of points may be skewed due to conditions at the time of measurement. In the present case, therefore, the chart combining all results (34 data points including summer, fall and winter seasons) may well be the most representative. It is also the most conservation plot.

For the four airblast plots presented table 7-8 provides the equations which represent the best fit lines. The upper limit lines are parallel to the best fit lines but are drawn to include all the data points below the limit line.

TABLE 7-8: EQUATIONS REPRESENTING THE BEST FIT LINE FOR EACH AIRBLAST PLOT OF OVERPRESSURE VERSUS SCALED DISTANCE

Chart	Equation of Best Fit Line
Beulah (figures 7-9)	$P = 457.422\ S.D.^{-2.71848}$
White (figure 7-11)	$P = 8.207\ S.D.^{-1.80576}$
Combined (figure 7-13)	$P = 34.902\ S.D.^{-2.15318}$
All combined (figure 7-15)	$P = 0.834\ S.D.^{-1.31779}$

Using these graphs the airblast pressure in pounds per square inch and decibles have been determined for various cube root scaled distances for each plot. The results are listed in table 7-9. The values tabulated are the maximums to be expected at the upper limit line.

TABLE 7-9: AIRBLAST PRESSURES AND DECIBLE LEVELS FROM EACH OVERPRESSURE VERSUS CUBE ROOT SCALED DISTANCE GRAPH CALCULATED FOR VARIOUS SCALED DISTANCES AT THE UPPER LIMIT LINE

Chart	Scaled Distance, ft/lb^1/3									
	50		75		100		125		150	
	psi	dB	psi	dB	psi	dB	psi	dB	psi	dB
Beulah (fig. 7-9)	0.0245	138.5	0.0120	132.9	0.0035	121.6	0.00200	116.8	0.00125	112.7
White (fig. 7-11)	0.0175	135.6	0.0082	129.0	0.0049	124.6	0.00340	121.4	0.00240	118.4
Combined (fig. 7-13)	0.0200	136.8	0.0085	129.9	0.0046	124.0	0.00295	120.1	0.00200	116.8
All Comb. (fig. 7-15)	0.0270	139.4	0.0155	134.6	0.0105	131.2	0.00800	126.8	0.00630	126.7

The results for the Beulah and White sites are very similar at low scaled distances. However for scaled distances of 100 $ft/lb^{1/3}$ and above the Beulah plot gives smaller levels of airblast pressure for the same scaled distance than the White Site. This is because of the steeper slope of the Beulah limit line. Most shots at Beulah had little detonating cord on surface. At the White test area considerably more was used for the pattern shots. Therefore the more gentle slope of the White site upper limit line may reflect the added contribution of more surface cord, even though this was buried.

The combined plot gives results intermediate between the Beulah and White areas as would be expected. The plot, which includes the prior results at Beulah, however, has a flatter slope than any of the others. The attenuation of airblast pressure with increasing scaled distance is not nearly as pronounced. This occurs because the earlier Beulah data had some cases where the scaled distances were high but the airblast also remained significant.

The graph in figure 7-15, therefore, yields the most conservative result. The equation of the best fit line for this data, comprising thirty-four data points, is
$P = 0.834\ S.D.^{-1.31779}$. In the earlier stages of an AML project this would be a good chart to use. Examining table 7-9 the minimum scaled distance to a residence should be 100 $ft/lb^{1/3}$. The other charts would suggest 75 $ft/lb^{1/3}$ as being acceptable albeit marginal in one case. The 100 $ft/lb^{1/3}$ scaled distance should insure that difficulties in damage should not result. Greater scaled distances may well be required to minimize citizen complaint.

Once the project is underway airblast measurements should be taken for each shot. A scaled distance plot can be prepared from which a site specific minimum scaled distance can be developed. Ongoing work on the AML project could then be done on that basis.

If one chooses to be quite conservative then the Bureau of Mines guidelines, outlined earlier in this Chapter could be used. In that case it is recommended that the coal mine highwall value of 180 $ft/lb^{1/3}$ be used for caving in rooms and entries and the quarry and mine value of 250 $ft/lb^{1/3}$ be used for closing individual sinkholes.

It is to be emphasized that the scaled distances resulting from this research involve good and proper procedures for protecting against airblast. All surface delays and detonating cord should be buried under at least 6 inches of drill cuttings. This is not difficult and is a common procedure in quarries excavating stone near to residential areas. Adequate stemming is to be used to avoid bursting through the upper surface. Dipping the hole with the blasting tape while loading the upper deck, to insure a correct and consistent load is a necessity. When blasting in individual sinkholes the charges should be given adequate burden to avoid bursting of gases and attendant airblast.

For a scaled distance of 100 ft/lb$^{1/3}$ there is no reason why properly controlled blasting cannot take place to 500 feet from residences. The weight per delay that can be shot at different distances for 100 ft/lb$^{1/3}$ scaled distance are shown in table 7-10. From the table one can observe that at 500 feet adequate weight per delay can be detonated to satisfy AML blasting needs when a 6-inch blasthole is being used. Beyond 500 feet increasing flexibility is allowed. In fact the weight per delay will be controlled by the ground vibrations rather than by airblast.

TABLE 7-10: WEIGHT OF EXPLOSIVE PER DELAY THAT CAN BE DETONATED AT VARIOUS DISTANCES FOR A SCALED DISTANCE OF 100 FT/LB$^{1/3}$

Distance (feet)	Weight of Explosive per Delay (pounds)
250	16
500	125
600	216
700	343
800	512
900	729
1000	1000

It is again to be emphasized that all proper precautions should be taken to minimize the airblast. These precautions were taken during this research and the results reflect this fact. If procedures such as burying delays and detonating cord are not to be used then the Bureau of Mines criteria should be followed.

7.4.3 Summary

In summary the following may be said:

1. The data collected at the two sites has shown moderate levels of ground vibration and airblast.

2. There were variances in frequency and vibration levels between the sites that suggest that the degree and nature of undermining may affect results as well as local differences in geology.

3. There were no citizen complaints to either ground vibration or airblast.

4. From the measurement of ground vibration at distances of 500 feet or less and the resultant peak particle velocity versus scaled distance plots it is concluded that AML blasting to within 500 feet of residences is feasible provided the residences are not located over mined areas.

5. From the measurement of airblast at distances between 270 and 500 feet and the airblast pressure versus scaled distance charts drawn it is concluded that blasting within 500 feet of residences is feasible if proper precautions are taken to minimize the airblast.

6. When blasting less than 1000 feet from residences a good program of homeowner education is recommended before blasting begins. Preblast survey may also be considered.

CHAPTER 8
ANALYSIS OF RESULTS OF BLASTING AT BEULAH SITE

8.1 GENERAL DISCUSSION

The filling of individual sink holes by blasting, tested at the Beulah Site, has been generally successful. The greatest problems experienced were in those cases where the vertical opening was small at the surface and large below. The opening was then undercut and it was more difficult to position the truck mounted drill in the best position. The second problem was openings that were large in dimension and had a large ramp declining into the mine workings. These situations tended to lead to ratholing at the ends of the sinkhole, where it met the old works. A second shot was necessary, in several cases, to further close the vertical opening.

For openings VO-11 and VO-12 two rows of holes were employed because a substantial volume of material was needed to fill the large openings. The use of two rows of blastholes was quite successful. Therefore, this approach could solve the problem of ratholing without the need for a second blast in many cases. The approach could also be useful when the vertical opening is undercut. However in these cases it may be necessary to perform the first shot to square up the opening and a second to complete the filling of it.

When filling vertical openings considerable field judgement is needed regarding the patterns on which the holes are laid out. The openings are often irregular in shape and blastholes must be adjusted to best suit the situation. When there is a declining ramp to the mine openings it is difficult to project the profile to surface so one must make estimates of where the holes along the decline should be drilled. Also, some thought must be given to how the blast is to be stopped so that disturbance beyond the area to be filled is minimized. It has been found that by careful field observation, combined with measurements of the the openings, holes could be placed to provide a good result. Measurements of the openings frequently had to be approximate, however, so good field observation and judgement was essential to success.

A concern when working equipment in areas where subsidence has occurred is that there will be collapse under the machinery which could lead to damage and injury. For this reason blastholes were carefully staked around the openings with the intent of not placing the drill in a precarious position. The driller's input was always sought concerning safe drill locations. During this research on two sites and in previous work there has been no case recorded of equipment being involved in a cave-in. In a few cases it was discovered that as little as ten feet of solid material lay beneath the drill. In these situations the drill was removed from the area immediately without incident.

It appears, therefore, that the drill can be maneuvered in the area with a good degree of safety. Caution should be exercised at all times to insure that safe operation is addressed. When the drill strikes void less than twelve feet beneath surface it should be moved. If the ground feels broken when drilling the unit should be moved, even if the void is fifteen feet below surface. With care safety can be provided. Certainly, there is no more risk than for other mitigation methods which also require equipment to work in the area of disturbance.

Further enhanced safety, for this or any other reclamation method, could be attained if there were reliable seismic methods for surveying the area in advance of operations. Such devices would need to reliably identify voids to a depth of 20 feet. There has been considerable study of this technology, however, much of the results have been applicable to quite shallow situations, or rather unreliable. This area of endeavour should be further considered however, because of the potential to improve operational safety.

A primary consideration when using blasting to fill individual sinkholes is that one does not want major disturbance beyond the sinkhole and does not want caving to occur over workings other than in the immediate area of the vertical opening. Were this to be a pronounced problem the method would be precluded.

Blasting at Beulah was performed on thirteen vertical openings some of which were blasted twice. A total of eighteen blasts were detonated. In no case did caving occur on an adjacent room, which indicated that the blasting could be well controlled.

In addition, there was little disturbance and caving, on the room where blasting was being conducted, beyond the immediate vicinity of the shot. The most pronounced case of added disturbance was VO-9 which was an extremely undercut and complex vertical opening. The depression formed was deeper and of larger area than expected. The indication was that vertical openings like VO-9 are more difficult to control. However, there was no disturbance not contiguous to the blasted area, so the result was manageable and a second shot alleviated most problems with ratholing.

The conclusion is that blasting isolated sinkholes can be controlled provided adequate but small powder loads are used to minimize vibration in adjacent areas, the shot is well delayed and there is sufficient relief in the direction of the opening.

8.2 ANALYSIS OF DATA FROM THE VERTICAL OPENING BLASTS

8.2.1 General Information

The data collected during blasting at Beulah has been analyzed by deck. The length of the decks, the explosive type and weight in the decks and the scaled burden and radius on the charges has been studied. Blast VO-1 and VO-2 are not included in this analysis. Methodology was still being worked out during the first two shots and not all the necessary data for this analysis was available.

The analysis is presented in tables 8-1 through 8-16. In these tables deck 1 is the lowest deck in the hole and deck 3 is the highest. Where only one column of explosive was loaded it is designated as deck 1. In the case of two explosive decks, these are shown as deck 1 and deck 2.

The burial is the distance from the hole to the face or previous row of holes. The spacing is the distance between holes. If the spacing between holes was not uniform then the spacing is one-half the distance to one hole plus one-half the distance to the hole on the other side of the blasthole of interest. The scaled radius is taken as one-half the stated spacing.

8.2.2 Explosive Density

The loading density of the explosive is listed for each blast. This density has been found by taking the recorded weight of the explosive used and the total volume loaded in the blast (according to the total deck length and a 6-inch hole diameter). These values are recorded.

It has been found that there is considerable variation in the loading densities. This is especially true where the PowerAN heavy ANFO product was used. Less variation was seen in ANFO densities, however, for some blasts the density was higher than would be expected. In a few cases the calculated densities were not reasonable and were adjusted to be within the expected range.

Variations in the HANFO density very likely result from the fact that the product is bagged in 5-inch diameter. Therefore, the explosive, as loaded, is decoupled which would reduce the loading density relative to the bagged density. Since there is likely to be variations from blast to blast in how much the bags compact as they are dropped, this will lead to variations in density as loaded.

Other factors, applicable to both explosives used include variations in hole diameter, in deck lengths and in estimating part bags of powder. All holes were drilled at 6-inch diameter. The material at Beulah is overconsolidated clays with some weak sandstone lenses. Therefore sluffing and caving of the borehole walls is possible which would alter the explosive loads and densities. Also, bit wear could reduce the actual hole diameter over a period of time.

TABLE 8-1: TECHNICAL DATA FOR BUELAH BLAST VO-3

Site: Beulah Blast Number: VO-3 Date: 8-8-88 Time: 12:15 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	23	6	Hanfo	0.91	66.9	7.0	7.0	1.72	0.86	3	Anfo	0.85	31.2	7.0	7.0	2.22	1.11
2	23	6	Hanfo	0.91	66.9	7.0	8.5	1.72	1.05	3	Anfo	0.85	31.2	7.0	8.5	2.22	1.35
3	11	2	Anfo	0.85	20.8	10.0	9.0	3.63	1.64								
4A	23	6	Hanfo	0.91	66.9	11.0	11.0	2.71	1.36	3	Anfo	0.85	31.2	11.0	11.0	3.49	1.75
5	23	6	Hanfo	0.91	66.9	8.0	10.0	1.97	1.23	3	Anfo	0.85	31.2	8.0	10.0	2.54	1.59
TOTALS:		26			288.4					12			125.0				
Mean:		5.2			57.7			2.35	1.23	3.0			31.2			2.62	1.45
St. Dev:		1.6			0.0			0.74	0.26	0.0			0.0			0.52	0.24
Maximum:		6.0			66.9			3.63	1.64	3.0			31.2			3.49	1.75
Minimum:		2.0			20.8			1.72	0.86	3.0			31.2			2.22	1.11

TABLE 8-2: TECHNICAL DATA FOR BEULAH BLAST VO-4A

: Site: Beulah Blast Number: VO-4A   Date: 8-8-88   Time: 12:58 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	18	4	Hanfo	1.12	54.9	8	10	2.11	1.32	3	Anfo	0.82	30.1	8	10	2.57	1.61
2	18	4	Hanfo	1.12	54.9	8	10	2.11	1.32	3	Anfo	0.82	30.1	8	10	2.57	1.61
3	24	5	Hanfo	1.12	68.6	10	10	2.44	1.22	4	Anfo	0.82	40.2	10	10	2.92	1.46
4	20	5	Hanfo	1.12	68.6	10	10	2.44	1.22	3	Anfo	0.82	30.1	10	10	3.21	1.61
5	18	4	Hanfo	1.12	54.9	10	10	2.63	1.32	3	Anfo	0.82	30.1	10	10	3.21	1.61
6	20	5	Hanfo	1.12	68.6	8	10	1.95	1.22	3	Anfo	0.82	30.1	8	10	2.57	1.61
7	18	4	Hanfo	1.12	54.9	8	10	2.11	1.32	3	Anfo	0.82	30.1	8	10	2.57	1.61
8	18	4	Hanfo	1.12	54.9	8	10	2.11	1.32	3	Anfo	0.82	30.1	8	10	2.57	1.61
TOTALS:		35			480.3					25.0			251.2				
Mean:		4.4			60.0			2.24	1.28	3.1			31.4			2.78	1.59
St. Dev:		0.5			6.6			0.22	0.05	0.3			3.3			0.28	0.05
Maximum:		5.0			68.6			2.63	1.32	4.0			40.2			3.21	1.61
Minimum:		4.0			54.9			1.95	1.22	3.0			30.1			2.57	1.46

TABLE 8-3: TECHNICAL DATA FOR BEULAH BLAST VO-4A #2

Site: Beulah Blast Number: VO-4A#2 Date: 8-17-88 Time: 5:25 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	17.5	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
2	18.0	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
3	18.0	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
TOTALS:		9			93.7					9			93.7				
Mean:		3.0			31.2			2.54	1.59	3.0			31.2			2.54	1.59
St. Dev:		0.0			0.0			0.00	0.00	0.0			0.0			0.00	0.00
Maximum:		3.0			31.2			2.54	1.59	3.0			31.2			2.54	1.59
Minimum:		3.0			31.2			2.54	1.59	3.0			31.2			2.54	1.59

TABLE 6-4: TECHNICAL DATA FOR BEULAH BLAST VO-48

Site: Beulah Blast Number: VO-48 Date: 8-17-88 Time: 5:10 PM

Hole Number	Hole Depth	DECK 1 Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	DECK 2 Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	24	6	Anfo	0.82	60.3	8	10	2.04	1.28	4	Anfo	0.82	40.2	8	10	2.34	1.46
2	18	3	Anfo	0.82	30.1	8	10	2.57	1.61	3	Anfo	0.82	30.1	8	10	2.57	1.61
3	30	4	Hanfo	1.10	53.9	6	10	1.59	1.32	4	Anfo	0.82	40.2	6	10	1.75	1.46
4	34	6	Hanfo	1.10	80.9	6	10	1.39	1.16	5	Anfo	0.82	50.2	6	10	1.63	1.36
5	34	6	Hanfo	1.10	80.9	6	10	1.39	1.16	5	Anfo	0.82	50.2	6	10	1.63	1.36
6	30	4	Hanfo	1.10	53.9	6	10	1.59	1.32	4	Anfo	0.82	40.2	6	10	1.75	1.46
7	24	6	Anfo	0.82	60.3	8	10	2.04	1.28	4	Anfo	0.82	40.2	8	10	2.34	1.46
8	24	6	Anfo	0.82	60.3	8	10	2.04	1.28	4	Anfo	0.82	40.2	8	10	2.34	1.46
TOTALS:		41			480.5					33.0			331.5				
Mean:		5.1			60.1			1.83	1.30	4.1			41.4			2.04	1.45
St. Dev:		1.2			16.2			0.39	0.18	0.6			6.0			0.36	0.07
Maximum:		6.0			80.9			2.67	1.61	6.0			50.2			2.67	1.61
Minimum:		3.0			30.1			1.39	1.16	3.0			30.1			1.63	1.36

TABLE 8-4: CONTINUED

Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
DECK 3							
4	Anfo	0.82	40.2	6	10	1.75	1.46
4	Anfo	0.82	40.2	6	10	1.75	1.46
4	Anfo	0.82	40.2	6	10	1.75	1.46
4	Anfo	0.82	40.2	6	10	1.75	1.46
16.0			160.7				
4.0			40.2			1.75	1.46
0.0			0.0			0.00	0.00
4.0			40.2			1.75	1.46
4.0			40.2			1.75	1.46

TABLE 6-5: TECHNICAL DATA FOR BEULAH BLAST VO-5

Site: Beulah Blast Number: VO-5 Date: 6-16-88 Time: 4:15 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	18	3	Anfo	0.84	30.9	7	10	2.23	1.59	3	Anfo	0.84	30.9	7	10	2.23	1.59
2	15	6	Anfo	0.84	61.8	13	14	3.29	1.77								
3	23	5	Hanfo	0.92	56.4	8	13	2.09	1.70	4	Anfo	0.84	41.2	8	13	2.32	1.88
4	18	3	Hanfo	0.92	33.8	8	10	2.47	1.65	3	Anfo	0.84	30.9	8	10	2.55	1.59
5	17	3	Hanfo	0.92	33.8	8	10	2.47	1.65	3	Anfo	0.84	30.9	8	10	2.55	1.59
6	23	5	Hanfo	0.92	56.4	6	10	1.67	1.30	4	Anfo	0.84	41.2	6	10	1.74	1.45
7	16	7	Anfo	0.84	72.0	6	10	1.44	1.20								
8	20	5	Hanfo	0.92	56.4	6	10	1.67	1.30	4	Anfo	0.84	41.2	6	10	1.74	1.45
9	14	5	Anfo	0.84	51.5	6	10	1.61	1.34								
10	18	3	Hanfo	0.92	33.8	8	10	2.47	1.65	3	Anfo	0.84	30.9	8	10	2.55	1.59
11	18	3	Hanfo	0.92	33.8	8	10	2.47	1.65	3	Anfo	0.84	30.9	8	10	2.55	1.59
12	23	5	Hanfo	0.92	56.4	6	14	1.67	1.88	4	Anfo	0.84	41.2	6	14	1.74	2.03
13	18	3	Anfo	0.84	30.9	7	10	2.23	1.59	3	Anfo	0.84	30.9	7	10	2.23	1.59
TOTALS:		56.0			607.7					34.0			349.9				
Mean:		4.3			46.7			2.11	1.52	3.4			36.0			2.22	1.64
St. Dev:		1.3			13.7			0.52	0.18	0.5			5.0			0.34	0.17
Maximum:		7.0			72.0			3.29	1.88	4.0			41.2			2.55	2.03
Minimum:		3.0			30.9			1.44	1.20	3.0			30.9			1.74	1.45

TABLE 8-6: TECHNICAL DATA FOR BEULAH BLAST V0-6

Site: Beulah Blast Number: V0-6 Date: 8-22-88 Time: 4:10PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	14	5	Anfo	0.82	50.2	8	10	2.17	1.36								
2	15	6	Anfo	0.82	60.3	8	10	2.04	1.28								
3	17	3	Anfo	0.82	30.1	10	10	3.21	1.61	3	Anfo	0.82	30.1	10	10	3.21	1.61
4	18	3	Anfo	0.82	30.1	8	10	2.57	1.61	3	Anfo	0.82	30.1	8	10	2.57	1.61
5	18	3	Anfo	0.82	30.1	8	10	2.57	1.61	3	Anfo	0.82	30.1	8	10	2.57	1.61
6	15	6	Anfo	0.82	60.3	8	10	2.04	1.28								
TOTALS:		26.0			261.2					9.0			90.4				
Mean:		4.3			43.5			2.43	1.46	3.0			30.1			2.79	1.61
St. Dev:		1.4			13.8			0.41	0.15	0.0			0.0			0.30	0.00
Maximum:		6.0			60.3			3.21	1.61	3.0			30.1			3.21	1.61
Minimum:		3.0			30.1			2.04	1.28	3.0			30.1			2.57	1.61

TABLE 8-7: TECHNICAL DATA FOR BEULAH BLAST VO-788-N

Site: Beulah Blast Number: VO-788N Date: 8-19-88 Time: 4:20 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
7-14	14	5	Anfo	0.85	52.1	8	12	2.14	1.61								
7-15	15	6	Anfo	0.85	62.5	8	11	2.02	1.39								
7-16	18	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
7-17	24	5	Anfo	0.85	52.1	9	10	2.41	1.34	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-18	25	6	Anfo	0.85	62.5	9	10	2.27	1.26	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-19	30	4	Hanfo	1.14	55.9	9	10	2.35	1.31	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-20	30	4	Hanfo	1.14	55.9	9	10	2.35	1.31	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-21	35	6	Hanfo	1.14	83.8	9	10	2.06	1.14	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-22	35	6	Hanfo	1.14	83.8	9	10	2.06	1.14	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-23	30	4	Hanfo	1.14	55.9	9	10	2.35	1.31	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-24	30	4	Hanfo	1.14	55.9	9	10	2.35	1.31	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-25	25	6	Anfo	0.85	62.5	9	10	2.27	1.26	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-26	24	5	Anfo	0.85	52.1	9	10	2.41	1.34	4	Anfo	0.85	41.7	9	10	2.60	1.44
7-27	18	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
8-3	15	6	Anfo	0.85	62.5	8	12	2.02	1.51								
8-4	15	6	Anfo	0.85	62.5	8	12	2.02	1.51								
8-5	15	6	Anfo	0.85	62.5	8	10	2.02	1.26								
8-6	20	4	Anfo	0.85	41.7	6	10	1.73	1.44	3	Anfo	0.85	31.2	6	10	1.91	1.59
8-7	25	6	Anfo	0.85	62.5	6	10	1.51	1.26	4	Anfo	0.85	41.7	6	10	1.73	1.44
8-8	23	5	Anfo	0.85	52.1	6	10	1.61	1.34	3	Anfo	0.85	31.2	6	10	1.91	1.59
8-9	25	6	Anfo	0.85	62.5	6	10	1.51	1.26	4	Anfo	0.85	41.7	6	10	1.73	1.44
8-10	29	4	Hanfo	1.14	55.9	6	10	1.57	1.31	4	Anfo	0.85	41.7	6	10	1.73	1.44
8-11	35	6	Hanfo	1.14	83.8	6	10	1.37	1.14	5	Anfo	0.85	52.1	6	10	1.61	1.34
8-12	29	4	Hanfo	1.14	55.9	6	10	1.57	1.31	4	Anfo	0.85	41.7	6	10	1.73	1.44
8-13	27	5	Hanfo	1.14	69.8	6	10	1.46	1.21	3	Anfo	0.85	31.2	6	10	1.91	1.59
8-14	25	6	Anfo	0.85	62.5	6	10	1.51	1.26	4	Anfo	0.85	41.7	6	10	1.73	1.44
8-15	18	3	Anfo	0.85	31.2	6	10	1.91	1.59	3	Anfo	0.85	31.2	6	10	1.91	1.59
8-16	14	5	Anfo	0.85	52.1	8	10	2.14	1.34								
TOTALS:		139			1514.6					83			864.4				
Mean:		5.0			57.7			2.00	1.34	3.8			39.3			2.22	1.48
St. Dev:		1.1			13.3			0.36	0.18	0.5			5.4			0.40	0.07
Maximum:		6.0			83.8			2.54	1.61	5.0			52.1			2.60	1.59
Minimum:		3.0			31.2			1.37	1.14	3.0			31.2			1.61	1.34

TABLE 8-7: CONTINUED

Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
DECK 3							
4	Anfo	0.85	41.7	9	10	2.60	1.44
4	Anfo	0.85	41.7	9	10	2.60	1.44
4	Anfo	0.85	41.7	9	10	2.60	1.44
4	Anfo	0.85	41.7	9	10	2.60	1.44
4	Anfo	0.85	41.7	9	10	2.60	1.44
4	Anfo	0.85	41.7	9	10	2.60	1.44
3	Anfo	0.85	31.2	6	10	1.91	1.59
4	Anfo	0.85	41.7	6	10	1.73	1.44
3	Anfo	0.85	31.2	6	10	1.91	1.59
3	Anfo	0.85	31.2	6	10	1.91	1.59
37			385.3				
3.7			38.5			2.30	1.49
0.5			4.8			0.36	0.07
4.0			41.7			2.60	1.59
3.0			31.2			1.73	1.44

TABLE 8-8: TECHNICAL DATA FOR BEULAH BLAST VO-788N-#2

Site: Beulah Blast Number: VO-788N#2 Date: 8-25-88 Time Shot: 12:32 PM

Hole Number	Hole Depth	DECK 1 Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
2	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
3	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
4	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
5	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
6	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
7	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
8	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
9	15	6	Anfo	0.85	62.5	8	10	2.02	1.26
TOTALS:		54.0			562.4				
Mean:		6.0			62.5			2.02	1.26
St. Dev:		0.0			0.0			0.00	0.00
Maximum:		6.0			62.5			2.02	1.26
Minimum:		6.0			62.5			2.02	1.26

TABLE 8-9: TECHNICAL DATA FOR BEULAH BLAST VO-788-S

Site: Beulah Blast Number: VO-7885 Date: 8-23-88 Time: 3:50 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
7-1	15	6	Anfo	0.85	62.5	8	10	2.02	1.26								
7-2	18	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
7-3	23	5	Anfo	0.85	52.1	7	10	1.87	1.34	3	Anfo	0.85	31.2	7	10	2.22	1.59
7-4	24	5	Anfo	0.85	52.1	7	10	1.87	1.34	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-5	29	4	Anfo	0.85	41.7	7	10	2.02	1.44	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-6	30	4	Anfo	0.85	41.7	7	10	2.02	1.44	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-7	29	4	Anfo	0.85	41.7	7	10	2.02	1.44	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-8	29	4	Anfo	0.85	41.7	7	10	2.02	1.44	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-9	23	5	Anfo	0.85	52.1	7	10	1.87	1.34	3	Anfo	0.85	31.2	7	10	2.22	1.59
7-10	25	6	Anfo	0.85	62.5	7	10	1.76	1.26	4	Anfo	0.85	41.7	7	10	2.02	1.44
7-11	17	3	Anfo	0.85	31.2	8	11	2.54	1.75	3	Anfo	0.85	31.2	8	11	2.54	1.75
7-12	13	4	Anfo	0.85	41.7	8	12	2.31	1.73								
8-17	18	3	Anfo	0.85	31.2	8	10	2.54	1.59	3	Anfo	0.85	31.2	8	10	2.54	1.59
8-18	22	5	Anfo	0.85	52.1	8	10	2.14	1.34	3	Anfo	0.85	31.2	8	10	2.54	1.59
8-19	25	6	Anfo	0.85	62.5	8	10	2.02	1.26	4	Anfo	0.85	41.7	8	10	2.31	1.44
8-20	28	5	Anfo	0.85	52.1	8	10	2.14	1.34	3	Anfo	0.85	31.2	8	10	2.54	1.59
8-21	28	5	Anfo	0.85	52.1	8	10	2.14	1.34	3	Anfo	0.85	31.2	8	10	2.54	1.59
8-22	29	4	Anfo	0.85	41.7	8	10	2.31	1.44	3	Anfo	0.85	31.2	8	10	2.54	1.59
TOTALS:		81.0			843.6					55.0			572.8				
Mean:		4.5			46.9			2.12	1.43	3.4			35.8			2.29	1.53
St. Dev:		1.0			10.0			0.23	0.14	0.5			5.2			0.24	0.09
Maximum:		6.0			62.5			2.54	1.76	4.0			41.7			2.54	1.75
Minimum:		3.0			31.2			1.76	1.26	3.0			31.2			2.02	1.44

TABLE 8-9: CONTINUED

Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
DECK 3							
3	Anfo	0.85	31.2	7	10	2.22	1.59
4	Anfo	0.85	41.7	7	10	2.02	1.44
3	Anfo	0.85	31.2	7	10	2.22	1.59
3	Anfo	0.85	31.2	7	10	2.22	1.59
3	Anfo	0.85	31.2	8	10	2.54	1.59
3	Anfo	0.85	31.2	8	10	2.54	1.59
3	Anfo	0.85	31.2	8	10	2.54	1.59
22.0			229.1				
3.1			32.7			2.33	1.57
0.3			3.6			0.19	0.05
4.0			41.7			2.54	1.59
3.0			31.2			2.02	1.44

TABLE 8-10: TECHNICAL DATA FOR BEULAH BLAST VO-75-#2

Site: Beulah Blast Number: VO-75#2 Date: 8-25-88 Time: 12:32

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	17	3	Anfo	0.86	31.6	12	10	3.80	1.58	3	Anfo	0.86	31.6	12	10	3.80	1.58
2	17	3	Anfo	0.86	31.6	12	10	3.80	1.58	3	Anfo	0.86	31.6	12	10	3.80	1.58
3	19	4	Anfo	0.86	42.1	12	10	3.45	1.44	3	Anfo	0.86	31.6	12	10	3.80	1.58
4	19	4	Anfo	0.86	42.1	11	10	3.16	1.44	3	Anfo	0.86	31.6	11	10	3.46	1.58
TOTALS:		14.0			147.5					12.0			126.4				
Mean:		3.5			36.9			3.55	1.51	3.0			31.6			3.72	1.58
St. Dev:		0.5			5.3			0.27	0.07	0.0			0.0			0.14	0.00
Maximum:		4.0			42.1			3.80	1.58	3.0			31.6			3.80	1.58
Minimum:		3.0			31.6			3.16	1.44	3.0			31.6			3.46	1.58

TABLE 8-11: TECHNICAL DATA FOR BEULAH BLAST VO-9

Site: Beulah Blast Number: VO-9 Date: 8-18-88 Time: 4:18 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	20	5	Hanfo	1.1	67.4	9	8	2.21	0.98	3	Anfo	0.86	31.6	9	8	2.85	1.27
2	19	4	Hanfo	1.1	53.9	9	8	2.38	1.06	3	Anfo	0.86	31.6	9	8	2.85	1.27
3	20	5	Hanfo	1.1	67.4	9	8	2.21	0.98	3	Anfo	0.86	31.6	9	8	2.85	1.27
4	25	7	Hanfo	1.1	94.3	9	8	1.98	0.88	4	Anfo	0.86	42.1	9	8	2.59	1.15
5	20	5	Hanfo	1.1	67.4	9	8	2.21	0.98	3	Anfo	0.86	31.6	9	8	2.85	1.27
6	19	4	Hanfo	1.1	53.9	9	8	2.38	1.06	3	Anfo	0.86	31.6	9	8	2.85	1.27
7	19	4	Hanfo	1.1	53.9	9	8	2.38	1.06	3	Anfo	0.86	31.6	9	8	2.85	1.27
8	20	5	Hanfo	1.1	67.4	9	8	2.21	0.98	3	Anfo	0.86	31.6	9	8	2.85	1.27
9	25	7	Hanfo	1.1	94.3	9	8	1.98	0.88	4	Anfo	0.86	42.1	9	8	2.59	1.15
10	25	7	Hanfo	1.1	94.3	9	8	1.98	0.88	4	Anfo	0.86	42.1	9	8	2.59	1.15
TOTALS:		53.0			714.3					33.0			347.7				
Mean:		5.3			71.4			2.19	0.97	3.3			34.8			2.77	1.23
St. Dev:		1.2			16.0			0.16	0.07	0.5			4.8			0.12	0.05
Maximum:		7.0			94.3			2.38	1.06	4.0			42.1			2.85	1.27
Minimum:		4.0			53.9			1.98	0.88	3.0			31.6			2.59	1.15

TABLE 8-12: TECHNICAL DATA FOR BEULAH BLAST VO-9-#2

Site: Beulah Blast Number: VO-9-#2 Date: 8-24-88 Time: 6:13 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.82	60.3	8.0	10	2.04	1.28								
2	15	6	Anfo	0.82	60.3	10.0	10	2.56	1.28								
3	15	6	Anfo	0.82	60.3	8.0	8	2.04	1.02								
4	20	5	Anfo	0.82	50.2	11.5	9	3.12	1.22	3	Anfo	0.82	30.1	11.5	9	3.70	1.45
5	20	5	Anfo	0.82	50.2	11.0	10	2.98	1.36	3	Anfo	0.82	30.1	11.0	10	3.54	1.61
6	15	6	Anfo	0.82	60.3	8.0	10	2.04	1.28								
TOTALS:		34.0			341.6					6.0			60.3				
Mean:		5.7			56.9			2.46	1.24	3.0			30.1			3.62	1.53
St. Dev:		0.5			4.7			0.45	0.10								
Maximum:		6.0			60.3			3.12	1.36	3.0			30.1			3.70	1.61
Minimum:		5.0			50.2			2.04	1.02	3.0			30.1			3.54	1.45

TABLE 8-13: TECHNICAL DATA FOR BEULAH BLAST VO-10

Site: Beulah Blast Number: VO-10 Date: 8-16-88 Time: 11:15 AM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.85	62.5	6.0	9.0	1.51	1.13								
2	13	4	Anfo	0.85	41.7	6.0	9.5	1.73	1.37								
3	11	3	Anfo	0.85	31.2	9.0	12.0	2.86	1.91								
4	10	3	Anfo	0.85	31.2	6.0	11.5	1.91	1.83	3	Anfo	0.85	31.2	6.0	11.5	1.91	1.83
5	14	5	Anfo	0.85	52.1	6.0	9.0	1.61	1.21								
TOTALS:		21.0			218.7					3.0			31.2				
Mean:		4.2			43.7			1.92	1.49								
St. Dev:		1.2			12.1			0.49	0.32								
Maximum:		6.0			62.5			2.86	1.91								
Minimum:		3.0			31.2			1.51	1.13								

TABLE 8-14: TECHNICAL DATA FOR BEULAH BLAST VO-11

Site: Beulah Blast Number: VO-11 Date: 8-30-88 Time: 1:44 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.85	62.5	7	10.00	1.76	1.26								
2	15	6	Anfo	0.85	62.5	7	10.00	1.76	1.26								
3	15	6	Anfo	0.85	62.5	7	12.00	1.76	1.51								
4	20	5	Anfo	0.85	52.1	7	10.00	1.87	1.34	3	Anfo	0.85	31.2	7	10.00	2.22	1.59
5	19	4	Anfo	0.85	41.7	7	12.00	2.02	1.73	3	Anfo	0.85	31.2	7	12.00	2.22	1.91
6	24	5	Anfo	0.85	52.1	7	10.00	1.87	1.34	4	Anfo	0.85	41.7	7	10.00	2.02	1.44
7	19	4	Anfo	0.85	41.7	7	12.00	2.02	1.73	3	Anfo	0.85	31.2	7	12.00	2.22	1.91
8	25	6	Anfo	0.85	62.5	7	10.00	1.76	1.26	4	Anfo	0.85	41.7	7	10.00	2.02	1.44
9	25	6	Anfo	0.85	62.5	7	12.00	1.76	1.51	4	Anfo	0.85	41.7	7	12.00	2.02	1.73
10	25	6	Anfo	0.85	62.5	7	10.00	1.76	1.26	4	Anfo	0.85	41.7	7	10.00	2.02	1.44
11	25	6	Anfo	0.85	62.5	7	12.75	1.76	1.61	4	Anfo	0.85	41.7	7	12.75	2.02	1.84
12	28	5	Hanfo	1.20	73.5	7	10.00	1.67	1.19	3	Anfo	0.85	31.2	7	10.00	2.22	1.59
13	25	6	Anfo	0.85	62.5	7	13.00	1.76	1.64	4	Anfo	0.85	41.7	7	13.00	2.02	1.88
14	28	5	Hanfo	1.20	73.5	7	10.00	1.67	1.19	3	Anfo	0.85	31.2	7	10.00	2.22	1.59
15	30	4	Hanfo	1.20	58.8	7	13.00	1.80	1.67	4	Anfo	0.85	41.7	7	13.00	2.02	1.88
16	25	6	Anfo	0.85	62.5	7	10.00	1.76	1.26	4	Anfo	0.85	41.7	7	10.00	2.02	1.44
17	24	5	Anfo	0.85	52.1	7	12.50	1.87	1.67	4	Anfo	0.85	41.7	7	12.50	2.02	1.80
18	19	4	Anfo	0.85	41.7	7	10.00	2.02	1.44	3	Anfo	0.85	31.2	7	10.00	2.22	1.59
19	19	4	Anfo	0.85	41.7	7	12.00	2.02	1.73	3	Anfo	0.85	31.2	7	12.00	2.22	1.91
20	15	6	Anfo	0.85	62.5	7	10.00	1.76	1.26								
21	15	6	Anfo	0.85	62.5	7	12.00	1.76	1.51								
22	15	6	Anfo	0.85	62.5	7	10.00	1.76	1.26								
TOTALS:		117.0			1278.5					57.0			593.6				
Mean:		5.3			58.1			1.82	1.44	3.6			37.1			2.11	1.69
St. Dev:		0.8			9.3			0.11	0.19	0.5			5.2			0.10	0.18
Maximum:		6.0			73.5			2.02	1.73	4.0			41.7			2.22	1.91
Minimum:		4.0			41.7			1.67	1.19	3.0			31.2			2.02	1.44

TABLE 8-14: CONTINUED

DECK 3							
Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
3	Anfo	0.85	31.2	7	10	2.22	1.59
3	Anfo	0.85	31.2	7	10	2.22	1.59
4	Anfo	0.85	41.7	7	13	2.02	1.88
10.0			104.1				
3.3			34.7			2.16	1.68
0.5			4.9			0.10	0.14
4.0			41.7			2.22	1.88
3.0			31.2			2.02	1.59

TABLE 8-15: TECHNICAL DATA FOR BEULAH BLAST VO-12

Site: Beulah Blast Number: VO-12 Date: 8-31-88 Time: 12:40

Hole Number	Hole Depth	DECK 1 Deck Height	DECK 1 Expl. Type	DECK 1 Density gm/cc	DECK 1 Weight in Deck Lbs.	DECK 1 Burial Ft.	DECK 1 Spacing Ft.	DECK 1 Scaled DOB ft/lb^1/3	DECK 1 Scaled Radius ft/lb^1/3	DECK 2 Deck Height	DECK 2 Expl. Type	DECK 2 Density gm/cc	DECK 2 Weight in Deck Lbs.	DECK 2 Burial Ft.	DECK 2 Spacing Ft.	DECK 2 Scaled DOB ft/lb^1/3	DECK 2 Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.87	64.0	8.0	10	2.00	1.25								
2	19	4	Anfo	0.87	42.6	10.0	10	2.86	1.43	3	Anfo	0.87	32.0	10.0	10	3.15	1.58
3	25	6	Anfo	0.87	64.0	16.5	10	4.13	1.25	4	Anfo	0.87	42.6	16.5	10	4.72	1.43
4	17	3	Anfo	0.87	32.0	21.0	10	6.62	1.58	3	Anfo	0.87	32.0	21.0	10	6.62	1.58
5	20	5	Anfo	0.87	53.3	21.0	10	5.58	1.33	3	Anfo	0.87	32.0	21.0	10	6.62	1.58
6	19	4	Anfo	0.87	42.6	15.5	10	4.44	1.43	3	Anfo	0.87	32.0	15.5	10	4.88	1.58
7	24	6	Anfo	0.87	64.0	7.0	12	1.75	1.50	4	Anfo	0.87	42.6	7.0	12	2.00	1.72
8	17	3	Anfo	0.87	32.0	16.0	10	5.04	1.58	3	Anfo	0.87	32.0	16.0	10	5.04	1.58
9	23	5	Anfo	0.87	53.3	7.0	12	1.86	1.59	4	Anfo	0.87	42.6	7.0	12	2.00	1.72
10	25	6	Anfo	0.87	64.0	15.0	10	3.75	1.25	4	Anfo	0.87	42.6	15.0	10	4.29	1.43
11	24	6	Anfo	0.87	64.0	7.0	12	1.75	1.50	4	Anfo	0.87	42.6	7.0	12	2.00	1.72
12	25	6	Anfo	0.87	64.0	10.0	10	2.50	1.25	4	Anfo	0.87	42.6	10.0	10	2.86	1.43
13	15	6	Anfo	0.87	64.0	8.0	10	2.00	1.25								
TOTALS:		66.0			703.5					39.0			415.7				
Mean:		5.1			54.1			3.41	1.40	3.5			37.8			4.02	1.57
St. Dev:		1.1			12.2			1.58	0.14	0.5			5.3			1.66	0.11
Maximum:		6.0			64.0			6.62	1.59	4.0			42.6			6.62	1.72
Minimum:		3.0			32.0			1.75	1.25	3.0			32.0			2.00	1.43

TABLE 8-16: TECHNICAL DATA FOR BEULAH BLAST VO-13

Site: Beulah Blast Number: VO-13 Date: 8-31-88 Time 5:30 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
2	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
3	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
4	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
5	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
6	19	4	Anfo	0.95	46.6	8	11	2.22	1.53	3	Anfo	0.95	34.9	8	11	2.45	1.68
7	24	6	Hanfo	1.14	83.8	7	10	1.60	1.14	4	Anfo	0.95	46.6	7	10	1.95	1.39
8	24	6	Hanfo	1.14	83.8	7	10	1.60	1.14	4	Anfo	0.95	46.6	7	10	1.95	1.39
9	30	5	Hanfo	1.14	69.8	7	10	1.70	1.21	4	Hanfo	1.14	55.9	7	10	1.83	1.31
10	35	6	Hanfo	1.14	83.8	7	13	1.60	1.49	5	Hanfo	1.14	69.8	7	13	1.70	1.58
11	30	5	Hanfo	1.14	69.8	5	15	1.21	1.82	4	Hanfo	1.14	55.9	5	15	1.31	1.96
12	35	6	Hanfo	1.14	83.8	7	12	1.60	1.37	5	Hanfo	1.14	69.8	7	12	1.70	1.46
13	30	5	Hanfo	1.14	69.8	7	10	1.70	1.21	4	Hanfo	1.14	55.9	7	10	1.83	1.31
14	24	6	Hanfo	1.14	83.8	7	10	1.60	1.14	6	Anfo	0.95	69.8	7	10	1.70	1.21
15	25	6	Hanfo	1.14	83.8	7	10	1.60	1.14	6	Anfo	0.95	69.8	7	10	1.70	1.21
16	18	3	Anfo	0.95	34.9	8	11	2.45	1.68	3	Anfo	0.95	34.9	8	11	2.45	1.68
17	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
18	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
19	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
20	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
21	15	6	Anfo	0.95	69.8	8	12	1.94	1.46								
TOTALS:		118.0			1492.2					48.0			609.9				
Mean:		5.6			71.1			1.82	1.40	4.4			55.4			1.87	1.47
St. Dev:		0.8			11.8			0.26	0.18	1.0			12.9			0.32	0.22
Maximum:		6.0			83.8			2.45	1.82	6.0			69.8			2.45	1.96
Minimum:		3.0			34.9			1.21	1.14	3.0			34.9			1.31	1.21

TABLE 8-16: CONTINUED

Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
DECK 3							
3	Anfo	0.95	34.9	7	10	2.14	1.53
4	Anfo	0.95	46.6	7	13	1.95	1.81
3	Anfo	0.95	34.9	5	15	1.53	2.29
4	Anfo	0.95	46.6	7	12	1.95	1.67
3	Anfo	0.95	34.9	7	10	2.14	1.53
17.0			197.9				
3.4			39.6			1.94	1.77
0.5			5.7			0.22	0.28
4.0			46.6			2.14	2.29
3.0			34.9			1.53	1.53

Deck lengths were taped to insure that the required rise was achieved. Therefore, the explosive decks were quite close to design. There can however be some small errors of measurement which are not recorded. These are estimated to be no more than 0.5 feet.

In many cases a part bag of ANFO or PowerAN was left after the blast was loaded. Therefore, the amount used had to be estimated. Thus, some error could be introduced in this manner.

None of these variances are large. However, taken together these could result in significant variations in loading density from shot to shot. The density variation will affect, to some extent the scaled depth of burial and radius. Since the calculations involve the cube root of the weight these variations will not be major.

8.2.3 Explosive Selection

The primary explosive used was ANFO. Heavy ANFO (HANFO) was used in most bottom decks if the hole contained three decks. For holes of intermediate depth, where two decks were used HANFO was often used in the bottom deck. In other cases ANFO was employed in both decks, either to evaluate the effect of using ANFO rather than PowerAN in the bottom decks or because the hole was near the depth where only one deck would be used rather than two. In short holes, loaded with only one column of explosive, ANFO was always used. This was also true where a second shot was required to complete the filling of the vertical opening.

A primary reason for using heavy ANFO in the lower deck was to guard against water attack. This was important when blasting individual sinkholes because many blastholes were not drilled to void and if there was seepage water could accumulate in the bottom of the hole. Due to exceedingly dry conditions during the time the field work was conducted there was not much problem with water, however, the precaution was generally taken.

The other reason for using heavy HANFO in the lower decks was to provide added energy in a region where the depths of burial could be larger. For sinkholes with sloping sides the burden becomes larger as the holes are deeper. In general one could place the drill 8 to 10 feet from the rim of the opening safely. In some cases though a greater set back was required. To insure good movement of overburden from the bottom out into the vertical opening it was felt that the added energy input would be useful.

All of the second shots were loaded only with ANFO. In addition blasts VO-6, VO-7&8-S, VO-10 and VO-12 were loaded with

ANFO only. Shot VO-11 used HANFO only in the bottom deck of 3 deck holes. The results of the blasts are described in Chapter 4.

The results of test blasts show that ANFO is almost always quite suitable for blasting the overburden around the opening. If water is present then a waterproof product is advised, at least in the lower deck, to guard against poor performance of the explosive.

If a vertical opening has side slopes of 65 degrees or less and the hole depths are greater than 20 feet then the depth of burial becomes large at the bottom of the hole. For ANFO the scaled depths of burial will exceed 4.5 $ft/lb^{1/3}$ if an 8 foot standoff, at the crest, is assumed. Under these circumstances use of a more energetic product is warranted for good displacement. The same conclusion is reached for cases where the set back is large due to considerations of safe drill placement.

Since ANFO is less costly, the use of this product is recommended whenever possible. For AML work explosives are purchased in smaller lots and the price differential can be quite significant. However, when situations, such as those described above, occur more energetic waterproof products can be well utilized.

8.2.4 Analysis of Cratering Data

The tables list the cratering data by deck and the mean values for the given deck in the blast. The scaled depth of burial of the charge and the scaled radius are reported. The method by which these values are calculated are discussed above.

Table 8-17 is a summary of the crater data for each blast at Beulah from VO-3 to VO-13. The mean values for the scaled depth of burial range from 2.10 $ft/lb^{1/3}$ to 2.28 $ft/lb^{1/3}$. Examining individual blasts shows that the scaled depth of burial for deck 2 was invariably greater than deck 1. The reasons are that ANFO was almost always used in deck 2 whereas HANFO was frequently used in deck 1 and a longer column was usually employed for deck 1, in the bottom of the hole, than for subsequent decks.

The different explosive and somewhat longer columns in deck 1 reflect the need to obtain good movement of material in the toe region of the hole. This is necessary to adequately displace overburden to fill the hole and to assist in keeping near surface, more suitable plant growth material on top of the displaced strata.

The blastholes used were 6-inch diameter. For an 8:1 ratio of explosive deck length to diameter a 4 foot length is required. Bottom deck lengths tended to exceed this and were often 5 to 6 feet long. Therefore, these deck charges may have

TABLE 8-17: MEAN SCALED DEPTHS OF BURIAL AND RADIUS FOR BLASTS AT BEULAH

	DECK 1		DECK 2		DECK 3	
Blast Number	Mean Scaled DOB ft/lb^1/3	Mean Scaled Radius ft/lb^1/3	Mean Scaled DOB ft/lb^1/3	Mean Scaled Radius ft/lb^1/3	Mean Scaled DOB ft/lb^1/3	Mean Scaled Radius ft/lb^1/3
VO-3	2.03	1.22	2.82	1.58		
VO-4A	2.24	1.28	2.78	1.59		
VO-4A-#2	2.54	1.59	2.54	1.59		
VO-4B	1.83	1.30	2.04	1.45	1.75	1.46
VO-5	2.11	1.62	2.22	1.64		
VO-6	2.43	1.45	2.79	1.61		
V)-7&8-N	2.00	1.34	2.22	1.48	2.30	1.49
VO-7&8N-#2	2.02	1.26				
VO-7&8-S	2.12	1.43	2.29	1.53	2.33	1.57
VO-7S-#2	3.55	1.51	3.72	1.58		
VO-9	2.19	0.97	2.77	1.23		
VO-9-#2	2.46	1.24	3.62	1.53		
VO-10	1.92	1.49	1.91	1.83	-	
VO-11	1.82	1.44	2.11	1.69	2.16	1.68
VO-12	3.41	1.40	4.02	1.57		
VO-13	1.82	1.40	1.87	1.47	1.94	1.77
Mean:	2.28	1.37	2.65	1.56	2.10	1.59
St. Dev:	0.50	0.16	0.65	0.13	0.22	0.12
Maximum:	3.55	1.62	4.02	1.83	2.33	1.77
Minimum:	1.82	0.97	1.87	1.23	1.75	1.46

been close to performing like linear cratering charges for which square root scaling applies. However, since the variations were only one to two feet these charges have been assumed to act like spherical cratering charges.

These longer decks resulted because experience had indicated that successful cratering of the bottom of the hole required somewhat reduced scaled depths of burial. This had proved especially true for cases where holes were drilled to a void and there was appreciable thickness of coal in the roof.

The other factor was hole depth. Near the depths where a transfer would be made from one to two explosive decks, or from two to three decks the deck lengths would become longer until the transfer point was reached. Due to more restricted movement low in the hole the bottom deck was lengthened first and then adjustments were made to other decks as needed. Thus operational reality often means that one can not adhere exactly to the 8:1 length to diameter ratio usually considered maximum for an explosive charge to act as a spherical point charge.

It is important to state that the scaled depths of burial for the bottom deck are somewhat less accurate than those for other decks. Several of the vertical openings blasted were relatively new and steep sided. In these cases the burden at the crest and toe were quite similar. In other cases there was more variation. The most difficult situations were when the opening was undercut. Sinkholes VO-9 and VO-12 were the most pronounced examples of this situation. In those cases direct measurements could not be made so the burden had to be estimated. The other case where it was difficult to estimate toe burdens was when there was a sloping ramp down to the rooms. Again, direct measurement was not possible and estimates were required.

Reviewing the individual blasts one observes that the third deck had a scaled depth of burial that was almost always the same or slightly greater than the second deck. This reflects the fact that the explosives were the same in these decks and the deck lengths were usually no more than one foot different.

Deck lengths for decks 2 and 3 were commonly within the 8:1 length to diameter ratio. In six holes the second deck was longer than 4 feet. In no case was a third deck longer than 4 feet. Therefore, the second and third decks, almost invariably were expected to function as spherical cratering charges.

For the top deck the potential existed for the charge to crater toward the opening or toward the top surface. It was found that 9 feet of stemming worked well in these holes. Therefore the depth of burial to the center of the top charge was 10.5 to 11 feet. This is quite similar to or less than the depth of burial toward the vertical opening. Thus it could be expected that displacement would be some combination of movement toward the sinkhole and upward. High-speed camera films and field observations of the results generally confirmed this to be the case. In the tables the scaled depth of burial has been reported in the direction of the opening. The scaled depths of burial to the upper surface ranged from 3.1 to 3.2 $ft/lb^{1/3}$. Thus cratering of the top deck toward the opening was often the preferential direction. Movement to the top would be preferred in those cases where there was prior infilling of the opening by sluffed material or where a second row of holes had more restricted movement toward the opening.

Figure 8-1 is a chart showing the mean scaled depths of burial for the blasts tabulated in table 8-17. The blasts are in the same order on the graph as in the table with number 1 representing VO-3 and number 16 representing VO-13.

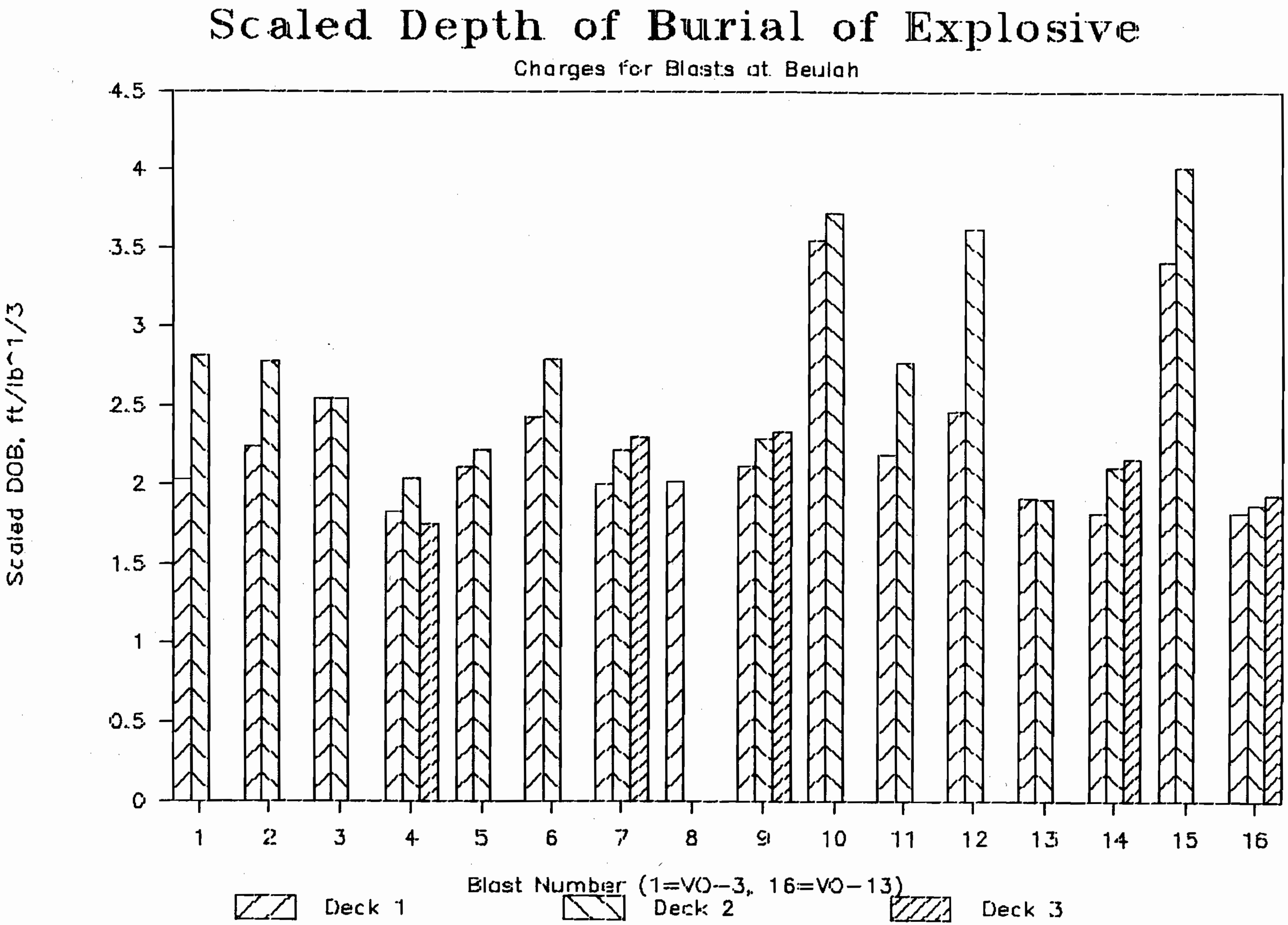
The graph shows that scaled depths of burial were generally greater than 2.0 $ft/lb^{1/3}$. This is consistent with the theory outlined in chapter 2. Three blasts show scaled depths of burial greater than 3.0 $ft/lb^{1/3}$. Two of these (10 and 12) were second shots on a given opening (VO-7-S and VO-9) and hole placement was limited by disruption from the first blast. The third (number 15 = VO-12) was an irregularly shaped opening with considerable undercutting and some holes had to be placed a long distance from the opening. While the results of these shots proved satisfactory depths of burial above 3.0 $ft/lb^{1/3}$ are not advised unless field circumstances require.

The graph shows the trend of scaled depth of burial to greater values in the upper decks. It also shows there is variance from shot to shot related to changing field conditions. In most cases the variations are modest, except for the three blasts discussed above.

Based on the blast results and the data computed in the tables it appears that scaled depth of burial should be in the range of 2.0 to 2.6 $ft/lb^{1/3}$. This results in good displacement of the overburden and causes sufficient fragmentation and swell to provide good infilling of the individual vertical opening. It is advised that the lowest deck have a scaled depth of burial of not more than 2.3 $ft/lb^{1/3}$ because of the decreased freedom to displace. The scaled depth of burial, of the upper deck, to the top surface should not be less than the scaled depth of burial, of that deck, to the vertical opening. Otherwise, there will be a tendency to displace material vertically and not cast it over into the void.

In Chapter 2 it was indicated that the optimum scaled radius was likely to be in the range of 2.0 to 2.5 $ft/lb^{1/3}$. In order to allow good overlap between craters it was planned to use spacings between holes of 8 to 10 feet. This corresponds to radii of 4 to 5 feet. When compared to optimum radii of 7 to

FIGURE 8-1


Scaled Depth of Burial of Explosive
Charges for Blasts at Beulah
Scaled DOB, ft/lb^1/3
4.5
4
3.5
3
2.5
2
1.5
1
0.5
0
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
Blast Number (1=VO-3, 16=VO-13)
Deck 1
Deck 2
Deck 3

8.7 feet the potential for overlap, fragmentation and loosening of the overburden appeared quite good.

Review of the individual blast tables shows that the proposed range of spacings were largely followed. However, there were 25 holes that had 12-foot spacings and fewer that carried other spacings ranging from 11 to 15 feet.

Table 8-17 gives the mean scaled radii for the different shots. In general these were less than 2.0 ft/lb$^{1/3}$. Given the desire to have overlap between the craters, so as not to isolate ledges of material between holes these values were expected. The value of 0.97 ft/lb$^{1/3}$ calculated for deck 1 of shot VO-9 was somewhat anomalous and resulted from the need to get an adequate number of holes around this opening, which was of very small diameter at the surface and of much larger diameter at its base.

Except for VO-9 the scaled radius ranged from 1.23 ft/lb$^{1/3}$ to 1.83 ft/lb$^{1/3}$. From the individual tables and the summary table it can be seen that the standard deviations indicate a close distribution about the mean and a narrower distribution than was true for the scaled depths of burial. This reflects the fact that in field practice it was often easier to layout uniform spacings than burdens, due to the nature of the vertical openings and the need for safety of personnel and equipment.

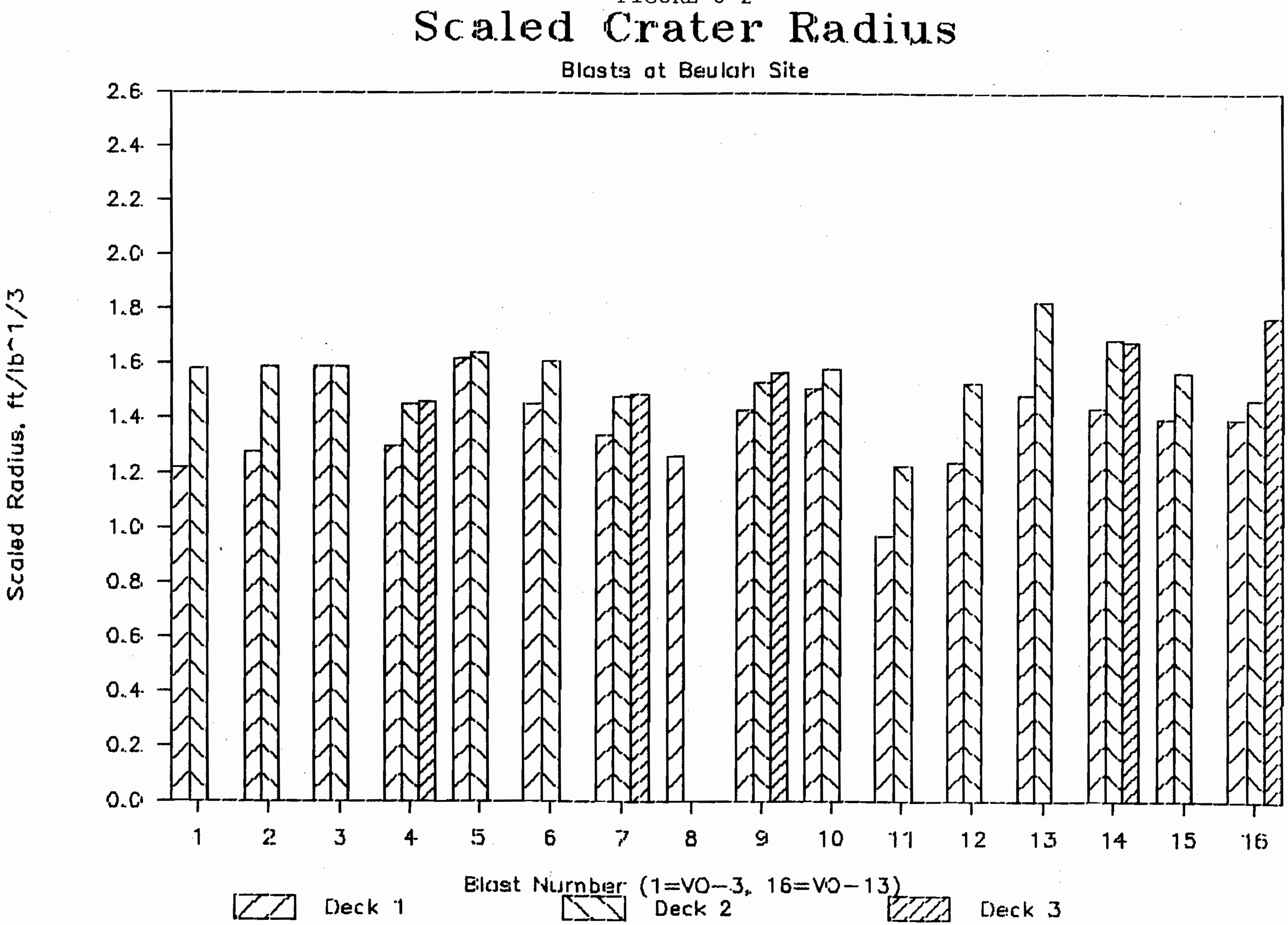
Based on field observations of the results spacings between the holes of 12 feet appear suitable. This value yields a scaled radius of 1.75 ft/lb$^{1/3}$ in a 4-foot deck of ANFO. For a 3-foot deck of ANFO the scaled radius is 1.91 ft/lb$^{1/3}$ which will generally be adequate for decks higher in the hole.

Figure 8-2 is a plot of the mean scaled crater radius by deck for each shot. It can be seen that these values are generally quite uniform between shots. The trend for explosive decks higher in the hole to have higher scaled radius is also observed. Often, in a hole with 3 decks, the middle and top explosive decks have almost the same scaled radius. This is not so in shot 16 (VO-13) because added amounts of PowerAN were used in deck 2.

Based on field observation of the results and the data analysis a pattern of 10 feet on burden by 12 feet on spacing is expected to provide good results for blasting in individual vertical openings in the overconsolidated clays associated with many abandoned lignite mining operations. There will be variations to the pattern, of course, based on the need to place blastholes around what can often be a quite irregular opening.

Powder factor is not the governing principle for work of this type. Also, powder factor can vary significantly depending

FIGURE 8-2


Scaled Crater Radius
Blasts at Beulah Site
Scaled Radius, ft/lb^1/3
2.6
2.4
2.2
2.0
1.8
1.6
1.4
1.2
1.0
0.8
0.6
0.4
0.2
0.0
1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
Blast Number (1=VO-3, 16=VO-13)
Deck 1
Deck 2
Deck 3

on the depth to which the blast holes must be drilled. However, as a guideline, the suggested pattern with two decks and a twenty-five foot hole depth would yield a powder factor of 0.93 pounds per cubic yard of overburden. This value is not dissimilar to powder factors used in blast casting in dragline mines and indicates that vertical openings can successfully be closed with energy inputs similar to those experienced in blast casting shots.

8.3 BLAST DELAY METHODS

Two principles associated with these blasts were to maximize the relief to allow good displacement of material into the vertical opening and to minimize the vibration as much as possible, not only to avoid distress to nearby neighbors, but also to minimize the disturbance felt at surrounding uncollapsed rooms and crosscuts. The intent was to avoid unwanted collapse.

To achieve this purpose each explosive deck in a blast was independently delayed. A combination of surface and down-the-hole millisecond delay systems were used. The typical combination was 42 millisecond surface delays between holes and down-the-hole delays that yielded 50 ms of delay between decks. Down-the-hole delays beginning at period 4 (100 ms) were used and every other period was employed to obtain the desired 50 ms delay. In some cases adjacent periods were used between a second and third deck because of the reduced restriction in the upper areas of the bank and to avoid overlaps when a blast contained holes with different numbers of decks. One blast (VO-4B) was detonated with all adjacent periods. Four blasts were shot with adjacent periods between the second and third decks. The remaining 13 blasts employed 50 milliseconds between all decks.

For these blasts the bottom deck was shot first followed by the upper decks in succession. The intent was to throw overburden laterally keeping the better plant growth material near the top. Observations after each blast indicated that this is largely what happened. There was some mixing of strata due to the disruption but much of the soil materials remained near the top. Therefore, in many cases, topsoil would not have to be removed before blasting begins. Figure 8-3 illustrates growth material at the top of blasted overburden.

The surface delays were Nonel noiseless trunkline delays. These were used for the reduced noise generated and simplicity of use. Down-the-hole, HD Primadets were used with slider primers and RX Primaline. The system was chosen because it offers an easily used system when decked holes are employed.

Blasting results indicated that this method worked well. Good displacement and in-filling of the vertical openings was usually obtained. Upper overburden was generally located in the top region of the blasted material. There were no interruptions of the blast on surface. Also, there was no evidence of

FIGURE 8-3: AN EXAMPLE OF A FILLED VERTICAL OPENING AT BEULAH SHOWING BETTER GROWTH MEDIUM REMAINING IN THE TOP REGION OF THE BLASTED MATERIAL

misfires. Misfire possibility could not be fully assessed, however, because the blasted material is not excavated.

When a blast contains holes of different depths, and different numbers of decks per hole, care must be taken in designing the delay system to avoid overlaps for vibration reasons and to avoid out of rotation detonation. Figure 8-4 illustrates this for a blast with one, two and three decks per hole. Different sequences may be necessary down-the-hole to generate the best result.

8.4 EXPLOSIVE DECKING SYSTEM

Results of the blasts at Beulah have shown that the explosive decking system works well. The theoretical basis for employing the method was described in Chapter 2. The results have confirmed the validity of this approach.

The primary purpose has been to cause material to crater toward the vertical opening, or, in cases when a void was struck to crater down into the void as well as to the side. The crater related results and scaled burial and radius have been discussed above.

Use of the deck system led to modest levels of vibration. Therefore, use of this blasting method is possible closer to built up areas than would be the case if a continuous column and larger patterns were used, that generated considerably higher vibration levels.

This approach also kept vibration lower at adjacent workings. This is very important because high vibration could cause unwanted collapse in areas near the blast. Based on the graphs in Chapter 7 vibration can still be quite high at a distance of 25 ft. (>10 ins/sec) but the frequencies would also be high at this close distance. Field results showed that with low powder weights in multiple decks immediate collapse of areas beyond the blast did not occur.

It is somewhat more difficult to assess what will happen over a longer period of time. In other words, the question arises whether blasting individual sinkholes could result in additional subsidence at a future time.

Review of the site in January 1990 suggested this did not generally happen. However, there were new, small openings in the vicinity of VO-7 and 8 and VO-9. Since the area has actively been caving for several years it is not clear whether the current collapse is much related to the blasting. However, the possibility cannot be ruled out.

There were a few other new vertical openings observed in January as well. These were well removed from any sinkholes blasted in the test program. The seismograph studies discussed in Chapter 7 suggests that vibration levels would have been

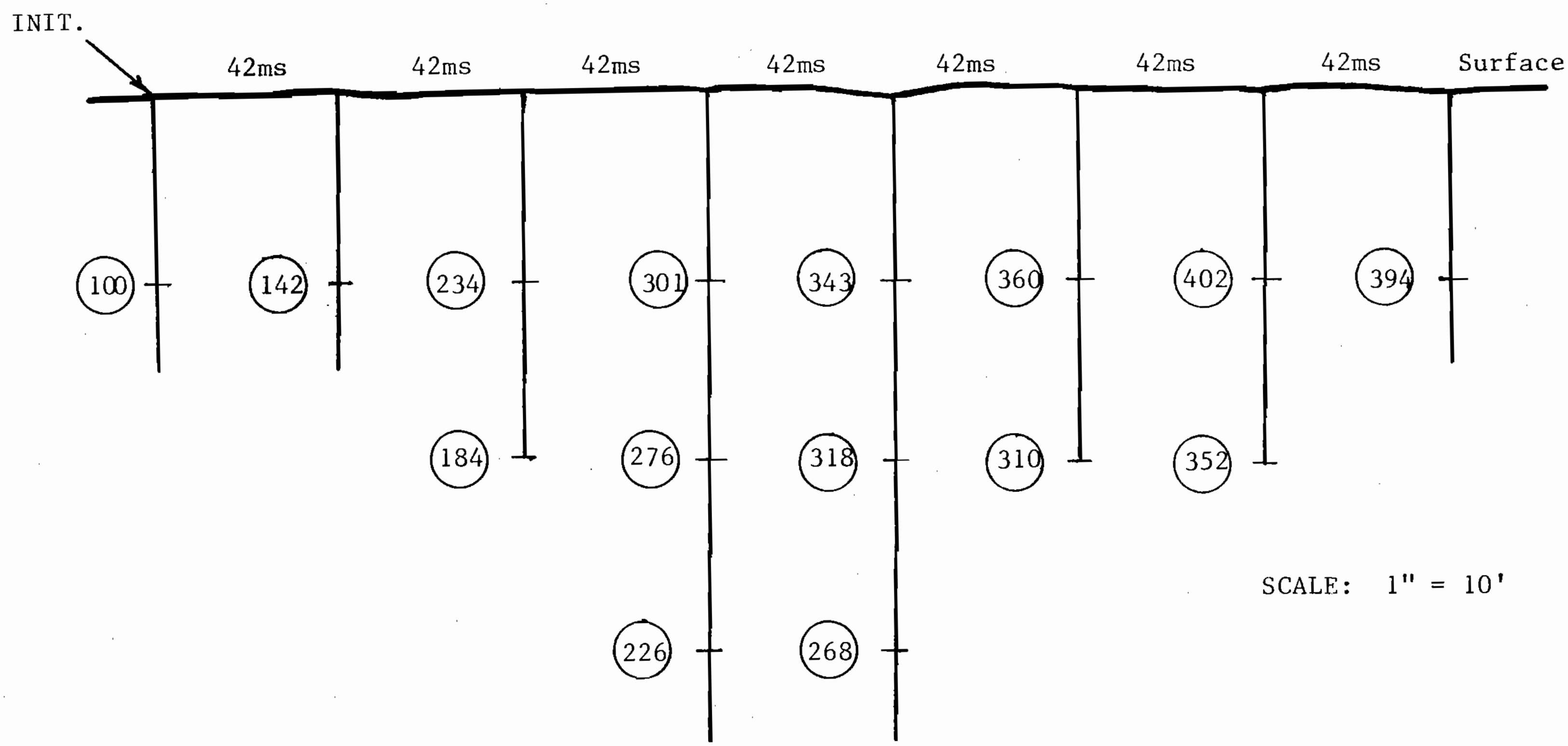


FIGURE 8-4: FIRING SEQUENCE FOR A BLAST AROUND A VERTICAL OPENING HAVING BLASTHOLES WITH ONE, TWO AND THREE EXPLOSIVE DECKS

quite modest at these locations. Therefore, it is most likely that these features represent ongoing collapse in the area.

In addition to the technical reasons for using this technique, and the opportunity to reduce vibration, decked loading also helps to control costs. If a continuous column is used powder factors will increase because one cannot pull the patterns out enough to compensate for the added explosive, and still get adequate displacement toward the sinkhole. This is especially true when drilling can be done at low cost. In these materials one would not expect the drilling cost to go beyond $1.50 per foot so the best cost can be obtained, in most cases by decking and using somewhat reduced pattern dimensions.

It has been observed, in this research, that a decked loading system is applicable when shooting vertical openings. The system, therefore, has general application when performing AML reclamation by blasting.

CHAPTER 9
ANALYSIS OF BLASTING RESULTS AT THE WHITE TEST SITE

9.1 GENERAL DISCUSSION

In general, the test blasts at the White Test Site were successful. In two cases a second shot was needed to fully collapse the workings. The potential for unwanted collapse, that interfered with the next shot was significant. This resulted in the redesign of two blasts after the prior shot was detonated. The research allowed a good assessment of the needs for blasting irregular works for which there was little historical data concerning the mine layout

The methods used for finding the old workings consisted primarily of exploration drilling and T.V. camera studies using a down hole T.V. system. The techniques have been described in Chapter 3. The principle conclusion from the research at this location is that identifying the old works is time consuming and costly when the mining was not systematic, and, there is little documentation available upon which planning and blasthole layout can be based.

The problem is most acute when the attempt is made to blast in individual rooms using blastholes centered over the abandoned workings. For this technique it is important to have an exact indication of the room locations. Since the rooms were not advanced in a straight line, close exploration hole spacings were necessary to define the opening and the T.V. camera often could not function over longer distances. Nearly as many exploration holes were drilled as production holes.

The conclusion is that identifying the workings is the major difficulty when blasting sites similar to the White site. The result is a costly exploration program. The cost can, however, be reduced if area blasting is employed.

The nature of this test location led to the investigation of several techniques for filling the workings being tested and evaluated. The primary methods were blasting individual rooms and area blasting using a regular pattern. In both methods techniques regarding explosive loads and placement were studied.

Maps of the workings were shown in Chapter 3. It can be seen that workings and pillars were irregular in shape and placement. Large, poorly supported intersections often resulted. Shots B1 and B2 showed that considerable disruption beyond the shot was possible. This can leave areas, that are not fully collapsed, difficult to access for the next shot. Therefore, variations in loading and placement of explosive was tried to eliminate the problem. Also, subsequent reclamation can be minimized if the surface displacement is small.

Increasing the depth of burial to the upper surface by adding more stemming achieves reduced disruption. Considerable success was had with this approach.

Where ill-defined mines are found a considerable amount of field judgement is needed in blast design and implementation. This includes the hole layout and loading requirements. Explosive deck lengths and number must vary according to the hole depth. Less judgement is required when an area shot, on a regular pattern, is employed. In the latter case the primary decision is the hole by hole loading requirements.

Little difficulty was had maneuvering equipment safely in the work area. There were two sinkholes on the property, otherwise there had not been much subsidence on the one acre site. Voids were often small in height which reduces surface disturbance. There were no incidences of equipment being involved in any sudden caving.

Some holes drilled quite short to a void. Depths were in the range of ten to fifteen feet. The drill was moved away from such holes when encountered. There was no problem of collapse under the drill when these circumstances prevailed.

At the White test area topsoil was removed prior to blasting since the area supported farming operations. It was intended, by the State to regrade and replace the topsoil after the work was completed. In this way crop yields could be insured.

Blasting at this location produced only shallow depressions or modest swelling in the blasted areas. It is estimated that regrading could be completed in one day. Replacing the topsoil and seeding would require approximately one day of work. Thus the site could be fully restored in about two days.

Contributing to the reduced surface disturbance was adjustments made to the stemming heights. This is discussed below. It does appear that, in some cases, blasting without prior topsoil removal would be quite possible. Also increased stemming reduces the surface cracking around the shot, thereby reducing hazards in ungraded areas.

9.2 BLAST DATA ANALYSIS

9.2.1 Overview

At the White Site there were eight blasts detonated in the test program. Blast B1 closed the adit, the entry leading into the mine from the adit and the first rooms. Blasts B2 to B5 were subsequent blasts designed to close specific rooms by cratering down into the works. Variations in powder loads and stemming were made to try to optimize results. T.V. camera setups were used to evaluate performance.

Blasts B6 to B8 were area patterns. Shot B6 was divided into a 10x10 foot pattern and 12x12 foot pattern. Blast B7 and B8 both used a 15x15 foot pattern. Shot B8 was a large pattern of 65 blastholes. These shots were in an area of the site where there was considerable water. Special precautions had to be taken to protect explosives from water attack.

Information concerning the blasts has been tabulated in a similar fashion to that provided in Chapter 8 for the Beulah site. Information is provided in table 9-1 to 9-7 concerning individual blasts.

In these tables the burial is the depth from the void to the center of the charge for the first deck. For subsequent decks it is the distance from the top of the explosive deck below to the center of the charge above. Holes placed above the open void will crater down to the workings, which is the only direction in which there is freedom for the material to move.

The spacing is the distance between holes, as was the case for the Beulah site data. The scaled radius is based on one-half the spacing divided by the cube root of the weight.

Blasts B6 and B7 were area blasts, consequently not all blastholes drilled into a void. Where a void was not struck this is noted in the table. In these holes a burial depth is not provided. Cratering would be to the side into the void and burdens cannot be exactly stated. Therefore, it was deemed better to exclude these results than to introduce errors through estimation. Holes that did strike a void, however, have the depth of burial provided.

The scaled radius remains related to the spacing between holes. Therefore, spacings are given for each hole. These are usually the same for each blasthole, since a regular pattern was used.

Most blasts contained two decks in at least some holes. Shot B7 had three decks. Blast B4, on the other hand, had only one deck.

A table was not prepared for Blast B8. This was a large area shot in which about one-half the holes were in voids and the remaining half were in pillars. It was difficult to put the data into the same format as for the other shots. A summary of data for this blast is provided in table 9-9.

TABLE 9-1: TECHNICAL DATA FOR BLAST B1 AT THE WHITE TEST SITE

Site: White Blast Number: B1 Date: 9-12-88 Time: 12:42

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
1	15	6	Anfo	0.94	69.1	8.0	10	1.95	1.22								
2	15	6	Anfo	0.94	69.1	8.0	10	1.95	1.22								
3	14	5	Anfo	0.94	57.6	8.0	10	2.07	1.29								
4	15	6	Anfo	0.94	69.1	8.0	10	1.95	1.22								
5	15	6	Anfo	0.94	69.1	8.0	10	1.95	1.22								
6	16	4	Anfo	0.94	46.1	5.0	9	1.39	1.26								
7	18	4	Anfo	0.94	46.1	6.0	9	1.67	1.26								
8	19	3	Hanfo	1.25	45.9	3.5	9	0.98	1.26	3	Anfo	0.94	34.6	3.5	9	1.07	1.38
9	23	3	Hanfo	1.25	45.9	6.5	9	1.82	1.26	3	Anfo	0.94	34.6	4.5	9	1.38	1.38
10	20	3	Hanfo	1.25	45.9	3.5	9	0.98	1.26	3	Anfo	0.94	34.6	4.5	9	1.38	1.38
11	20	3	Hanfo	1.25	45.9	3.5	9	0.98	1.26	3	Anfo	0.94	34.6	4.5	9	1.38	1.38
12	22	3	Hanfo	1.25	45.9	5.5	9	1.54	1.26	3	Anfo	0.94	34.6	4.5	9	1.38	1.38
TOTALS:		52.0			655.9					15.0			172.8				
Mean:		4.3			54.7			1.60	1.25	3.0			34.6			1.32	1.38
St. Dev:		1.3			10.7			0.41	0.02	0.0			0.0			0.12	0.00
Maximum:		6.0			69.1			2.07	1.29	3.0			34.6			1.38	1.38
Minimum:		3.0			45.9			0.98	1.22	3.0			34.6			1.07	1.38

TABLE 9-2: TECHNICAL DATA FOR BLAST B2 AT THE WHITE TEST SITE

Site: White Blast Number: B2 Date: 9-13-88 Time: 3:57 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
13	26	4	Hanfo	1.17	57.3	5	10	1.30	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
14	26	4	Hanfo	1.17	57.3	3	10	0.78	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
15	25	4	Hanfo	1.17	57.3	3	10	0.78	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
16	26	4	Hanfo	1.17	57.3	3	10	0.78	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
17	26	4	Hanfo	1.17	57.3	4	22	1.04	2.85	4	Anfo	0.94	46.1	7.0	22	1.95	3.07
18	25	4	Hanfo	1.17	57.3	3	10	0.78	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
19	24	4	Hanfo	1.17	57.3	4	10	1.04	1.30	3	Anfo	0.94	34.6	5.5	10	1.69	1.54
20	28	6	Hanfo	1.17	86.0		10		1.13	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
21	26	4	Hanfo	1.17	57.3	5	22	1.30	2.85	4	Anfo	0.94	46.1	7.0	22	1.95	3.07
22	25	6	Hanfo	1.17	86.0		10		1.13	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
23	23	4	Hanfo	1.17	57.3	4	10	1.04	1.30	4	Anfo	0.94	46.1	7.0	10	1.95	1.39
TOTALS:		48.0			688.1					43.0			495.2				
Mean:		4.4			62.6			0.96	1.55	3.9			45.0			1.93	1.71
St. Dev:		0.8			11.1			0.20	0.62	0.3			3.3			0.08	0.64
Maximum:		6.0			86.0			1.30	2.85	4.0			46.1			1.95	3.07
Minimum:		4.0			57.3			0.78	1.13	3.0			34.6			1.69	1.39

TABLE 9-3: TECHNICAL DATA FOR BLAST B3 AT THE WHITE TEST SITE

Site: White Blast Number: B3 Date: 9-15-88 Time: 4:19 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
25	28	3	Hanfo	1.08	39.7	7.5	10	2.20	1.47	3	Anfo	0.87	32.0	5.5	10	1.73	1.58
26	28	3	Hanfo	1.08	39.7	6.5	10	1.91	1.47	3	Anfo	0.87	32.0	6.5	10	2.05	1.58
27	30	4	Hanfo	1.08	52.9	8.0	10	2.13	1.33	4	Anfo	0.87	42.6	6	10	1.72	1.43
28	lost																
29	lost																
30	26	3	Hanfo	1.08	39.7	6.5	10	1.91	1.47	3	Anfo	0.87	32.0	7.5	10	2.36	1.58
31	24	3	Hanfo	1.08	39.7	5.5	10	1.61	1.47	3	Anfo	0.87	32.0	6.5	10	2.05	1.58
32	22	3	Hanfo	1.08	39.7	3.0	10	0.88	1.47	3	Anfo	0.87	32.0	7.5	10	2.36	1.58
33	30	4	Hanfo	1.08	52.9	6.0	10	1.60	1.33	4	Anfo	0.87	42.6	6	10	1.72	1.43
34	26	3	Hanfo	1.08	39.7	6.5	12	1.91	1.76	3	Anfo	0.87	32.0	7.5	12	2.36	1.89
35	20	4	Hanfo	1.08	52.9	7.0	12	1.86	1.60	3	Anfo	0.87	32.0	4.5	12	1.42	1.89
36	21	2	Hanfo	1.08	26.5	5.0	10	1.68	1.68	3	Anfo	0.87	32.0	4.5	10	1.42	1.58
37	13	3	Anfo	0.87	32.0	2.5	10	0.79	1.58								
38	15	4	Anfo	0.87	42.6	3.5	10	1.00	1.43								
39	20	2	Hanfo	1.08	26.5	4.0	10	1.34	1.68	3	Anfo	0.87	32.0	4.5	10	1.42	1.58
TOTALS:		41.0			524.5					35.0			373.1				
Mean:		3.2			40.3			1.60	1.52	3.2			33.9			1.87	1.61
St. Dev:		0.7			8.5			0.45	0.13	0.4			4.1			0.37	0.14
Maximum:		4.0			52.9			2.20	1.76	4.0			42.6			2.36	1.89
Minimum:		2.0			26.5			0.79	1.33	3.0			32.0			1.42	1.43

TABLE 9-4: TECHNICAL DATA FOR BLAST B4 AT THE WHITE TEST SITE

Site: White Blast Number: B4 Date: 9-16-88 Time: 3:09 PM

		DECK 1							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
40	27	3	Hanfo	1.22	44.8	6.5	10	1.83	1.41
41	27	3	Hanfo	1.22	44.8	6.5	10	1.83	1.41
42	27	3	Hanfo	1.22	44.8	6.5	10	1.83	1.41
43	24	3	Hanfo	1.22	44.8	6.5	10	1.83	1.41
44	24	3	Hanfo	1.22	44.8	6.5	10	1.83	1.41
TOTALS:		15.0			224.2				
Mean:		3.0			44.8			1.83	1.41
St. Dev:		0.0			0.0			0.00	0.00
Maximum:		3.0			44.8			1.83	1.41
Minimum:		3.0			44.8			1.83	1.41

TABLE 9-5: TECHNICAL DATA FOR BLAST B5 AT THE WHITE TEST SITE

Site: White Blast Number: B5 Date: 9-17-88 Time: 12:49PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
45	33	4	Hanfo	1.23	60.3	7	10	1.79	1.28	3	Anfo	0.9	33.1	6.5	10	2.03	1.56
46	33	4	Hanfo	1.23	60.3	7	10	1.79	1.28	3	Anfo	0.9	33.1	6.5	10	2.03	1.56
47	34	4	Hanfo	1.23	60.3	7	10	1.79	1.28	3	Anfo	0.9	33.1	6.5	10	2.03	1.56
TOTALS:		12.0			180.8					9.0			99.2				
Mean:		4.0			60.3			1.79	1.28	3.0			33.1			2.03	1.56
St. Dev:		0.0			0.0			0.00	0.00	0.0			0.0			0.00	0.00
Maximum:		4.0			60.3			1.79	1.28	3.0			33.1			2.03	1.56
Minimum:		4.0			60.3			1.79	1.28	3.0			33.1			2.03	1.56

TABLE 9-6: TECHNICAL DATA FOR BLAST B6 AT THE WHITE TEST SITE

Site: White Blast Number: B6 Date: 9-19-88 Time: 2:51 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
48	32	5	Hanfo	1.24	76.0	7.5	10	1.77	1.18								
49	17	3	Anfo	0.91	33.4	3.5	10	1.09	1.55								
50	17	3	Anfo	0.91	33.4	3.5	10	1.09	1.55								
51	10	1.1	Anfo	0.91	12.3	1.9	10	0.82	2.17								
52nv	33	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
53nv	35	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
54nv	28	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
55nv	36	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
56nv	38	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
57nv	39	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
58nv	39	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
59	36	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	10.5	10	3.26	1.55
60	34	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	8.5	10	2.64	1.55
61nv	32	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
62nv	21	3	Hanfo	1.24	45.6		10		1.40	2	Anfo	0.91	22.3		10		1.78
63	32	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
64	34	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
65	24	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	5.5	10	1.71	1.55
66nv	35	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
67nv	39	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
68	36	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	10.5	10	3.26	1.55
69nv	38	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
70	33	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
71	35	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
72nv	32	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
73	33	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
74nv	36	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
75	22	3	Hanfo	1.24	45.6	5.5	10	1.54	1.40	3	Anfo	0.91	33.4	4.5	10	1.40	1.55
76	33	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
77nv	37	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
78	32	3	Hanfo	1.24	45.6	6.5	10	1.82	1.40	3	Anfo	0.91	33.4	6.5	10	2.02	1.55
79nv	39	3	Hanfo	1.24	45.6		10		1.40	3	Anfo	0.91	33.4		10		1.55
80	31	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
81	32	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
82nv	26	3	Hanfo	1.24	45.6		12		1.68	3	Anfo	0.91	33.4		12		1.86
83	29	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
84	32	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
85	32	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
86	29	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
87	27	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	5.5	12	1.71	1.86
88nv	21	3	Hanfo	1.24	45.6		12		1.68	3	Anfo	0.91	33.4		12		1.86
89	31	3	Hanfo	1.24	45.6	6.5	12	1.82	1.68	3	Anfo	0.91	33.4	6.5	12	2.02	1.86
90	15	3	Anfo	0.91	33.4	2.5	12	0.78	1.86				0.0				
91	29	4	Hanfo	1.24	60.8	7.5	12	1.91	1.53	3	Anfo	0.91	33.4	5.5	12	1.71	1.86
92	22	5	Hanfo	1.24	76.0	3.5	12	0.83	1.42								
93nv	35	3	Hanfo	1.24	45.6		12		1.68	3	Anfo	0.91	33.4		12		1.86
TOTALS:		141.1			2102.9					119.0			1326.8				
Mean:		3.1			45.7			1.64	1.50	3.0			32.4			2.09	1.65
St. Dev:		0.5			8.9			0.35	0.17	0.2			5.4			0.44	0.14
Maximum:		5.0			76.0			1.91	2.17	3.0			33.4			3.26	1.86
Minimum:		1.1			12.3			0.78	1.18	2.0			0.0			1.40	1.55

TABLE 9-7: TECHNICAL DATA FOR BLAST B7 AT THE WHITE TEST SITE

Site: White Blast Number: B7 Date: 9-22-88 Time: 1:50 PM

		DECK 1								DECK 2							
Hole Number	Hole Depth	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3	Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
94	36	4.0	Hanfo	1.04	51.0	3.0	15	0.81	2.02	3	Anfo	0.80	29.4	7.5	15	2.43	2.43
95nv	33	3.0	Hanfo	1.04	38.2		15		2.23	2	Anfo	0.80	19.6		15		2.78
96nv	38	5.0	Hanfo	1.04	63.7		15		1.86	3	Anfo	0.80	29.4		15		2.43
97nv	39	4.0	Hanfo	1.04	51.0		15		2.02	3	Anfo	0.80	29.4		15		2.43
98nv	37	2.5	Hanfo	1.04	31.9		15		2.37	3	Anfo	0.80	29.4		15		2.43
99nv	35	3.0	Hanfo	1.04	38.2		15		2.23	3	Anfo	0.80	29.4		15		2.43
100nv	37	3.5	Hanfo	1.04	44.6		15		2.12	3	Anfo	0.80	29.4		15		2.43
101	36	4.0	Hanfo	1.04	51.0	3.0	15	0.81	2.02	3	Anfo	0.80	29.4	6.5	15	2.11	2.43
102	31	3.0	Hanfo	1.04	38.2	5.5	15	1.63	2.23	3	Anfo	0.80	29.4	5.5	15	1.78	2.43
103	15	3.0	Anfo	0.80	29.4	3.0	15	0.97	2.43								
104nv	34	3.0	Hanfo	1.04	38.2		15		2.23	3	Anfo	0.80	29.4		15		2.43
105nv	36	3.0	Hanfo	1.04	38.2		15		2.23	3	Anfo	0.80	29.4		15		2.43
106	34	3.0	Hanfo	1.04	38.2	6.5	15	1.93	2.23	3	Anfo	0.80	29.4	6.5	15	2.11	2.43
107nv	39	5.0	Hanfo	1.04	63.7		15		1.86	3	Anfo	0.80	29.4		15		2.43
108nv	38	4.0	Hanfo	1.04	51.0		15		2.02	3	Anfo	0.80	29.4		15		2.43
109nv	39	4.0	Hanfo	1.04	51.0		15		2.02	3	Anfo	0.80	29.4		15		2.43
110	36	3.0	Hanfo	1.04	38.2	6.5	15	2.52	2.23	3	Anfo	0.80	29.4	5.5	15	1.78	2.43
111nv	39	6.0	Hanfo	1.04	76.5		15		1.77	3	Anfo	0.80	29.4		15		2.43
112nv	38	3.5	Hanfo	1.04	44.6		15		2.12	3	Anfo	0.80	29.4		15		2.43
113	34	2.0	Hanfo	1.04	25.5	4.0	15	1.36	2.56	3	Anfo	0.80	29.4	5.5	15	1.78	2.43
TOTALS:		71.5			902.3					56.0			548.9				
Mean:		3.6			45.1			1.43	2.14	2.9			28.9			2.00	2.45
St. Dev:		0.9			12.2			0.59	0.19	0.2			2.2			0.24	0.08
Maximum:		6.0			76.5			2.52	2.55	3.0			29.4			2.43	2.78
Minimum:		2.0			25.5			0.81	1.77	2.0			19.6			1.78	2.43

TABLE 3-7: CONTINUED

DECK 3							
Deck Height	Expl. Type	Density gm/cc	Weight in Deck Lbs.	Burial Ft.	Spacing Ft.	Scaled DOB ft/lb^1/3	Scaled Radius ft/lb^1/3
2	Anfo	0.80	19.6	5	15	1.85	2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6	6	15	2.23	2.78
2	Anfo	0.80	19.6	4	15	1.48	2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6	6	15	2.23	2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6	6	15	2.23	2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6		15		2.78
2	Anfo	0.80	19.6	6	15	2.23	2.78
38.0			372.5				
2.0			19.6			2.04	2.78
0.0			0.0			0.28	0.00
2.0			19.6			2.23	2.78
2.0			19.6			1.48	2.78

9.2.2 Cratering Data

The mean scaled depth of burial, for holes cratering to the void below is calculated for each blast. So, too, is the scaled radius. Mean explosive deck heights and powder weights are also provided for each shot.

The tables show that, at the White location, explosive deck lengths were typically within the 8:1 length to diameter ratio for spherical cratering charges, which is a maximum of 4 feet for a 6-inch blasthole. A few decks were longer due to the need to fit the explosives into the given column length. The situation occurred in the bottom decks and primarily in the first two shots. Upper decks invariably remained within the 8:1 criterion.

Explosive loads were greater in the first deck than in the upper decks. This was due to heavy ANFO commonly being the explosive in the lower decks, whereas ANFO was used in the upper decks.

Table 9-8 is a summary of the mean scaled depths of burial and scaled radius for shots B1 to B7. The values are given by deck. For deck 1 the values ranged from 1.60 $ft/lb^{1/3}$ to 1.83 $ft/lb^{1/3}$ except for blast B2 which had low burial depths in the first deck and a scaled burial of 0.78 $ft/lb^{1/3}$. The second deck generally had scaled depths of burial greater than those in the bottom deck. The exception was blast B1 where the blastholes were just at the beginning of depths requiring a second deck and, therefore, the full burial could not be obtained. In B7 where there was a third deck it also had a higher scaled depth of burial than deck 1.

The mean scaled depths of burial for deck 2 ranged from 1.32 $ft/lb^{1/3}$ (B1) to 2.09 $ft/lb^{1/3}$ and averaged 1.87 $ft/lb^{1/3}$. The mean scaled depth of burial for the third deck in blast B7 was 2.04 $ft/lb^{1/3}$.

The values of scaled radius were similar to those for the blasts at the Beulah site. Blast B7, however, had higher scaled radius because a 15x15 foot pattern was used. For this shot the scaled radius ranged from 2.14 $ft/lb^{1/3}$ in the bottom deck to 2.78 $ft/lb^{1/3}$ in the top decks.

The results show that the mean scaled radius, for a given blast was lowest in deck 1 and increased in deck 2 and also in deck 3 in the case of blast B7. The increase was due to ANFO being used in the upper decks rather than PowerAN and the upper deck lengths often being shorter than deck 1.

Figure 9-1 shows the trend in scaled depth of burial for shots at the White site. Figure 9-2 illustrates the trend in scaled radius.

TABLE 9-8: MEAN SCALED DEPTHS OF BURIAL AND RADIUS FOR BLASTS AT THE WHITE TEST SITE

Blast Number	DECK 1 Mean Scaled DOB ft/lb^1/3	DECK 1 Mean Scaled Radius ft/lb^1/3	DECK 2 Mean Scaled DOB ft/lb^1/3	DECK 2 Mean Scaled Radius ft/lb^1/3	DECK 3 Mean Scaled DOB ft/lb^1/3	DECK 3 Mean Scaled Radius ft/lb^1/3
B1	1.60	1.25	1.32	1.38		
B2	0.98	1.55	1.93	1.71		
B3	1.60	1.52	1.87	1.61		
B4	1.83	1.41				
B5	1.79	1.28	2.03	1.56		
B6	1.64	1.50	2.09	1.65		
B7	1.43	2.14	2.00	2.45	2.04	2.78
Mean:	1.55	1.52	1.87	1.73	2.04	2.78
St. Dev:	0.26	0.27	0.26	0.34		
Maximum:	1.83	2.14	2.09	2.45		
Minimum:	0.98	1.25	1.32	1.38		

FIGURE 9-1: SCALED DEPTH OF BURIAL

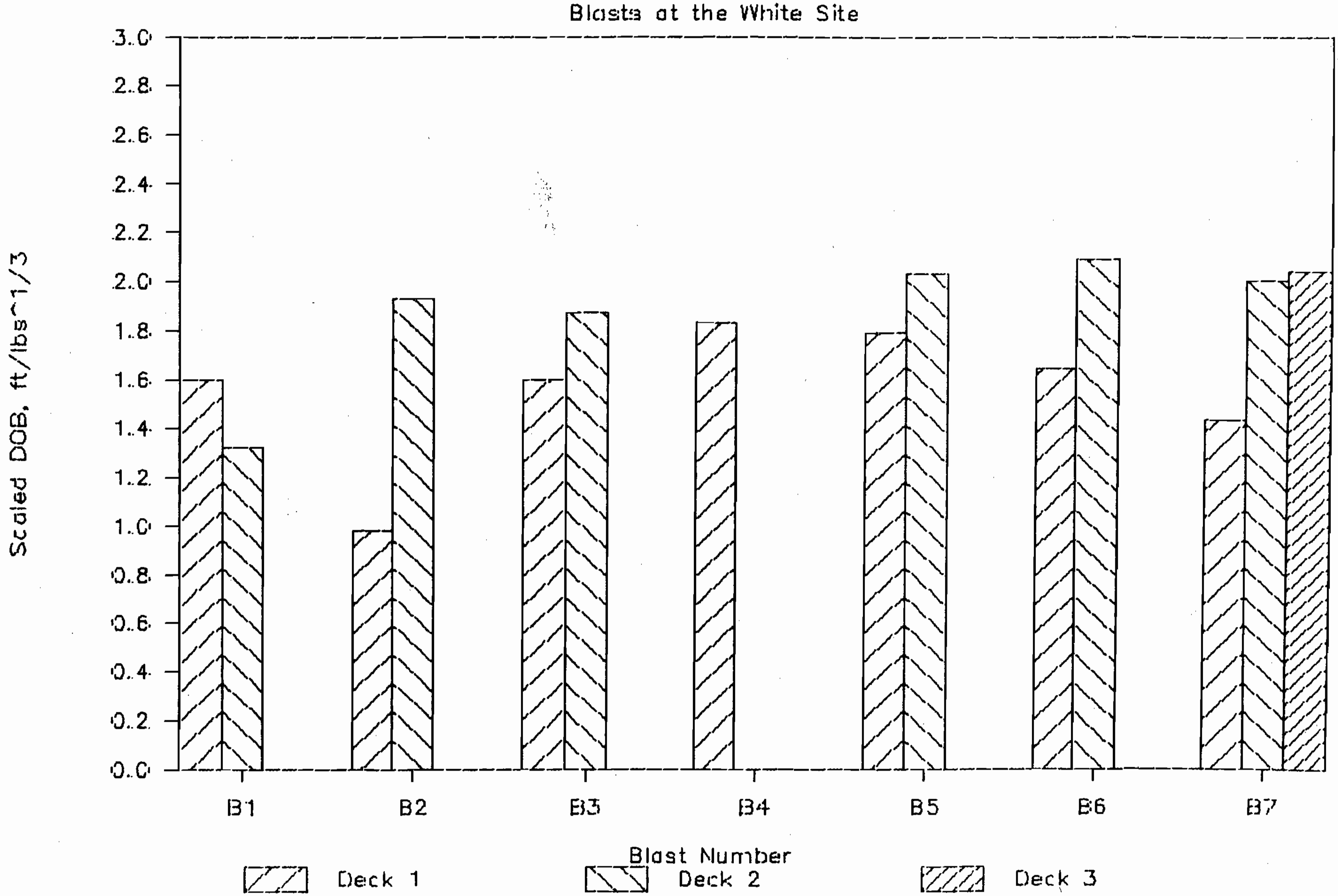
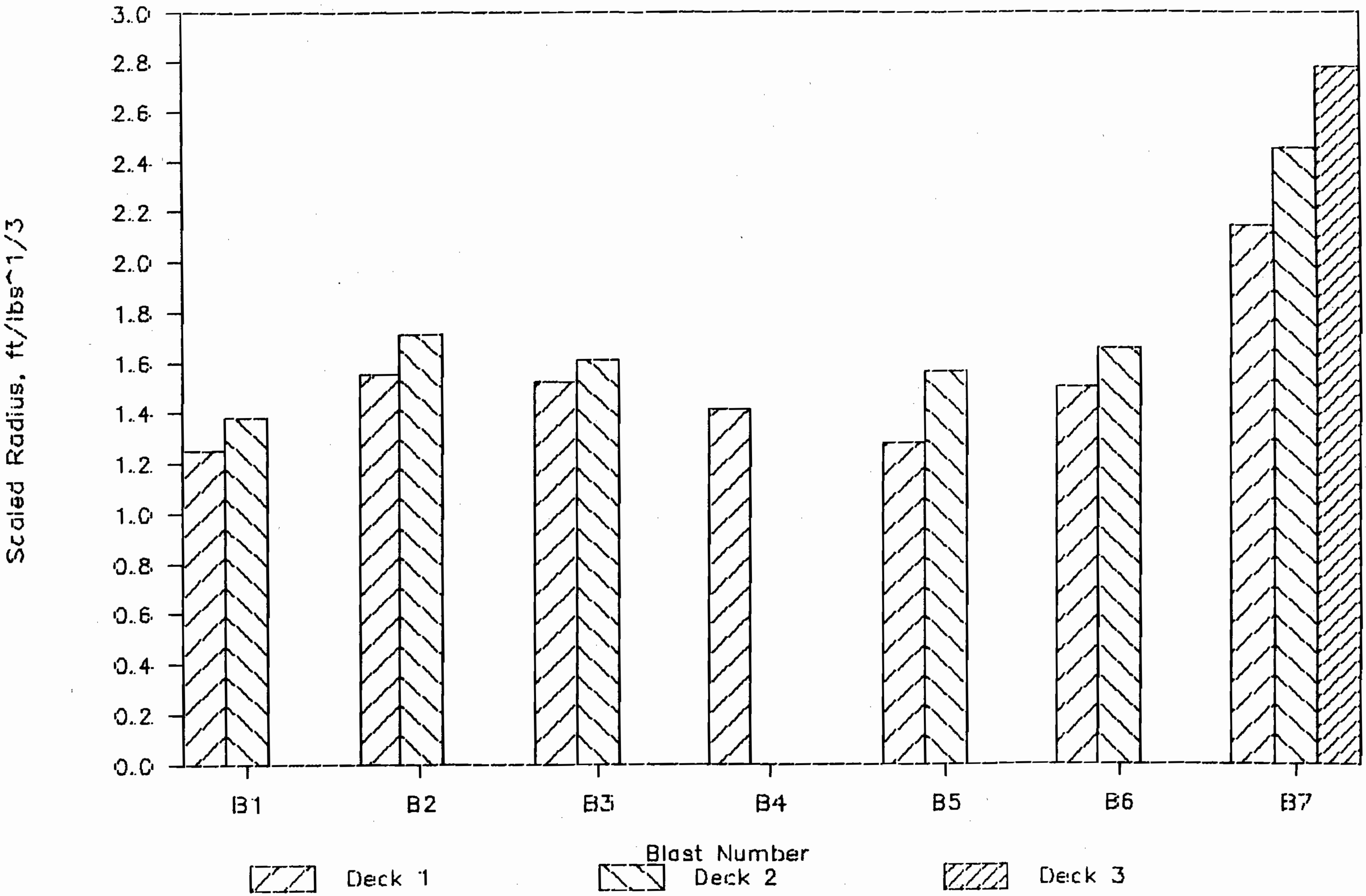


FIGURE 9-2: SCALED CRATER RADIUS

Blasts at the White Site



Blast B8 was a large area blast containing 65 holes. Thirty-three holes struck a void and thirty-two drilled into pillars or unmined coal along the perimeter. The pattern for the shot was 15x15 feet, which was based on this pattern having worked successfully for B7. The holes were loaded with three decks of powder as was B7, however, the explosive loads were increased to insure full collapse of the area. The location of the decks was similar to those in Blast B7.

For design purposes it was planned to place 60 pounds of PowerAN in the bottom decks. However, where the holes were dry ANFO was sometimes used in deck 1. Deck 2 was to be loaded with 30 pounds of ANFO and deck 3 was also designed to be loaded with 30 pounds of ANFO. Some variations to this loading plan occurred to account for the needs of individual holes. A few holes in B8 had less than three decks due to penetrating voids at quite shallow depths.

The bottom deck was positioned 5 feet above void areas and 2 feet above the floor in blastholes that intersected pillars. In the latter case the charge would crater off the side but the scaled depth of burial cannot be determined exactly.

The second deck was three feet long. It often had to be positioned to avoid water attack. In these cases a hole plug was placed just above the rise of water and one to two feet of stemming was added. Therefore, some variation in depth of burial could result in the second deck.

The top deck was a three foot deck that quite consistently was located thirteen feet from surface. The location above deck 2 typically ranged from 5 to 7 feet.

The blast was designed with a four foot deck of PowerAN in the bottom. However, to adjust for individual hole depths the length of this deck often ranged to 5 or 6 feet in length. The standoff to the void (when struck) was quite consistently 5 feet, but did range from 4 to 6 feet. For holes drilled into a void the scaled depths of burial ranged from 1.6 $ft/lb^{1/3}$ to 2.04 $ft/lb^{1/3}$.

The second deck had to be placed with water in mind when the holes were wet. Holes that struck a void, however, were often dry, where the second deck location was planned. Therefore, the standoff to the bottom deck was most often 5 feet, but did range from 4 to 6 feet. For a 5-foot standoff and ANFO the scaled depth of burial was 2.09 $ft/lb^{1/3}$. The range was from 1.77 $ft/lb^{1/3}$ to 2.41 $ft/lb^{1/3}$.

The third deck was also most commonly spaced 5 feet above the deck below. This deck had three feet of ANFO. The same scaled depths of burial result as for deck 2.

This blast was drilled on a 15x15 foot pattern. Therefore, the scaled radii reflect this pattern. For deck 1 the typical value was 1.92 $ft/lb^{1/3}$. For deck 2 the average value was 2.41 $ft/lb^{1/3}$ which also was true for the third deck. Information for blast B8 is summarized in table 9-9.

This blast was observed to be quite successful in collapsing the area. It also generated only modest surface swell.

Based on all the data collected it is concluded that, for irregular openings with low, narrow voids a scaled depth of burial of 1.65 $ft/lb^{1/3}$ should be designed for the bottom deck. This is lower than proposed in Chapter Two but reflected the situation in the field.

Upper decks can be designed to a higher scaled depth of burial. A value of 2.0 $ft/lb^{1/3}$ will be acceptable in most cases.

For shots to collapse individual rooms a scaled radius in the bottom deck of 1.50 $ft/lb^{1/3}$ will be a good design. For upper decks 1.60 $ft/lb^{1/3}$ would be acceptable. To achieve such a variation will require a denser explosive in the bottom deck and ANFO in upper decks.

For area shots a higher scaled radius seems acceptable. Observation indicates that a 15-foot square pattern can be successful, but is considered to be near the upper limit. For area shots the scaled radius for the bottom deck would typically be 1.92 $ft/lb^{1/3}$. For decks above the first it should not exceed 2.41 $ft/lb^{1/3}$. Again, the approach requires a denser explosive in the bottom deck to achieve these results.

For a pattern blast 15x15 foot dimensions yield a spacing that is 2.14 times the depth of burial. This represents the upper end of what is possible. A 12x12 foot pattern gives a spacing of 1.75 times the depth of burial (for a 7 foot DOB) which is more typical. For work of this type it would be best to start at a 12x12 foot pattern in area shots and subsequently move toward a 15x15 foot pattern if field results warrant this.

9.2.3 Stemming Heights and Surface Disruption

Reducing surface disruption from blasts to cave in old works has the advantage of making drilling on subsequent blasts easier. It is also important because marginal surface displacement will reduce the amount of subsequent reclamation required on the site. When little or no disturbance is created at the surface it may be possible to leave topsoil in place and to avoid post-blasting reclamation.

TABLE 9-9: SUMMARY OF CRATERING DATA FOR HOLES THAT INTERSECTED A VOID IN BLAST B8

DECK NUMBER	SCALED DEPTH OF BURIAL			SCALED RADIUS		
	Typical $ft/lb^{1/3}$	Maximum $ft/lb^{1/3}$	Minimum $ft/lb^{1/3}$	Typical $ft/lb^{1/3}$	Maximum $ft/lb^{1/3}$	Minimum $ft/lb^{1/3}$
1	1.79	2.04	1.60	1.92	2.12	1.69
2	2.09	1.77	2.41	2.41	2.41	2.19
3	2.09	1.77	2.41	2.41	2.41	2.19

Therefore, stemming heights were varied to determine the effect upon surface displacement. In blast B1 a 9 foot stemming height was used. This gave a scaled depth of burial to the upper surface of 3.06 to 3.22 $ft/lb^{1/3}$ depending on whether a 3 foot or 4 foot explosive column was involved. Considerable surface disruption resulted. Blast B2 also had 9 foot collars and the same scaled depths of burial. In blast B3 some holes had 12 feet of stemming. Scaled depths of burial ranged form 4.01 $ft/lb^{1/3}$ to 4.25 $ft/lb^{1/3}$. Since the depths of burial below the top deck ranged from 1.74 $ft/lb^{1/3}$ to 2.05 $ft/lb^{1/3}$ it could be expected that, to the extent that there was room to displace, the motion would tend to be toward the old works. In addition, the area of surface doming and cracking around the hole would be reduced. Surface disruption should decrease. This was generally what was observed in B3. Therefore, it appeared that increasing the stemming had advantages for controlling disturbance at the top surface.

Blast B4 was shot with only one deck of explosive, which was the heavy ANFO product. Sixteen and nineteen feet of stemming were used. The explosive deck was 3 feet long in each case. The scaled depth of burial was 4.92 $ft/lb^{1/3}$ at 16 feet of stemming and 5.77 $ft/lb^{1/3}$ at 19 feet of stemming.

The shot produced almost no surface disturbance at all. This is an ideal result for the design of subsequent blasts and for post-blasting reclamation. However, T.V. camera work showed that the void was not completely filled around one hole. Instead the void was moved up 6 feet and reduced in height by 1 foot. Therefore, the increased depth of burial was successful in negating surface displacement and cracking but inadequte explosive was present to insure full collaspe.

From the results of B4, blast B5 was designed. In this case a deck of PowerAN was placed in the bottom as before. A second deck, consisting of ANFO, was now placed above the first with a standoff of 5 feet. Then 16 to 17 feet of stemming was used to fill the blasthole. The scaled depth of burial to the surface was 5.45 $ft/lb^{1/3}$ at 16 feet and 5.76 $ft/lb^{1/3}$ at 17 feet.

This shot gave good results. The void in the vicinity of the blastholes completely collapsed. The surface disturbance was minor with only a small one-foot hump on the centerline of the production holes seen. Hairline cracks radiating out about 5 feet from the blastholes were also seen. When blasting individual workings this result represented an ideal conditon.

Blast B6 was the first of the area shots. Typically holes drilled in the pillars had stemming heights of 20 feet or more. Blastholes that intersected a room had variable stemming heights based on the hole depths and the need to fit the explosive decks into this total hole length. However, based on the previous blasts, the attempt was made to use greater stemming lengths where possible. Therefore, most were in the range of 13 to 16 feet. A few were 9 feet due to the need to place a powder deck in a quite short blasthole.

Typically, the holes that struck void had a scaled depth of burial to the surface of from 4.6 $ft/lb^{1/3}$ to 5.55 $ft/lb^{1/3}$. For pillar holes depths of burial were large and ranged from 5.55 $ft/lb^{1/3}$ to 7.46 $ft/lb^{1/3}$.

This shot provided successful collapse of the workings. Surface disturbance was modest, however, some severe cracking did result along the east side in an area where the mined intersections were large. Surface profile varied from 2 feet of depression to 3 feet of swell which is minimal and easily regraded level if desired.

Blast B7 was also an area pattern shot consisting of 20 holes on a 15x15 foot pattern. The stemming heights were similar to B6. Again there was a range based on the needs of individual blastholes. Stemming heights ranged from 12 to 18 feet. Thus scaled DOB ranged from 4.28 $ft/lb^{1/3}$ to 6.19 $ft/lb^{1/3}$. The most common stemming height was 13 feet (13 holes) which, with a 3-foot deck of ANFO gives a scaled DOB of 4.60 $ft/lb^{1/3}$.

Regarding surface disturbance this blast provided reasonable results. When detonated the area rose uniformly to an estimated height of 6 feet and then resettled. There were some areas of surface swell ranging from 0.5 to 3.0 feet high. There were no areas of depression. There were large, upended chunks scattered throughout the blast. Surface cracking extended a maximum of 15 feet.

The design, therefore, left a quite acceptable surface area after the shot, with minimum humping. Surface cracking did not interfere with subsequent production drilling. Surface cracks and chunks would be eliminated if regrading was performed.

Blast B7 left some voids where holes were drilled into pillars. Therefore, the explosive loads were somewhat increased for shot B8. One result was less stemming in the blastholes. In this case almost all the 65 holes had 10 feet of stemming. The top deck was a 3-foot deck of ANFO. Therefore, the scaled depth of burial was 3.65 $ft/lb^{1/3}$ to the top surface.

The result was that surface cracking outside the blast area extended 10 to 12 feet. There was some surface swell, but this did not exceed 4 feet in height. The area could be easily regraded level if desired.

Based on all the experimentation performed a scaled depth of burial of 5.0 $ft/lb^{1/3}$ is recommended to minimize surface disturbance and swell while insuring complete collapse of the area. For a 3-foot deck of ANFO this is a stemming height of 14 feet and for a 4-foot ANFO deck a stemming height of 16 feet is required. In most cases this design will give a minimum of surface heave and cracking, but will provide explosive sufficiently high in the bank to insure collapse.

It is important to recognize that adjustments of this kind can best be made when the voids are small in comparison to the overburden above the workings. In these cases the swell of the blasted material will fill the void and allow the total bank to remain near the same elevation or somewhat swelled. Then, a profile can be achieved that is easily regraded and avoids mixing of the soils.

When the rooms are large relative to the cover above the workings this is less successful. In such cases collapse of the room can leave depressions of as much as 10 to 15 feet. Regrading need increase significantly. Also soils become more mixed, although a substantial proportion of growth material remains near the top.

Based on the results at the White location it is possible to postulate that for quite deep cover, relative to the void dimensions, the swelled overburden would close the void without the need to blast all the way to surface. In these cases the area could be reclaimed with little or no surface disturbance whatsoever. Thus, prior topsoil removal and post-blasting regrading could be eliminated while leaving the site in good condition. Previous studies indicate an average of 18 percent swell occurs (4). To be conservative estimates of the ability to close the voids should be made based on a 12 percent swell factor.

9.3 BLASTS IN INDIVIDUAL ROOMS

It was found that a primary concern for blasting in an environment such as that presented at this test location is the ability to identify the location of the abandoned workings. For the one acre test site this was a manageable problem, but would become increasingly difficult for larger sites. Both time and cost would be significant factors.

These results are in contrast to results in regular workings for which good documentation is available (4). Exploration time and cost was quite reasonable in those cases.

Once the rooms are located successful collapse can be effected by placing blastholes above the works and collapsing material into the void below. It is important that the blastholes be accurately placed for best results. The use of wooden stakes is a successful way to locate the blastholes in the field. The prime problem with this approach however is the high cost of exploration work.

Another problem that occurred when blasting in individual rooms was difficulty in controlling the area caved. This also was unique to this setting and had not been experienced in previous work on rooms that were of consistent width and direction, with uniform rib pillars (18 feet wide) between each room. At the White site the irregular sized pillars, which were randomly positioned, and the wide intersections made it much more difficult to control surface disturbance from a blast.

The resulting difficulty is that results is that zones outside the blast area, that partially cave, become difficult to access with the drill for the next shot. Therefore, the possibility exists that there will be pockets of uncollapsed void but the drill can not be repositioned to drill off the area. If the equipment can be repositioned it was found that drilling had to be accomplished through broken material which is difficult and it is less certain that the hole can be held open until it can be loaded with explosive. The use of 6-inch diameter PVC pipe to collar such holes was quite successful in keeping them open for drilling and loading.

An important goal, then, is to avoid the disruption. This can be partly achieved by increasing the stemming at the top of the hole, thereby increasing the explosive burial relative to the surface. The approach has been discussed above.

A second solution is to examine the mine map developed from the exploration program and find a place to stop the blast that is less likely to result in unwanted collapse. Such a spot would be one with good pillars nearby and an area that does not have unusually large spans. A blast such as B4 for example would be less likely to create problems than B1, which stopped in a large, relatively unsupported void.

By controlling the stemming height and the blast orientation the problem of unwanted collapse could be controlled reasonably well. However, where there are wide intersections it is not likely that the situation can be fully resolved. This was proved in blast B3 where steps to reduce backbreak, such as increasing stemming and reducing powder in the bottom deck reduced cracking to 15 feet (from 25 to 30 feet) but did not eliminate it. Cracks behind the middle section of B6 also indicated that large intersections could still be a problem.

At the White test location successful methods of collapsing individual works were found. However, the method was not entirely satisfactory because of the extensive exploration required to identify the works and the problems with unwanted collapse. Both of these characteristics contribute to added time and cost for performing the work.

9.4 AREA BLASTS

As described previously area blasts were drilled on a regular pattern without regard for the location of the workings. Therefore, some holes intersected void while others did not. All holes were loaded and detonated in the blast. For blast B6 And B8 fifty to sixty percent of holes struck the void.

The method reduces exploration requirements. The T.V. camera was eliminated. This was necessary in any event since the workings in the area of B6 to B8 contained much water and the camera could not be used in such conditions

Also, prior exploration drilling can be reduced since there is not the same need to identify the workings as for the case of shooting in individual rooms. Drilling would be primarily to identify the perimeter of the mining and to provide a general idea of void locations and dimensions. Exploration drilling can be reduced by at least one-half when area blasting is employed.

When an area blast is shot those holes that have intersected a void can be expected to crater down into the rooms below. Crater design data applicable to these situations have been provided above. Blastholes that are drilled into pillars will crater to the side. However, it cannot be exactly known where these holes are relative to the nearest void. Therefore, it was found that the bottom deck, located in the pillar, should be loaded more heavily than blastholes situated over the void. One to two additional feet of powder is required. This need was based on the evidence of blast B7, where some voids near pillars were not completely collapsed initially. Therefore, in Blast B8 the powder consumption in the lower decks was increased from 45 to 60 pounds of heavy ANFO.

The deck loading and cratering approach was successful for the area shots. As before, the powder loads were designed to generate cratering effects. Therefore, powder factor is not a good benchmark for design. However, for comparison the area blasts on a 15x15 foot pattern, assuming an average hole depth of 35 feet gave a powder factor of 0.41 pounds per cubic yard. On a 12x12 pattern the powder factor was approximately 0.64 pounds per cubic yard.

Area shots created quite a uniform surface disturbance. At the White site this was primarily a few feet of surface swell with some small depressions also seen. With properly chosen stemming heights cracking and backbreak beyond the blasting area was quite well controlled. By contrast shooting individual rooms left a more variable profile. Also, backbreak was as much as 30 feet behind the blast for individual room shots. However, in later individual room shots this was controlled to less than 15 feet.

Based on the results of shots B6 to B8 the 10x10 foot pattern is smaller than necessary for an area blast in these types of materials. The 12x12 foot pattern provides quite acceptable results. It would be a good pattern for starting a new reclamation project.

The 15x15 foot pattern also produced good results, provided there was adequate powder load. Each hole required 120 pounds of explosives of which 60 pounds was placed in the bottom deck and 30 pounds were placed in the middle and top decks.

The best approach to area blasting on similar sites would be to start with a 12x12 pattern. For a hole with two decks the bottom deck should contain 60 pounds of HANFO, which will give a 4 to 5 foot rise. The upper deck would consist of 30 pounds of ANFO (3 feet). If three decks are needed then each of the upper two decks would contain 30 pounds of ANFO. Initial stemming heights should be about 14 feet.

If results warrant the patterns could be expanded toward 15x15 feet with the same explosive loading. If surface disruption were more than desired stemming heights could be increased to 16 feet.

The area pattern blasts shot at the White site did not suffer from backbreak problems affecting the next shot to the extent that the individual room blasts did. Also surface swell was modest and more uniform than for the individual shots. However, some of the surface disruption of the individual blasts was eliminated when longer stemming lengths were employed.

More importantly the area blasting technique reduced time consuming and costly exploration requirements If a site were completely reclaimed using this technique such costs would be considerably reduced.

Area blasting will be most economical where extraction ratios were high meaning a large proportion of the area was undermined. When recovery was low increasing numbers of holes will be in pillars rather than void and the cost will tend to increase on an extraction basis relative to blasting individual rooms with holes located only above the voids.

9.5 BLAST TIMING

The timing of the blasts was similar to that used at Beulah. A prime consideration was to allow adequate time for each charge to crater toward the void below before the next charge in the hole detonated. Also, charges cratering in succession would provide some additional relief to the side for subsequently detonating charges.

At the White Site all blasts provided 50 ms of delay between individual explosive decks in a hole. To accomplish this every other period down the hole delays were used. These ranged from 100 ms to 200 ms and were number 4, 6 and 8 period delays used in conjunction with detonating cord and slider primers.

To delay between holes noiseless trunkline delays were used. In all cases these were 42 ms surface delays.

The area shots were delayed somewhat differently than the individual room shots. For these blasts multiple holes were detonated per surface delay. This was possible because the distance to buildings was long and vibration was not a major concern.

Shooting multiple holes per delay would add to the particle velocities near the shot. However, the down-the-hole delays result in only one deck firing per blasthole simultaneously. Therefore, this helped to control close-in vibration. Even with considerably more explosive weight detonated per delay there was no evidence of collapse outside the blast area due to high vibration level. Also, the higher vibration levels did not create more surface cracking and disruption than was experienced for the individual room blasts, where holes were independently delayed.

The multiple hole detonations did lead to higher vibration levels at the seismograph location, as reported in Chapter 7. This was not an important concern due to the remote nature of the site. However, it is important to realize that vibration levels could be further reduced by adding more surface delays if one was reclaiming a site near to residences or other buildings.

The combination of 42 ms surface delays and 50 ms between decks down-the-hole worked well. This method is quite satisfactory when shooting individual rooms. If a site is to be reclaimed using area pattern shots then one should give added consideration to the surface delay combinations. For example, if it would be advantageous to detonate the shot on a chevron configuration, then a combination of 42 ms and 17 ms surface delays, or other suitable combination, could be used to delay between chevrons and along each individual chevron. The goal, as before, would be to maximize the ability for overburden to move toward and fill the mined rooms.

9.6 EFFECTS OF WATER IN THE WORKINGS

The White site location afforded the opportunity to blast in some rooms that were water filled. The east side of the property, where the pattern blasts were performed was the area containing water.

It was necessary to use the waterproof heavy ANFO in the bottom deck of the blastholes in these shots, unless the drill hole struck a void at shallow depth. This was a clear example of a property where a waterproof product has important application. The most difficult holes were those that were drilled in pillars as these exhibited the greatest water rise.

Holes that intersected a void could have some water rise depending on water conditions in the works below.

Changes had to be made to the loading procedures in those holes that contained water. After the bottom deck of heavy ANFO was loaded a seismic hole plug was placed just above the water level. Then, one to two feet of cuttings were placed on the plug followed by the second explosive deck, consisting of ANFO. In this manner the ANFO deck was protected against water attack.

Depending on the amount of water present this method could lead to longer spacings between decks than would be optimum. Therefore, another approach would be to load a waterproof product in the second deck as well. Doing so would increase the explosives cost, but would also assure the best placement of the explosive and protection against water attack. However, plugging the hole above the water rise and using ANFO did work at the test location.

Prior to blasting on this site there was some concern as to the effect of water on blasting effectiveness. Observation at the site indicated little difference in results in wet and dry areas. Some water was forced out of the area by venting through unloaded exploration drillholes. There was no slumping observed that could be attributed to slurrying and flow of the blasted overburden. Observation of the site in the summer of 1989 indicated that time did not result in such effects either.

The primary effect of the presence of water then was to increase the time taken to load the shot. The extra time resulted from added difficulty loading the bagged HANFO in wet holes and the time to place the plugs and backfill. Provided proper wet hole loading procedures were used there was not a problem with reduced performance of the explosive.

It was harder to assess what happened to the water upon detonation of the blast and what effect this could have on overall site reclamation, because there is no means of directly studying the situation. At this location only a portion of the workings were water filled and the indication is that displaced water was either vented or absorbed in the workings as a whole. For a site where most or all of the works were water filled it would be wise to monitor for seepage of water from the reclaimed mine. This would be especially important if the mine contained acidic water or water with other contaminants. Provision for the collection and treatment of such water might be required.

The results of this research indicate that such problems are not likely to arise unless the mine as a whole is substantially water filled.

9.7 BLASTING TECHNIQUE VERSUS MINING RECOVERY

In many older mines the amount of coal extracted was low. Thus, as little as twenty-five percent of the area may have actually been mined. In other cases, if there was good stability of the workings, extraction could be considerably higher.

At the White test site the extraction was substantial as can be seen in the mine map in figure 3-10 found in Chapter 3. Interior pillars were often small and were located in a haphazard fashion.

For this type of site area blasting can prove quite economical. The number of holes needed to pattern blast over the area may not be much more than individual room blasthole requirements. For example, area pattern blasting all of the White Site on a 15x15 foot pattern would have required approximately 195 blast holes. This compares with 178 holes actually shot at the White location, of which 95 blastholes were located in individual shots in blasts B1 to B5. Given the reduction in exploration time and effort required overall efficiency and cost would be enhanced using the pattern approach.

The area approach will be more useful in cases where the extraction was substantial because more blastholes will intersect the void and crater down into it. Pillars will be smaller, so that holes intersecting pillars will be closer to a void and more effective.

The area method is sensitive to the pattern dimensions as well. If a 12x12 pattern were necessary at the White test area then 306 holes would be needed. Any savings would then depend on the ability to greatly reduce exploration costs.

When the extraction is low shooting on an individual room basis will reduce the number of production holes and explosives needed to reclaim the site. Also exploration costs could be less because there are fewer workings to find.

Therefore, the choice of method will be related to the degree to which the site was mined. Both methods work and the choice will be related to cost. Since individual sites can vary substantially such an analysis will need to be made based on costs and technical specifics pertaining to the given location.

CHAPTER 10
PROJECTED COSTS FOR PERFORMING AML RECLAMATION BY BLASTING ON A PRODUCTION BASIS

10.1 INTRODUCTION

Cost analyses have been performed to estimate the cost of reclaiming AML sites using blasting methods. The costs are based on performing the work on a production basis and, therefore, do not include costs associated with research components of the current study. Such costs include high-speed camera work, research related field labor, data analysis and report writing. Thus, the costs used are strictly related to blasting on a production basis.

Some advance engineering work is required when blasting old mining sites. However, engineering is needed no matter what mitigation technique is chosen. Therefore, costs for preliminary engineering design and planning are not included in the analysis.

Exploration work is necessary to identify the location and depth of old workings. This need is somewhat specific to the use of blasting to reclaim old works. Also, different blasting methods have different exploration requirements. Therefore, exploration costs have been estimated for these methods, which include vertical opening closure, blasting over individual works and area pattern blasting.

The unit costs used in the analysis are based on the costs experienced for this research. If necessary, modifications have been made to reflect a production rather than research and development orientation.

Table 10-1 lists unit costs for drilling and explosives that have been used in the analysis.

TABLE 10-1: UNIT DRILLING AND EXPLOSIVES COSTS USED IN THE COST ANALYSIS

Item	Beulah Test Site	White Test Site
Drilling	$0.85/ft.	$1.00/ft.
ANFO	$17.15/CWT	$18.075/CWT
Heavy ANFO	$36.75/CWT	$36.82/CWT
Surface Delays (42ms NTD)	$263.50/C	$267.50/C
Down-The-Hole Delays (#4-#8)	$223.50/C	$225.50/C
One Pound Cast Primers	$3.57 ea.	$3.50 ea.
15 Grain Detonating Cord	$80.50/M ft.	$81.00/M ft.

The costs differ slightly between sites, but are largely comparable. The drilling cost increase was due to the more remote nature of the site. At Beulah the drilling contractor could leave the site to perform other work and return when required. For the White site the equipment and crew were more than three hundred miles from home base and, as such, were captive to the site.

The ANFO and heavy ANFO costs reflect the cost of purchasing explosives in small lots of bagged powder. The larger the order that can be placed the more favorable the pricing. On a bulk basis the ANFO cost could be reduced by 4 to 5 cents per pound and the HANFO price could similarly be reduced by 10 to 15 cents per pound for the 70/30 emulsion to ANFO mix. However, in AML work small lot pricing will commonly be experienced unless a site is unusually large. Therefore, the pricing in the table ought to be quite realistic for most AML locations.

Table 10-2 lists other equipment and supply needs for the site.

TABLE 10-2: EQUIPMENT AND ACCESSORY COSTS FOR AML BLASTING

Item	Beulah Test Site	White Test Site
Explosive Magazine	\$400/mo	\$250/mo.
Seismograph	\$775/mo	\$605/mo
Field Office Trailer	\$450/mo	\$400/mo
Blasting Equipment	\$5/hr	\$8/hr
Stakes and Paint	\$22.48	\$20.45
EPCO Holes Plugs	\$345.00	\$345.00
Miscellaneous Items	\$82.00	\$82.00

The field office trailer was necessary for the current work because the research required considerable design and data recording in the field. It also provided storage for seismographs, high-speed camera, film, etc. For a small production project the trailer likely could be eliminated. For a large project, performed over a period of time, the field office facility is needed.

The blasting equipment included the use of a suitably outfitted pickup for loading explosives, blasting machine, blasting galvanometer, lead wire, blasting signs and siren. All equipment complied with blasting regulations.

Table 10-3 lists the labor costs estimated as being appropriate to this work. It is assumed that the drill crew labor cost is included in the drilling footage cost. Experience at the White site indicated that, for larger shots, it may be necessary to have the drill crew assist in blasthole loading.

TABLE 10-3: PROJECTED UNIT LABOR RELATED COSTS FOR AML BLASTING PROJECT

Labor Type	Hourly Costs
Blasting Engineer	$39.75
Blaster	21.75
Blaster Helper	14.50

These rates reflect base salary or wage and all payroll taxes and benefits. It is considered that a three-man crew will be appropriate. It is expected that the blasting engineer will assist in loading holes and other tasks associated with preparing and detonating the blasts. For small, straight forward blasts time requirements for the blasting engineer may well be considerably reduced.

An additional cost is that of transportation and subsistence for persons working on site. This is difficult to estimate because it depends on whether drilling and blasting expertise is available near to the site or whether it requires crews to work at remote locations.

For this cost analysis, travel costs for the Beulah site have been limited to transportation costs only. This reflects the fact that the contractor was located near the site.

For the White site additional travel costs have been assigned. This test area was over three hundred miles from the location of the contractor. Therefore, added costs were necessary.

Thus these analyses portray both the situation where travel is a small item and where it is a larger factor due to remoteness of the site. Specific information on how travel costs were applied is given below.

10.2 BLASTING VERTICAL OPENINGS - BEULAH SITE

10.2.1 Cost Calculations for Vertical Openings

For the closure of vertical openings, eighteen blasts including 199 blastholes have been analyzed. The nature of each blast has been described in Chapter 5.

The cost analysis of closing vertical openings is best performed on a per opening basis. This provides better results than attempting a cost per acre since there could be great variation from site to site in the number of sinkholes found on an acre of mined land.

The costs are based on just the production requirements to mitigate the features. As described in the introduction to this Chapter research and development components of the work are excluded.

Table 10-4 lists the costs for each sinkhole blast at the Beulah Site. These have been itemized in seven categories as shown in the table. Totals and average costs are also provided.

The exploration cost for each vertical opening are those that were listed in Chapter 3. There were no exploration costs for the second shot on a sinkhole, where such blasts were necessary.

The drilling cost for a sinkhole is based on the footage obtained from the field data sheets. The Beulah drilling rate of $0.85 per foot is applied to this footage to obtain the total drilling cost for the vertical opening.

The consumption of explosive by type was maintained for each blast. For some shots there was a part bag of explosive left after loading for which the weight had to be estimated. Therefore, ANFO weights could be off by a maximum of 25 pounds and HANFO weights by as much as 15 pounds. These quantities will not represent significant variations in the cost projections. The unit explosives costs have been listed in table 10-1.

The quantities of primers used was recorded for each blast. So too were the number of down-the-hole and surface delays. Detonating cord consumption was estimated on the total hole footage and the pigtail lengths on surface. These estimates were within twenty-five feet so the maximum cost variation is $2.00 per blast, an insignificant sum.

TABLE 10-4: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH.

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$456.00	$1,639.48
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$380.00	$1,496.74
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$228.00	$859.68
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$304.00	$1,036.18
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$228.00	$579.56
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$532.00	$1,315.27
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$608.00	$1,531.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$304.00	$849.53
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$760.00	$2,745.93
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$228.00	$743.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$608.00	$1,922.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$152.00	$545.79
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$532.00	$1,568.15
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$228.00	$665.37
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$190.00	$641.04
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$456.00	$1,895.72
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$380.00	$1,338.72
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$456.00	$1,988.69
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$7,030.00	$23,363.61
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	390.56	1297.98

The cost listed as accessory and supply cost were those fixed project expenses which were listed in table 10-2. Added to those were transportation costs for getting to and from the site. Total transportation cost was estimated at $972.05.

It is difficult to determine how best to incorporate these site costs. In this case the accessory and supply cost has been divided equally among the blasts at the site. The total estimated cost was $3,782.00 for this category. Divided among the 18 blasts fired at Beulah, the unit cost becomes $210.11 per shot.

This approach is not exact and in reality some variation would occur. However, if the estimate was made on the ratio of holes in a blast to the total number of blastholes, for example, the result would not reflect that some of these costs do not vary. Therefore, applying these charges equally among the blasts was deemed most appropriate.

The labor charges are based on an estimate of the amount of time spent directly preparing to detonate a blast. This varies by the size and complexity of the shot. Blasts with more decks, for example, take longer to load. Blasts having more holes that strike a void below and require a plug also take longer to load, as the plugs must be placed using loading poles. The labor required for research purposes such as design, measurement, analysis and camera work is not included here.

The proposed labor crew is that needed for performing the work on a production basis. It is recommended that the crew include an experienced blasting engineer. This type of blasting is never routine and technical expertise on site is important to good results. In addition, when blasting near residences vibration must be carefully controlled and a suitable technical background is necessary to perform appropriate designs.

In table 10-4 the total cost for each veritical opening is provided. The totals by category are also shown. The average cost for each category and the total effort are also shown in the table.

10.2.2 Cost Analysis for Vertical Openings

Observation of table 10-4 shows that the cost varies considerably from one sinkhole to another. The most costly openings to close are those that include a large depression and an open slope down into the mined room. The most notable examples are VO-7 & 8-N, VO-7 & 8-S, VO-11 and VO-13. Vertical openings of this sort require large shots and are, typically, the type that may require a second shot to close.

Openings that are undercut, providing a profile that is small at surface but large at the base, are the next most costly. These features are difficult to drill around and the collapse is more difficult to control. Openings VO-9 and VO-12 are the examples of this condition.

Finally, other openings, that are smaller and do not have an open ramp to the works, appear to require the least cost to close. Examples of this type of opening are VO-3, VO-6 and VO-10. Costs to mitigate these features are quite economical

Table 10-4 provides average values for each cost category and for the total cost to close the feature. Averages are somewhat misleading due to the great variation in size and form of individual vertical openings. However, based on the sinkholes shot at Beulah, it is estimated that the average cost to mitigate a feature is $1,297.98.

Some of the blasts listed are the second shot on the same opening. The average above is based on the total eighteen blasts. When the costs on the same sinkhole are combined then a total of fourteen features were reclaimed. Applying this number the average cost is $1,668.83 per vertical opening blasted closed.

This second average is likely the more accurate value. It accounts for the periodic need for more than one blast to fully close a single feature. Since this need is likely to occur on any site the average obtained on this basis is more representative. Since the blasts at Beulah covered a wide range of opening types and sizes the value will be good for estimating purposes for any projects in similar materials and depths.

Figure 10-1 is a chart illustrating the category costs by percentage. The category labeled primers on the chart includes surface and down-the-hole delays and detonating cord.

The graph illustrates that, for vertical openings, exploration costs are modest. Since the workings below can often be seen it takes fewer exploration holes to determine the trend of the workings.

Labor is the single largest cost. It is almost double the next highest cost. Therefore, efficient use of labor will be a key to optimizing project costs.

When the two explosive cost percentages are combined the remaining categories represent quite similar portions of the costs. Combined these are 64.6 percent of the total cost to close a sinkhole.

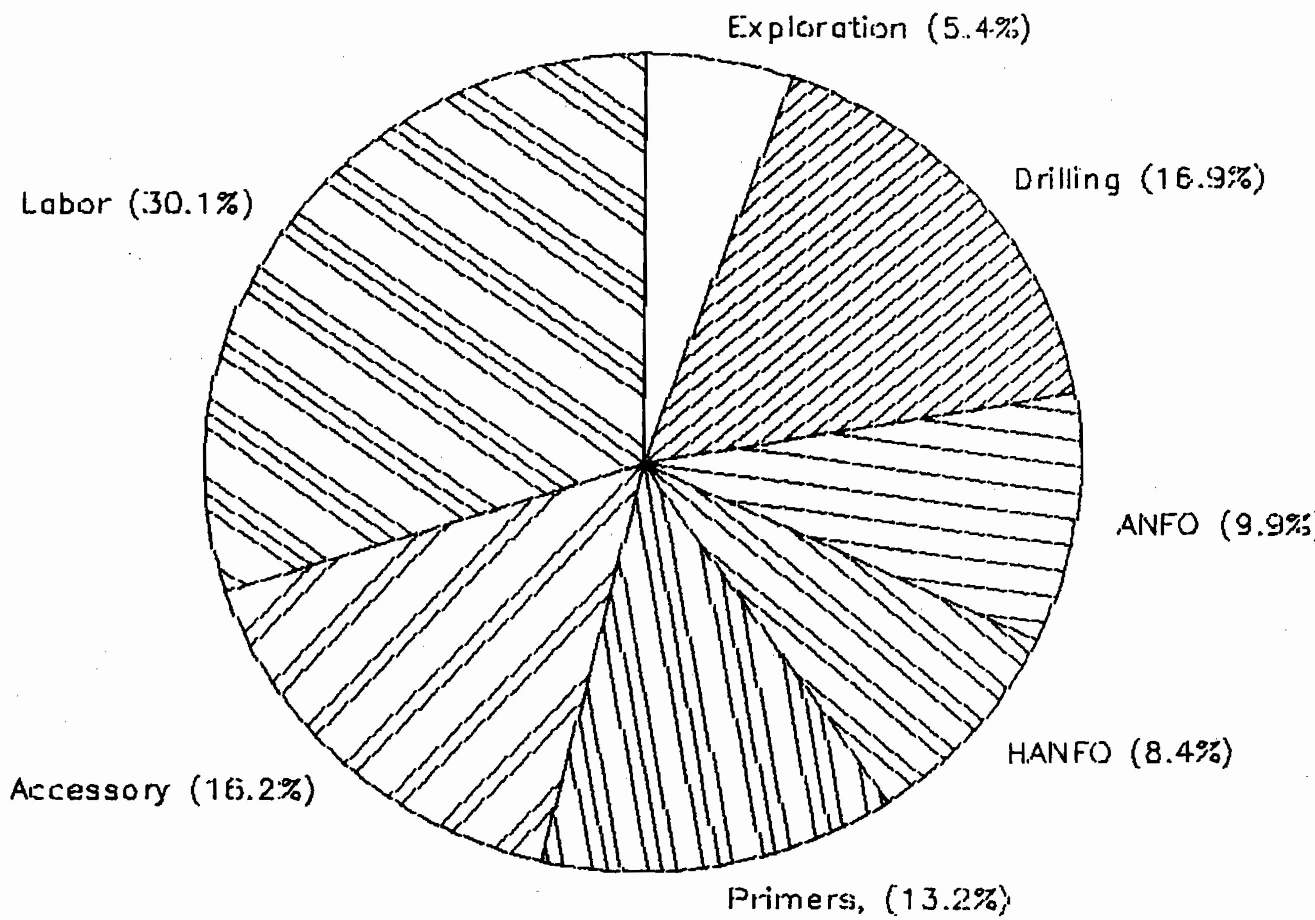
There are few savings to be made in exploration costs. Substituting ANFO for HANFO wherever possible would reduce costs. Using a primer of less than one pound weight could also be a feasible way to reduce cost to some extent.

For these shots a quite favorable drilling cost was obtained. Therefore, this category is likely to be most sensitive to cost increase. However, one sees contract drilling

FIGURE 10-1

ESTIMATED PRODUCTION BLASTING COSTS

Vertical Openings



costs in 6½-inch diameter in granite quarries being performed for \$1.69 to \$2.00 per foot. Therefore, it is unlikely that drilling costs would exceed \$1.50 per foot in this application.

10.2.3 Variations in Costs

Some factors may cause the cost of closing a sinkhole to increase or decrease. The factors which could have the greatest effect are listed below:

- Changes in unit drilling cost
- Use of ANFO for all explosives loading
- Changes in labor cost

The impact of these factors has been analyzed and are discussed below.

10.2.3.1 Changes in Unit Drilling Costs

The rate obtained for drilling for this research project was quite favorable and it is unlikely that a lesser rate could be realized. However, increases would be a possibility, especially when drilling rigs are less available due to high levels of available work. As a result the Beulah blasts have been analysed for a drilling cost of \$1.50 per foot as opposed to the \$0.85 per foot used in the original estimate.

The resulting increase in drilling costs can be seen in table 10-5. The average cost per blast has risen from \$219.02 to \$386.50 an increase of 76 percent.

The total cost per blast has risen from an average of \$1,297.98 to \$1,465.46 which is an increase of \$167.48 or 13 percent. The average, on the basis of fourteen sinkholes blasted is \$1,884.16 as opposed to the \$1,668.83 calculated earlier also a cost increase of 13 percent.

The increase in unit drilling cost results in the drilling category representing 26.4 percent of the total cost. It is equal to the labor cost which now equals 26.7 percent of the total. Other requirements are reduced accordingly.

The analysis does indicate that the cost of reclaiming vertical openings by blasting is sensitive to the drilling cost. If the cost of drilling were to increase to more than \$1.50 per foot drilling would become the greatest single cost involved in mitigating these features.

On the other hand a cost rise to \$1.50 per foot would not preclude the use of blasting to close vertical openings. The total increase is, on average, about \$200 per sinkhole which is manageable. An increase to \$2.50 per foot, which would increase the cost of reclaiming an average feature to \$2,215.45, could

TABLE 10-5: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. DRILLING COST INCREASED TO $1.50

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$547.06	$120.05	$242.55	$214.07	$210.11	$456.00	$1,876.54
VO-2	7	$354.45	$328.13	$42.86	$209.48	$113.90	$210.11	$380.00	$1,638.93
VO-3	5	$96.90	$221.26	$25.73	$99.23	$74.33	$210.11	$228.00	$955.56
VO-4A	8	$38.25	$243.00	$42.88	$176.40	$126.84	$210.11	$304.00	$1,141.48
VO-4A#2	3	$0.00	$90.00	$42.88	$0.00	$47.57	$210.11	$228.00	$618.56
VO-4B	8	$33.15	$336.00	$102.90	$99.23	$147.48	$210.11	$532.00	$1,460.87
VO-5	13	$68.85	$394.50	$102.90	$132.30	$185.31	$210.11	$608.00	$1,701.97
VO-6	6	$93.50	$168.00	$68.60	$0.00	$78.12	$210.11	$304.00	$922.33
VO-7&8-N	28	$66.30	$1,068.00	$377.30	$242.55	$484.47	$210.11	$760.00	$3,208.73
VO-7&8N#2	9	$0.00	$202.50	$102.90	$0.00	$88.04	$210.11	$228.00	$831.55
VO-7&8-S	20	$60.35	$750.00	$282.98	$0.00	$336.54	$210.11	$608.00	$2,247.98
VO-7S#2	4	$0.00	$120.00	$51.45	$0.00	$64.23	$210.11	$152.00	$597.79
VO-9	10	$110.50	$333.00	$68.60	$297.68	$160.56	$210.11	$532.00	$1,712.45
VO-9#2	6	$0.00	$154.50	$68.60	$0.00	$71.11	$210.11	$228.00	$732.32
VO-10	5	$73.10	$123.00	$42.88	$0.00	$55.25	$210.11	$190.00	$694.34
VO-11	22	$55.25	$747.00	$334.43	$77.18	$339.45	$210.11	$456.00	$2,219.42
VO-12	14	$89.25	$447.00	$214.37	$0.00	$191.69	$210.11	$380.00	$1,532.42
VO-13	21	$37.40	$684.00	$214.37	$374.85	$308.36	$210.11	$456.00	$2,285.09
TOTALS	199	$1,263.95	$6,956.95	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$7,030.00	$26,378.29
Average	11.00	70.22	386.50	128.15	108.41	171.52	210.11	390.56	1465.46

begin to make the approach less competitive with other techniques. However, $2.50 per foot would represent a high unit cost in overburden that drills easily. Large diameter blasthole drilling costs in surface coal mines, by contrast, seldom exceed $2.00 per foot.

10.2.3.2 Effect of Explosives Usage

Costs could be reduced if only ANFO were used and the HANFO were eliminated. The savings would be $0.196 per pound. The basic ANFO product has adequate energy output to perform as required except, possibly, in cases where a dangerous or undercut sinkhole requires a greater than usual drill setback from the rim of the vertical opening. Generally, ANFO will be acceptable.

Table 10-6 present the results when all explosive is ANFO. The average cost for explosive drops from $236.56 per feature on average to $178.74 per feature. The total cost per sinkhole decreases from an average of $1,297.98 to $1,240.16 per opening.

The ANFO cost is now 14.4 percent of the total. This compares to a combined percentage of 18.3 percent where a combination of both products was considered (figure 10-1).

This decrease is not particularly significant. The reasons are the HANFO was used only in the bottom deck, it was not used for second shots on an opening and it was not used for three initial blasts on a sinkhole. Therefore, the quantities being replaced are not large and the cost decrease is limited.

The use of HANFO in the bottom deck can guard against water attack. It also adds energy at the toe where burdens may be long. Therefore, replacing the HANFO does not give a sufficient cost decrease, when blasting vertical openings, to warrant the loss of technical advantages.

A more significant situation results if all waterproof HANFO is used. This could occur if a property was particularly wet and there was concern about the use of ANFO, which is very susceptible to water attack.

Table 10-7 contains the data for the situation where the blasts at Beulah are assumed to be performed using only HANFO. In this case the average explosive cost increases to $383.02 an increase of 62 percent over use of the combined products. The average cost per blast rises to $1,444.43 an increase of $146.45 per blast. Explosives are now 26.5 percent of the total cost and are equal to the labor cost involved.

TABLE 10-6: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. ANFO ONLY

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$233.24	$214.07	$210.11	$456.00	$1,510.12
VO-2	7	$354.45	$185.94	$140.62	$113.90	$210.11	$380.00	$1,385.02
VO-3	5	$96.90	$125.38	$72.04	$74.33	$210.11	$228.00	$806.76
VO-4A	8	$38.25	$137.70	$125.20	$126.84	$210.11	$304.00	$942.10
VO-4A#2	3	$0.00	$51.00	$42.88	$47.57	$210.11	$228.00	$579.56
VO-4B	8	$33.15	$190.40	$149.21	$147.48	$210.11	$532.00	$1,262.34
VO-5	13	$68.85	$223.55	$164.64	$185.31	$210.11	$608.00	$1,460.46
VO-6	6	$93.50	$95.20	$68.60	$78.12	$210.11	$304.00	$849.53
VO-7&8-N	28	$66.30	$605.20	$490.49	$484.47	$210.11	$760.00	$2,616.57
VO-7&8N#2	9	$0.00	$114.75	$102.90	$88.04	$210.11	$228.00	$743.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$336.54	$210.11	$608.00	$1,922.98
VO-7S#2	4	$0.00	$68.00	$51.45	$64.23	$210.11	$152.00	$545.79
VO-9	10	$110.50	$188.70	$207.52	$160.56	$210.11	$532.00	$1,409.39
VO-9#2	6	$0.00	$87.55	$68.60	$71.11	$210.11	$228.00	$665.37
VO-10	5	$73.10	$69.70	$42.88	$55.25	$210.11	$190.00	$641.04
VO-11	22	$55.25	$423.30	$370.45	$339.45	$210.11	$456.00	$1,854.55
VO-12	14	$89.25	$253.30	$214.97	$191.69	$210.11	$380.00	$1,338.72
VO-13	21	$37.40	$387.60	$389.30	$308.36	$210.11	$456.00	$1,788.77
TOTALS	199	$1,263.95	$3,942.27	$3,217.36	$3,087.28	$3,781.98	$7,030.00	$22,322.84
Average	11.00	70.22	219.02	178.74	171.52	210.11	390.56	1240.16

TABLE 10-7: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. HANFO ONLY

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$499.80	$214.07	$210.11	$456.00	$1,776.68
VO-2	7	$354.45	$185.94	$301.32	$113.90	$210.11	$380.00	$1,545.72
VO-3	5	$96.90	$125.38	$154.37	$74.33	$210.11	$228.00	$889.09
VO-4A	8	$38.25	$137.70	$268.29	$126.84	$210.11	$304.00	$1,085.19
VO-4A#2	3	$0.00	$51.00	$91.89	$47.57	$210.11	$228.00	$628.56
VO-4B	8	$33.15	$190.40	$319.73	$147.48	$210.11	$532.00	$1,432.87
VO-5	13	$68.85	$223.55	$352.80	$185.31	$210.11	$608.00	$1,648.62
VO-6	6	$93.50	$95.20	$147.00	$78.12	$210.11	$304.00	$927.93
VO-7&8-N	28	$66.30	$605.20	$1,051.05	$484.47	$210.11	$760.00	$3,177.13
VO-7&8N#2	9	$0.00	$114.75	$220.50	$88.04	$210.11	$228.00	$861.40
VO-7&8-S	20	$60.35	$425.00	$606.39	$336.54	$210.11	$608.00	$2,246.39
VO-7S#2	4	$0.00	$68.00	$110.25	$64.23	$210.11	$152.00	$604.59
VO-9	10	$110.50	$188.70	$444.68	$160.56	$210.11	$532.00	$1,646.55
VO-9#2	6	$0.00	$87.55	$147.00	$71.11	$210.11	$228.00	$743.77
VO-10	5	$73.10	$69.70	$91.89	$55.25	$210.11	$190.00	$690.05
VO-11	22	$55.25	$423.30	$793.82	$339.45	$210.11	$456.00	$2,277.92
VO-12	14	$89.25	$253.30	$459.36	$191.69	$210.11	$380.00	$1,583.71
VO-13	21	$37.40	$387.60	$834.21	$308.36	$210.11	$456.00	$2,233.68
TOTALS	199	$1,263.95	$3,942.27	$6,894.34	$3,087.28	$3,781.98	$7,030.00	$25,999.82
Average	11.00	70.22	219.02	383.02	171.52	210.11	390.56	1444.43

The conclusion from this analysis is that the use of heavy ANFO should be limited to only those situations requiring such a product. These include the presence of water or large burdens. However, elimination of all HANFO is not necessary and its use in the bottom decks does have advantages as described above.

10.2.3.3 Effect of Changing Labor Cost

Labor costs could be reduced if the blasting engineer position was eliminated. As stated earlier in the Chapter there are technical reasons why this may not be wise, but the analysis is provided for comparison purposes.

Table 10-8 illustrates the effect of this change. Only a blaster and helper are engaged in the work. The hours have been adjusted to provide the same total manhours as before. Compared to the original case the average cost of labor is reduced from $390.56 per shot to $279.43. The total cost to blast an opening reduces from $1,297.98 to $1,186.85, a $111.13 reduction. The two man crew reduces the labor portion of the cost from 30.1 percent to 23.5 percent.

The other factor that affects the cost is the required time to load and shoot a blast. If actual requirements are greater than estimated above the cost will increase.

To examine the effect of this a twenty-five percent increase in required time has been studied. Table 10-9 presents the results for a three-man crew. In this case the average labor cost to shoot a blast increases to $488.19. The total cost per shot increases to $1,395.62 an increase of $97.64 per shot. The labor cost is then 35 percent of the total.

Table 10-10 provides the results of the same time increase when a two-man crew is utilized. The labor cost per blast is now $349.28 and the average cost of a shot is $1,256.71. The labor cost as a percent of the total is 27.8 percent. Thus, the two-man crew, with a twenty-five percent increase in the time required to perform a shot is about equivalent to the cost of the original three-man crew.

The results of the analyses indicate a range from about a $100 per shot decrease to a $100 per shot increase in cost. The $100 decrease by eliminating the blasting engineer is not likely warranted. Elimination of this position could adversely affect the project and, in total, cause an increase in cost.

The twenty-five percent increase in time is manageable. The increase beyond the estimate would likely have to be fifty percent or more before costs became adverse.

TABLE 10-8: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. TWO MAN CREW

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$326.25	$1,509.73
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$271.88	$1,388.62
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$163.13	$794.81
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$217.50	$949.68
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$163.13	$514.68
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$380.63	$1,163.89
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$435.00	$1,358.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$217.50	$763.03
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$543.75	$2,529.68
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$163.13	$678.92
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$435.00	$1,749.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$108.75	$502.54
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$380.63	$1,416.78
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$163.13	$600.49
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$135.94	$586.98
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$326.25	$1,765.97
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$271.88	$1,230.59
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$326.25	$1,858.94
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$5,029.69	$21,363.30
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	279.43	1186.85

TABLE 10-9: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH-THREE MAN CREW-TWENTY FIVE PERCENT TIME INCREASE

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$570.00	$1,753.48
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$475.00	$1,591.74
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$285.00	$916.68
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$380.00	$1,112.18
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$285.00	$636.56
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$665.00	$1,448.27
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$760.00	$1,683.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$380.00	$925.53
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$950.00	$2,935.93
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$285.00	$800.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$760.00	$2,074.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$190.00	$583.79
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$665.00	$1,701.15
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$285.00	$722.37
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$237.50	$688.54
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$570.00	$2,009.72
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$475.00	$1,433.72
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$570.00	$2,102.69
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$8,787.50	$25,121.11
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	488.19	1395.62

TABLE 10-10: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH-TWO MAN CREW-TWENTY FIVE PERCENT TIME INCREASE

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$407.81	$1,591.29
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$339.84	$1,456.58
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$203.91	$835.59
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$271.88	$1,004.06
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$203.91	$555.46
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$475.78	$1,259.05
VO-5	13	$68.85	$223.55	$102.90	$132.30	$165.31	$210.11	$543.75	$1,466.77
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$271.88	$817.40
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$679.69	$2,665.62
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$203.91	$719.70
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$543.75	$1,858.73
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$135.94	$529.72
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$475.78	$1,511.93
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$203.91	$641.27
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$169.92	$620.96
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$407.81	$1,847.53
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$339.84	$1,298.56
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$407.81	$1,940.50
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$6,287.11	$22,620.72
Average	11.00	$70.22	$219.02	$128.15	$108.41	$171.52	$210.11	$349.28	$1,256.71

10.2.4 Comparison With Other Methods

The most direct comparison to other means of mitigating vertical openings is with direct fill. Direct fill is a method whereby material from a borrow pit is hauled to and dumped in the open voids. This method is subject to significant cost sensitivities including the haulage distance, the availability of material and the size of the vertical openings.

The direct fill method has been used in North Dakota to reclaim sinkholes in the past. Generally, the conditions of use have been a ready supply of overburden from surface mine spoil piles at no cost, a short haul distance and vertical openings similar to those studied in this research. The cost of filling an individual sinkhole has been estimated to be less than $1,000 per hole.

The cost per hole would quickly rise if fill had to be purchased. Also a long haul for fill would rapidly increase costs. On the other hand blasting costs would not increase as rapidly for remote sites. The primary cost increase would be travel expense and some mobilization cost for drilling. These costs could increase the average cost per sinkhole by $150 for a similar number of blasts as performed at Beulah.

When blasting is used more disturbance around the vertical opening will occur. Therefore, the regrading cost after the void is filled will be greater than for direct fill, if such further reclamation is carried out. In the worse case the cost by blasting could be 50 to 100 percent greater than for direct fill. However, when direct fill is not performed under the most favorable terms, the two methods will quickly become equivalent. In addition, the absence of free borrow material, or a long haul distance could preclude the use of the direct fill method. Blasting, on the other hand, would be precluded more for technical reasons. The most important of these would be openings less than 500 feet from residences or residences more than 500 feet away that are undermined.

10.3 COST ANALYSIS OF BLASTING IRREGULAR MINE WORKINGS

10.3.1 Cost Data for Production Blasting of Irregular Workings

The initial compilation of cost data for the White site has been performed on a per blast basis. The data has been divided into two groups; one for the blasts over individual rooms (B1 to B5) and one for the area pattern blasts (B6 to B8). As before these cost estimates are for production based work and do not include research related tasks required for the current project.

In this case the exploration costs are not computed on a per blast basis. It was found difficult to allocate these to individual blasts in a meaningful way, because exploration studies outside a given blast area often had an impact on the successful establishment of the location of rooms within the blast area. Therefore, the exploration cost is computed separately and then combined with the production costs.

If the entire site were reclaimed by blasting individual rooms then the exploration cost would be that given in table 3-2 in Chapter 3. This amounted to $21,395 for the total area. Even though part of the area was pattern blasted, exploration was thorough throughout so that the mine workings could be properly delineated for research purposes. Therefore, the cost estimate reflects the kind of exploration effort required for shooting individual rooms, where exact knowledge of the room size and trend is necessary.

Pattern blasting would considerably reduce the exploration cost. Some T.V. camera work might be needed to help orient the effort, but would not exceed 3 days work. The engineering time is estimated to reduce by one half. The eight-inch diameter drilling is estimated at 150 feet and the 4- or 6-inch diameter exploration drilling is estimated at 3000 feet. The cost then totals $9,790,000 or less than one-half that required for blasting individual rooms.

Another matter that had to be addressed was how to apportion the accessory and supply cost. In this case it was found that dividing the cost equally among the shots gave unrealistically high cost for the individual room blasts and too low a cost to the pattern blasts, each of which affected a considerably larger area. After several theories were tested it was found best to divide this cost up based on the ratio of the square feet of mined area in the blast to the total square feet mined. This best related the cost to the effort required for each blast. Since the pattern blasts were much larger these absorbed a higher portion of the cost. The square footage was found by planimeter from the mine maps that were prepared. The map was provided earlier as figure 5-1 (page 127).

Table 10-11 provides the estimated production blasting costs for the individual room blasts. Table 10-12 gives the same data for the area pattern blasts.

The drilling cost is based on a $1.00 per foot cost and the total footage required for each shot. Each blasthole was taped and the depth recorded.

The explosive consumption was recorded for each blast by type. The weights were quite accurate, the only error possibility being the estimate of partial bags of powder remaining after a blast.

TABLE 10-11: PROJECTED COSTS FOR PRODUCTION BLASTING OVER MINED ROOMS AT THE WHITE TEST SITE.

Blast Number	Number of Holes	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
B1	12	$243.00	$117.49	$110.46	$151.40	$244.64	$456.00	$1,322.99
B2	11	$289.00	$99.41	$254.05	$181.25	$276.00	$532.00	$1,631.71
B3	13	$304.00	$81.34	$165.69	$200.14	$299.30	$632.00	$1,682.47
B4	5	$122.00	$0.00	$88.38	$52.83	$155.92	$304.00	$723.13
B5	3	$98.00	$18.08	$77.32	$51.11	$97.08	$228.00	$569.59
TOTALS	44	$1,056.00	$316.32	$695.90	$636.72	$1,072.94	$2,152.00	$5,929.88
Average	8.80	$211.20	$63.26	$139.18	$127.34	$214.59	$430.40	$1,185.98

TABLE 10-12: PROJECTED COSTS FOR AREA PATTERN BLASTING AT THE WHITE TEST SITE.

Blast Number	Number of Holes	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
B6	46	$1,425.00	$262.09	$751.12	$741.91	$1,104.29	$945.00	$5,229.41
B7	20	$744.00	$171.71	$320.33	$413.34	$480.61	$532.00	$2,661.99
B8	65	$2,347.00	$894.71	$1,060.42	$1,470.20	$2,979.16	$1,260.00	$10,011.49
TOTALS	131	$4,516.00	$1,328.51	$2,131.87	$2,625.45	$4,564.06	$2,737.00	$17,902.89
Average	43.67	$1,505.33	$442.84	$710.62	$875.15	$1,521.35	$912.33	$5,967.63

Primer and delays counts were kept for each blast. The length of detonating cord was estimated according to the hole depths and the surface pigtail lengths.

The accessory and supply cost calculation is based on the data in table 10-2. The method for apportioning the cost to individual blasts was explained above.

The labor cost is based on the data from table 10-3. The hours spent actually involved in production blasting were estimated and the labor cost applied to these numbers.

The total cost for each blast in tables 10-11 and 10-12 are found on the right-hand side of the table. These are production costs only. At this point the exploration costs are not included.

10.3.2 <u>Analysis of the Cost Data For The White Site</u>

Figures 10-2 and 10-3 show the cost breakdown for blasting above the rooms and pattern blasting respectively. There are differences between the two. When blasting above the rooms the labor cost is the greatest single cost as the process is more time consuming. Other costs are roughly similar when the explosive percentages are combined. This cost breakdown looks similar to that in figure 10-1 for the vertical openings.

When pattern blasting is employed the labor cost percentage is reduced by half. The drilling cost rises substantially and is equal to the accessory and supply cost. The increase in drilling cost is partly due to the depth to the workings increasing toward the east. Also, in pattern blasts up to half the holes were in pillars. These holes were typically drilled deeper so the total drilling footage tended to increase.

The total cost for each blast was determined and was listed in the tables in the previous section. The square feet of mined area has also been determined for each shot, from the maps. It is therefore, possible to determine a cost per square foot of mined area for each shot. An average is then determined.

This data is presented in table 10-13. The top part of the table is for the individual room blasts and the bottom portion is for the area pattern blasts. The table shows that using the pattern blasts is a more economical approach in the case of irregular workings. Blast B8, which was a large pattern blast on a 15x15 foot pattern was especially economical relative to the individual room method.

Taking the total cost and the total square footage mined in each case one finds a cost of \$1.65 per square foot for the individual room blasts and \$1.17 per square foot for the area pattern blasts. The former, applied to the total area of 18,882

FIGURE 10-2

ESTIMATED PRODUCTION BLASTING COSTS

White Site—Above Rooms

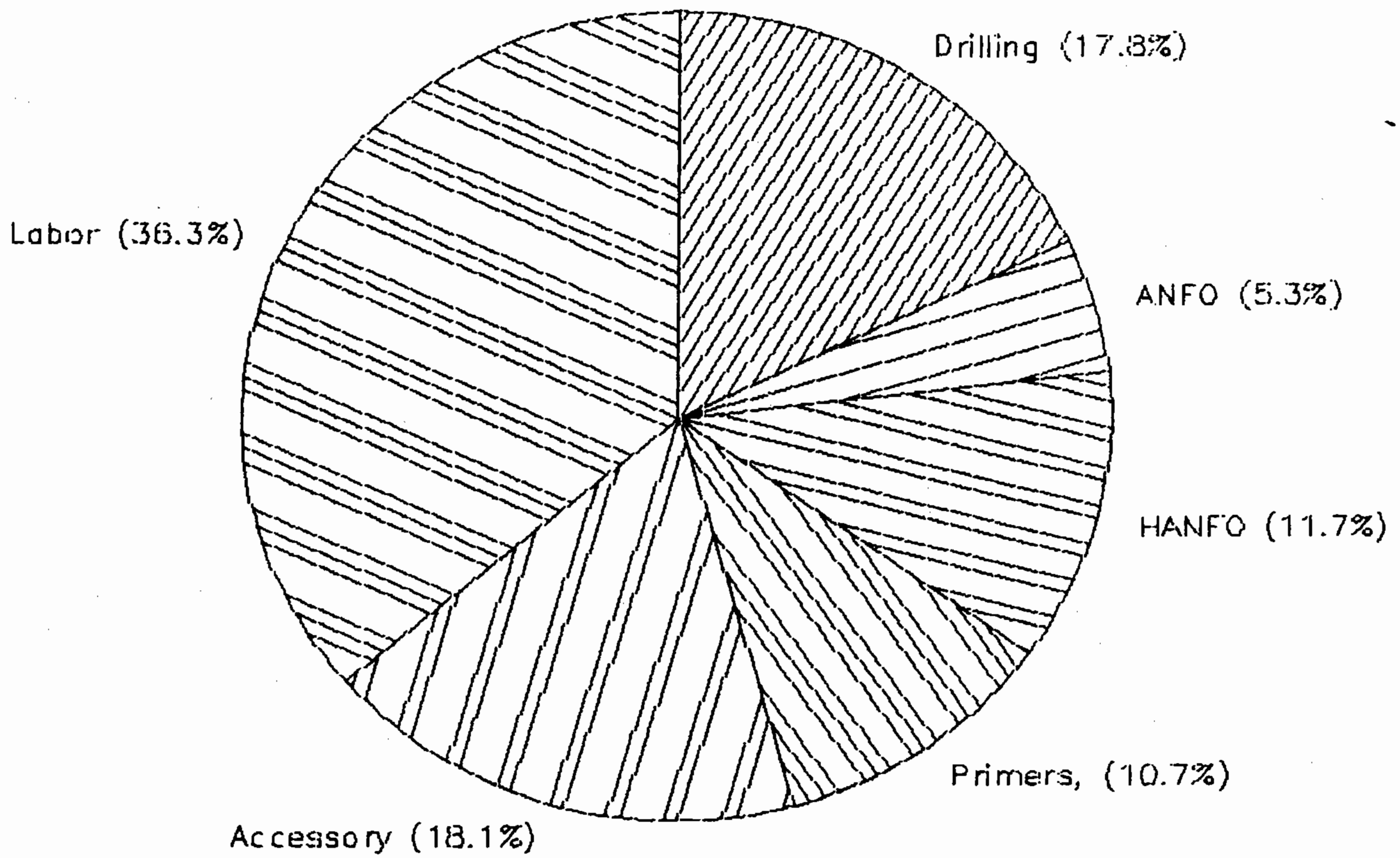


FIGURE 10-3

ESTIMATED PRODUCTION BLASTING COSTS

White Site—Pattern Blasts

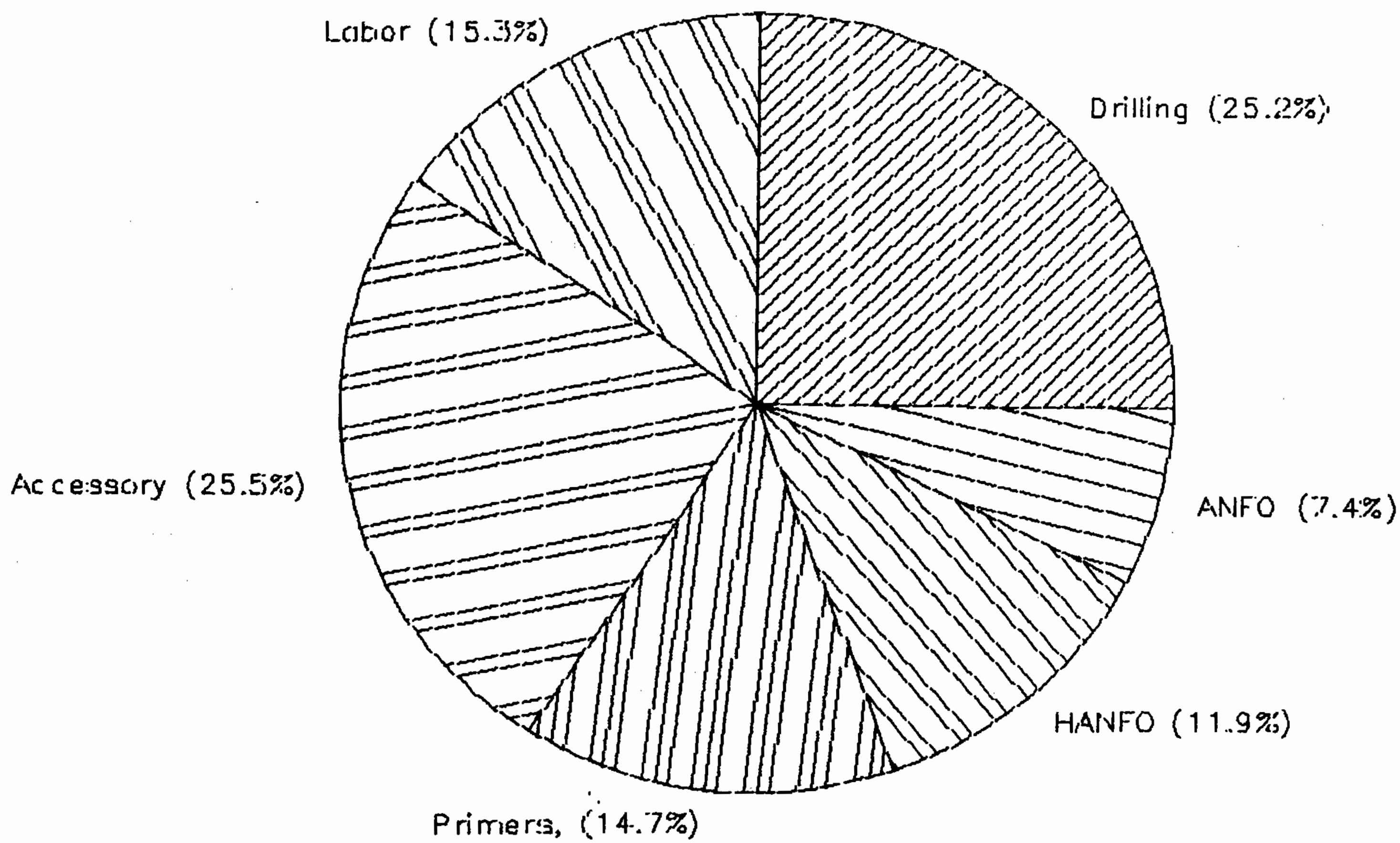


TABLE 10-13: UNIT COST OF RECLAIMING MINE WORKINGS BY BLASTING METHODS

Blast Number	Blast Cost	Mined Area sq. ft.	Cost Per Square Foot
B1	$1,322.99	819	$1.62
B2	$1,631.71	924	$1.77
B3	$1,682.47	1002	$1.68
B4	$723.13	522	$1.39
B5	$569.59	325	$1.75
Totals	$5,929.89	3592	$1.65
Average	$1,185.98	718	
B6	$5,229.41	3697	$1.41
B7	$2,661.99	1609	$1.65
B8	$10,011.49	9984	$1.00
Totals	$17,902.89	15290	$1.17
Average	$5,967.63	5097	

square feet, yields a cost of $31,155. For the pattern blasts the resulting total cost is $22,092. This represents a significant reduction in total cost.

The projected exploration costs for each case were given above. These costs applied to the same square footage of mined area as do the production costs. Therefore the cost per square foot of mined area for exploration when individual room blasting is used is $1.13 per square foot. When a pattern blasting technique is used for the whole area the cost per square foot is $0.52.

The combined exploration and production costs for shooting individual rooms is, therefore, $2.78 per square foot. The combined cost for blasting on an area pattern basis is $1.69 per square foot. For the total 18,882 square feet of mined area this amounts to a cost of $52,492 for shooting individual rooms and $31,910 for blasting by area pattern techniques. Clearly, there are significant savings to be obtained using pattern blasting techniques.

The surface area affected by the blasting was approximately one acre. A planimeter run on the mine map gives an affected area of about 36,000 square feet which is close to one acre. Therefore, the costs quoted above can be considered as applicable on a per acre basis.

These costs make clear that the pattern blasting approach is the better method for mines such as that exemplified by the White site. This is due especially to the substantial variance in exploration costs. However, production costs are also lower on a square footage basis when a 15x15 foot pattern can be used.

10.3.3 Sensitivity Analysis of Costs

Cost factors such as drilling, explosives and labor can be varied as was done for the Beulah test blasts. Some variation in costs will result from such changes. However, one of the more important changes that will affect the cost is the recovery experienced, or, the amount of an acre of land that was undermined.

For the individual room method the internal recovery, that is the ratio of rooms to pillars will also offset the cost since only rooms are blasted. This aspect is less important when pattern blasting as holes are placed without regard to being in room or pillar.

Table 10-14 are estimated costs for different field recovery rates. It can be seen that, as the amount of an acre that was undermined increases, the cost per acre also rises. At the White Site 18,882 square feet were undermined for a recovery of 43 percent of the one acre site. Since field recovery is

TABLE 10-14: ESTIMATED COST FOR PATTERN BLASTING AS A FUNCTION OF FIELD RECOVERY

Field Recovery %	Square Feet Undermined	Cost For Pattern Blasts
25	10890	$18,404.10
30	13068	$22,084.92
35	15246	$25,765.74
40	17424	$29,446.56
45	19602	$33,127.38
50	21780	$36,808.20
55	23958	$40,489.02
60	26136	$44,169.84

unlikely to be less than 25 percent or more than 60 percent,costs per acre should be within the project range.

Table 10-15 shows the different field recoveries and also different internal recoveries. The internal recovery relates to the area of the internal pillars that were left to support the mine during operation. In this analysis recoveries are shown that reflect mining that left from 20 to 55 percent of the mine area as pillars. At the White location internal pillars had an area of 3,712 square feet. The internal recovery was 80 percent which is quite high. For many mines a recovery of 50 to 60 percent, on a square footage basis would be common.

Examining tables 10-14 and 10-15 one observes that the cost of reclaiming an acre is about equal for each field recovery, when the internal recovery is 60 percent. For internal recoveries less than 60 percent individual room blasting should be more economical, even taking into account the high exploration cost.

When the internal recovery is greater than 60 percent pattern blasting becomes increasingly economical. At 80 percent, which was the recovery at the White test site pattern blasting is much less costly than individual room blasting, reflecting the significant differences in exploration cost.

The conclusion then is that there will be an advantage to performing the additional exploration and drilling production blastholes only over the workings when there are few workings to be found. If the area is extensively undermined then pattern blasting, without regard for whether the blastholes are in pillar or room will be more economical than the time consuming effort to exactly locate all the rooms and pillars. This conclusion applies primarily to reclamation of irregular workings where the rooms cannot easily be located.

Another factor which will affect the cost is the depth to the workings. For this field test the average depth was about 35 feet. If the mine is considerably deeper both drilling cost and explosives costs will increase. It is difficult to estimate in advance how much cost will be added. For any given project the design should be developed as recommended in this report. If there is significant difference in drilling depth the added cost can then be developed from the design.

For mines that are at considerable depth the surface collapse and disruption that makes more shallow mines hazardous and destroys land use values is not likely to occur. The swell of collapsing roof material will tend to fill the void before it manifests itself at surface.

For example, consider a mined room having dimensions of 8 feet wide by 6 feet high. Consider also that the material swells by 18 percent as it caves into the workings. Then about 33 feet of height would be required to fill the void. The room

TABLE 10-15: ESTIMATED COSTS FOR RECLAIMING MINED ROOMS USING THE INDIVIDUAL ROOM METHOD FOR DIFFERENT FIELD AND INTERNAL RECOVERY RATES

Field Recovery %	Square Feet Undermined	Cost Per Acre for Different Internal Recoveries							
		Internal Recovery Ratio, %							
		45	50	55	60	65	70	75	80
25	10890	$13,623.39	$15,137.10	$16,650.81	$18,164.52	$19,678.23	$21,191.94	$22,705.65	$24,219.36
30	13068	$16,348.07	$18,164.52	$19,980.97	$21,797.42	$23,613.88	$25,430.33	$27,246.78	$29,063.23
35	15246	$19,072.75	$21,191.94	$23,311.13	$25,430.33	$27,549.52	$29,668.72	$31,787.91	$33,907.10
40	17424	$21,797.42	$24,219.36	$26,641.30	$29,063.23	$31,485.17	$33,907.10	$36,329.04	$38,750.98
45	19602	$24,522.10	$27,246.78	$29,971.46	$32,696.14	$35,420.81	$38,145.49	$40,870.17	$43,594.85
50	21780	$27,246.78	$30,274.20	$33,301.62	$36,329.04	$39,356.46	$42,383.88	$45,411.30	$48,438.72
55	23958	$29,971.46	$33,301.62	$36,631.78	$39,961.94	$43,292.11	$46,622.27	$49,952.43	$53,282.59
60	26136	$32,696.14	$36,329.04	$39,961.94	$43,594.85	$47,227.75	$50,860.66	$54,493.56	$58,126.46

dimensions in this example are typical of older mines, and would be somewhat high where thin seams were mined. It would appear then that overburden heights, where reclamation of this sort is required, would usually not be much over 50 feet. This height of overburden would fill rooms of 8x9 feet or equivalent. However, for those fewer mines with large rooms it can take deep cover to close the void before surface expression is seen. For example rooms 20 feet wide by 10 feet high require caving of 139 feet of material to fill the void when an 18 percent swell is assumed. This is why such large sinkholes can appear at Beulah even in deeper cover.

Since most mines to be reclaimed are likely to have openings of 8x9 feet or less the costs from this research will apply to most cases. However, one should be aware of and take into account sensitivity to the depth to the workings.

10.3.4 Comparison With Other Methods

Prior estimates have been made by the State of North Dakota concerning the cost of AML reclamation by other methods. In some cases these methods have been used to reclaim old works locally. The costs for methods of complete reclamation of AML sites are given in table 10-16.

TABLE 10-16: PROJECTED COSTS FOR VARIOUS RECLAMATION METHODS FOR UNDERGROUND MINES

Method	Per Acre Costs
Daylighting - without coal removal	
(a) Shallow Mine	$10,000 - $20,000
(b) Deep Mine	$100,000 - $200,000
Remote Backfill	$59,000 - $100,000
Dynamic Consolidation (Shallow Mines)	$10,000 - $70,000

Examining these methods shows that blasting costs are within the same range and the method is acceptable on a cost basis. Blasting appears to be considerably cheaper than remote backfill, another frequently used method. Daylighting shallow mines may be less costly, depending on the extent of undermining involved but costs accelerate rapidly as the overburden depth increases. Dynamic consolidation has similar costs to blasting.

As tables 10-14 and 10-15 show, blasting may be especially attractive on a cost basis when recoveries are low. Cost attractiveness is further enhanced when, as at the White site it is possible to blast with minimal surface disruption and therefore minimum post-blasting regrading requirement.

CHAPTER 11: CONCLUSIONS

As a result of this research, in which the filling of individual vertical openings and the closure of erratic, undocumented underground workings has been studied, the following conclusions have been developed.

11.1 GENERAL CONCLUSIONS

The following general conclusions, concerning the project as a whole have been made:

1. Blasting can be used to successfully close individual vertical openings. A method employing cratering charges in decked blastholes works well. The cratering charges are designed to displace to the void. Adequate swell is generated in the material to fill the sinkhole. Examination of the site a year after blasting did not indicate much settlement of the blasted material.

2. Blasting can be employed to close irregular mine workings for which there is little documentation in the form of mine maps or surveys. Two methods are available to blast in these workings. One is to blast the individual rooms with charges placed directly over the opening, designed to crater down into the mined room. The second approach is to area pattern blast without regard for whether the blastholes are drilled in room or pillar. Charges are designed to crater down into the workings or to the side depending on whether the hole is over the room or drilled into the pillar.

3. For irregular workings with few pillars the pattern blasting approach has been indicated to be most effective. For mines with a low recovery and large or many internal pillars shooting in the individual rooms is likely best.

4. Blasting individual sinkholes is somewhat more costly than direct fill methods when optimum conditions are available for direct fill. However, as haul distances increase for the fill, or a source of free or very inexpensive fill is not available, blasting soon becomes as or more economical than direct filling with borrowed material.

5. Blasting closed irregular workings is cost competitive with other methods. It appears to be significantly less costly than remote fill (hydraulic or pneumatic), an often used method. It is competitive with daylighting and with dynamic consolidation techniques.

6. The current work has been in overburden depths of a maximum of forty feet. However, using four explosive decks and a similar loading arrangement blasting could be conducted in fifty to fifty-five feet of cover. It is considered that four

decks is about the maximum that can easily be placed and timed.

7. For deeper cover it can still be possible to blast the workings closed without requiring more than four explosive decks. Consider that, for eighteen percent swell a 10x6 foot void can be filled with thirty-three feet of blasted overburden. Similarly an 10x10 foot void could be filled by fifty-five feet of overburden. Therefore, in seventy-five feet of cover only the bottom thirty-five to fifty-five feet of cover would need to contain explosives, in these examples, to adequately fill the void. It would appear that quite deep workings could be shot by determining how much material actually needs to be broken to fill the void. An associated advantage with blasting deep works would be that the project could be completed with virtually no surface disturbance. This method will not likely be successful when the rooms are very large.

8. Blasting closed vertical shafts would require the same principles as blasting vertical openings. The blasts should be designed to cast material into the void. The scaled depth of burial and radius requirements would be in the same range as for the vertical openings. For deeper shafts in more remote areas longer linear charges could be used designed with square root scaling. In this case the scaled burdens should be in the range of 2.0 to 2.5 $ft/lb^{1/2}$ and the scaled radius should be about 2.0 $ft/lb^{1/2}$. More than one row of holes may be required. This can be calculated by calculating the volume of void to fill and comparing this to the volume of rock. A main concern with shafts will be the effect of any shaft support systems in place and shaft hardware if any. The effect of these items should be given careful consideration. It is concluded that much of the data obtained in this study for vertical openings may also apply to shafts.

11.2 CONCLUSIONS CONCERNING EXPLORATION NEEDS

The amount of exploration required varies depending on the type of blasting and the type of site. The following conclusions have been arrived at from the current research:

1. The least amount of exploration is required when vertical openings are being blasted closed. The nature of these features provides much visual information not available for other types of sites.

2. When blasting sinkholes exploration drilling costs averaged $79 per opening. The cost varied from $30.60 to $354.45 for an opening. Some cost is also required for engineering time to lay out holes and interpret results. This cost would range from $40 to $80 per sinkhole.

3. Exploration needs are much greater when the need is to collapse irregular, ill defined underground workings. The method whereby the rooms are collapsed using blastholes located

directly over the workings is most costly because the location of the rooms must be known exactly. For the White Site, having an extent of one acre, the exploration cost, under these circumstances was approximately $22,000.

4. When the method of area blasting with a regular pattern is used the exploration cost is estimated to reduce considerably. The workings do not need to be defined so exactly in this case. For the White Site exploration costs of $9,790 are estimated for the one acre extent.

5. The use of the down hole T.V. camera system developed by L.C. Hanson, Inc. was very helpful in aiding the attempt to locate the rooms and reduced the number of drill holes needed to establish the hole locations. The camera does not work, however, where the workings are water filled.

6. Borehole T.V. camera work was useful for assessing the results of blasting. The instrument could be used to look back at blasted areas to examine the closure obtained. Both the Hanson device and a more sophisticated commercial instrument from the State of North Dakota were used at the White site.

11.3 FIELD PROCEDURES AND TECHNIQUES

A variety of field procedures were studied. The conclusions developed are listed below.

11.3.1 Vertical Openings

1. The blasting of vertical openings does not result in the unwanted collapse of immediately adjacent rooms. In shooting fourteen openings using eighteen blasts there were no cases, where the detonation caused nearby collapse. A key to successfully avoiding unwanted disruption is the use of a method which keeps explosive charges of low weight and provides for independent detonation of successive decks, thereby minimizing vibrational effects.

2. Examination of the test area a year after blasting shows two incidents where a new sinkhole had opened up near one that was blasted. It is difficult to say how much the blasting affected the nearby area. However, there is some indication that blasting may affect nearby, unstable areas some time after blasting.

3. In general the vertical openings fill well upon blasting. Smaller, circular sinkholes provide the best result. Longer openings with an open ramp into the workings may require two blasts to fill. The biggest problem is the sloping ramp which, in some cases, ratholes. It has also been found that using two rows of holes can provide success with only one shot required.

4. Undercut vertical openings, with a small opening at surface and a large void at the base can also require two shots to fill. It is difficult to position the drill as one would wish due to safety needs. Two openings of this type (VO-9 and VO-12) were blasted. These gave the least satisfactory overall results of the vertical openings tested.

5. The decked loading system was quite successful for vertical openings. Good movement into the opening was obtained. Powder loads were controlled at a reasonable weight. The explosive decks successfully cratered toward the central opening.

6. Where a sinkhole had a sloping ramp, open to the room below, it was necessary to drill holes along both sides of the ramp. The blastholes were increased in depth as the ramp extended deeper. Different numbers of decks had to be used as the hole depths changed. This method worked well for sloping ramps.

7. It was important to fill out the blast summary sheets while the blast was being drilled and prepared for loading. Also, taking time to prepare the explosive loading charts prior to loading the blast was essential to good success. Decked loading of holes requires careful placement of explosives. Also the hole depth and location of any void below is important to designing the hole loads. Therefore, any project of this type should use recording procedures like those described in the report for best results.

8. Drill cuttings provided adequate service for deck stemming and for stemming the top of the hole. Since drill cuttings are the only stemming material likely to be present on site the use of this material is dictated by availability and costs. Problems should not arise unless a site is very wet, when drill cuttings may provide poor stemming material.

9. For the vertical openings the use of 42 ms delays on surface works well. Nonel noiseless trunkline delays were used. These were easy to connect up and helped control noise generated by blasting.

10. Down-the-hole delays of periods 4 to 8 (100 to 200 ms) were used with good success. The delays were Nonel HD Primadets used with one pound slider primers. Generally, every other period was used giving a 50 ms delay between decks. In some cases 25 ms delays were used between upper decks. Either approach worked. The use of 50 ms between decks is recommended as usually giving the best result.

11. There was no problem manoeuvering equipment around the site. Care did have to be taken, when positioning the drill on a blasthole, to insure the equipment was safely located. Undercut sinkholes were the most difficult features in terms of being able to position the blastholes where needed and still be able to position the drill safely. When necessary holes were placed in less than optimum locations to insure safe operation around undercut vertical openings.

12. The use of a combination of ANFO and a waterproof heavy ANFO (70/30) proved quite satisfactory. The heavy ANFO was useful in the bottom deck for protection against any water accumulation and for added energy. Only ANFO was needed in upper decks.

13. Second blasts on a feature were loaded with all ANFO. So were four primary blasts. Results were quite satisfactory. Therefore, the use of all ANFO in most cases is acceptable and will reduce costs. However, in blastholes that do not intersect a void the use of HANFO is recommended to guard against water attack.

11.3.2 <u>Collapsing Irregular Underground Mine Workings</u>

1. These abandoned mine features can be reclaimed using either a method that places each blasthole directly over the workings to be collapsed or by placing holes on a regular pattern, such that some holes intersect the void while others are drilled in pillars.

2. Irregular workings at the White Test Site tended to have large intersects and small, variable pillars. The result was that upon firing the shot there tended to be considerable surface disruption and collapse beyond the blast area. This made it difficult to properly position blastholes for the next shot, adjacent to the blasted area. The potential then exists for unwanted voids to remain.

3. The blasted area profile could be controlled by using more stemming to limit surface displacement. At the same time full collapse of the workings was achieved. It was found that twelve to sixteen feet of stemming could be used successfully. When twenty-five feet of stemming was used, in one case, the void was not filled but moved closer to the surface. In general, for a 6-inch diameter blasthole, not more than sixteen feet of stemming should be used.

4. It was also helpful to attempt to finish the blast at a point where there were pillars and/or the intersections were narrow. The mine would be more stable at such locations and less likely to generate much unwanted collapse beyond the blast.

5. The deck loading system worked well in these cases. For holes over the void the successive decks cratered down into the voids well. For holes in pillars the shot cratered off the

side, toward the nearby void. Cratering action would be more exact for holes directly over the rooms but designing for cratering charges worked well in either case.

6. The use of seismic plugs to seal the bottom of the hole before loading was successful. There were no incidents of loaded blastholes slumping due to failure of the hole plug. Tying the plug off at surface using baler twine is recommended to insure explosive and primers are not lost into the room below due to plug failure. Loading poles are required to force the plug down to the bottom of the hole, so these should be available.

7. When using the area pattern blasting technique the attempt should be made to place as many of the holes in voids as possible. The more holes penetrating voids the better the performance. At the White location about fifty percent of the blastholes typically struck void.

8. Throughout most of the White test site the surface alteration from blasting was modest. Typically the change ranged from three feet of swell to three feet of surface depression. Swell was more common than slumping. This reflected the small voids usually found below the shot. The small change in topography meant that subsequent regrading needs would be small.

9. The water contained in the workings was not a major problem. The primary result was to increase time requirements for loading the holes with explosives. Upon shooting the blast there was some venting of water through open boreholes, but there was no seepage of water from the workings.

10. The presence of water made the use of heavy ANFO necessary in the bottom decks of holes that contained water. This site was representative of explosive selection requirements where the workings contain water.

11. The method of placing a hole plug above the water, followed by one to two feet of stemming and then placing the first ANFO deck worked well to guard the ANFO against water attack. An alternative, which would speed loading but also increase explosive costs, would be to use heavy ANFO in the second deck as well. Doing so would help to meet design explosive placement criteria.

12. The first blast at the White site closed that adit and main entry into the workings. The adit was thoroughly closed by drilling holes on the slopes around it and throwing material into the trench in front of the adit. It was further closed by holes which collapsed the roof of the adit as it extended into the workings. It is concluded that adits, driven into the side of a hill can be successfully closed by this technique.

where possible. Therefore, most were in the range of 13 to 16 feet. A few were 9 feet due to the need to place a powder deck in a quite short blasthole.

Typically, the holes that struck void had a scaled depth of burial to the surface of from 4.6 $ft/lb^{1/3}$ to 5.55 $ft/lb^{1/3}$. For pillar holes depths of burial were large and ranged from 5.55 $ft/lb^{1/3}$ to 7.46 $ft/lb^{1/3}$.

This shot provided successful collapse of the workings. Surface disturbance was modest, however, some severe cracking did result along the east side in an area where the mined intersections were large. Surface profile varied from 2 feet of depression to 3 feet of swell which is minimal and easily regraded level if desired.

Blast B7 was also an area pattern shot consisting of 20 holes on a 15x15 foot pattern. The stemming heights were similar to B6. Again there was a range based on the needs of individual blastholes. Stemming heights ranged from 12 to 18 feet. Thus scaled DOB ranged from 4.28 $ft/lb^{1/3}$ to 6.19 $ft/lb^{1/3}$. The most common stemming height was 13 feet (13 holes) which, with a 3-foot deck of ANFO gives a scaled DOB of 4.60 $ft/lb^{1/3}$.

Regarding surface disturbance this blast provided reasonable results. When detonated the area rose uniformly to an estimated height of 6 feet and then resettled. There were some areas of surface swell ranging from 0.5 to 3.0 feet high. There were no areas of depression. There were large, upended chunks scattered throughout the blast. Surface cracking extended a maximum of 15 feet.

The design, therefore, left a quite acceptable surface area after the shot, with minimum humping. Surface cracking did not interfere with subsequent production drilling. Surface cracks and chunks would be eliminated if regrading was performed.

Blast B7 left some voids where holes were drilled into pillars. Therefore, the explosive loads were somewhat increased for shot B8. One result was less stemming in the blastholes. In this case almost all the 65 holes had 10 feet of stemming. The top deck was a 3-foot deck of ANFO. Therefore, the scaled depth of burial was 3.65 $ft/lb^{1/3}$ to the top surface.

The result was that surface cracking outside the blast area extended 10 to 12 feet. There was some surface swell, but this did not exceed 4 feet in height. The area could be easily regraded level if desired.

Based on all the experimentation performed a scaled depth of burial of 5.0 $ft/lb^{1/3}$ is recommended to minimize surface disturbance and swell while insuring complete collapse of the area. For a 3-foot deck of ANFO this is a stemming height of 14 feet and for a 4-foot ANFO deck a stemming height of 16 feet is required. In most cases this design will give a minimum of surface heave and cracking, but will provide explosive sufficiently high in the bank to insure collapse.

It is important to recognize that adjustments of this kind can best be made when the voids are small in comparison to the overburden above the workings. In these cases the swell of the blasted material will fill the void and allow the total bank to remain near the same elevation or somewhat swelled. Then, a profile can be achieved that is easily regraded and avoids mixing of the soils.

When the rooms are large relative to the cover above the workings this is less successful. In such cases collapse of the room can leave depressions of as much as 10 to 15 feet. Regrading need increase significantly. Also soils become more mixed, although a substantial proportion of growth material remains near the top.

Based on the results at the White location it is possible to postulate that for quite deep cover, relative to the void dimensions, the swelled overburden would close the void without the need to blast all the way to surface. In these cases the area could be reclaimed with little or no surface disturbance whatsoever. Thus, prior topsoil removal and post-blasting regrading could be eliminated while leaving the site in good condition. Previous studies indicate an average of 18 percent swell occurs (4). To be conservative estimates of the ability to close the voids should be made based on a 12 percent swell factor.

9.3 BLASTS IN INDIVIDUAL ROOMS

It was found that a primary concern for blasting in an environment such as that presented at this test location is the ability to identify the location of the abandoned workings. For the one acre test site this was a manageable problem, but would become increasingly difficult for larger sites. Both time and cost would be significant factors.

These results are in contrast to results in regular workings for which good documentation is available (4). Exploration time and cost was quite reasonable in those cases.

Once the rooms are located successful collapse can be effected by placing blastholes above the works and collapsing material into the void below. It is important that the blastholes be accurately placed for best results. The use of wooden stakes is a successful way to locate the blastholes in the field. The prime problem with this approach however is the high cost of exploration work.

Another problem that occurred when blasting in individual rooms was difficulty in controlling the area caved. This also was unique to this setting and had not been experienced in previous work on rooms that were of consistent width and direction, with uniform rib pillars (18 feet wide) between each room. At the White site the irregular sized pillars, which were randomly positioned, and the wide intersections made it much more difficult to control surface disturbance from a blast.

The resulting difficulty is that results is that zones outside the blast area, that partially cave, become difficult to access with the drill for the next shot. Therefore, the possibility exists that there will be pockets of uncollapsed void but the drill can not be repositioned to drill off the area. If the equipment can be repositioned it was found that drilling had to be accomplished through broken material which is difficult and it is less certain that the hole can be held open until it can be loaded with explosive. The use of 6-inch diameter PVC pipe to collar such holes was quite successful in keeping them open for drilling and loading.

An important goal, then, is to avoid the disruption. This can be partly achieved by increasing the stemming at the top of the hole, thereby increasing the explosive burial relative to the surface. The approach has been discussed above.

A second solution is to examine the mine map developed from the exploration program and find a place to stop the blast that is less likely to result in unwanted collapse. Such a spot would be one with good pillars nearby and an area that does not have unusually large spans. A blast such as B4 for example would be less likely to create problems than B1, which stopped in a large, relatively unsupported void.

By controlling the stemming height and the blast orientation the problem of unwanted collapse could be controlled reasonably well. However, where there are wide intersections it is not likely that the situation can be fully resolved. This was proved in blast B3 where steps to reduce backbreak, such as increasing stemming and reducing powder in the bottom deck reduced cracking to 15 feet (from 25 to 30 feet) but did not eliminate it. Cracks behind the middle section of B6 also indicated that large intersections could still be a problem.

At the White test location successful methods of collapsing individual works were found. However, the method was not entirely satisfactory because of the extensive exploration required to identify the works and the problems with unwanted collapse. Both of these characteristics contribute to added time and cost for performing the work.

9.4 AREA BLASTS

As described previously area blasts were drilled on a regular pattern without regard for the location of the workings. Therefore, some holes intersected void while others did not. All holes were loaded and detonated in the blast. For blast B6 And B8 fifty to sixty percent of holes struck the void.

The method reduces exploration requirements. The T.V. camera was eliminated. This was necessary in any event since the workings in the area of B6 to B8 contained much water and the camera could not be used in such conditions

Also, prior exploration drilling can be reduced since there is not the same need to identify the workings as for the case of shooting in individual rooms. Drilling would be primarily to identify the perimeter of the mining and to provide a general idea of void locations and dimensions. Exploration drilling can be reduced by at least one-half when area blasting is employed.

When an area blast is shot those holes that have intersected a void can be expected to crater down into the rooms below. Crater design data applicable to these situations have been provided above. Blastholes that are drilled into pillars will crater to the side. However, it cannot be exactly known where these holes are relative to the nearest void. Therefore, it was found that the bottom deck, located in the pillar, should be loaded more heavily than blastholes situated over the void. One to two additional feet of powder is required. This need was based on the evidence of blast B7, where some voids near pillars were not completely collapsed initially. Therefore, in Blast B8 the powder consumption in the lower decks was increased from 45 to 60 pounds of heavy ANFO.

The deck loading and cratering approach was successful for the area shots. As before, the powder loads were designed to generate cratering effects. Therefore, powder factor is not a good benchmark for design. However, for comparison the area blasts on a 15x15 foot pattern, assuming an average hole depth of 35 feet gave a powder factor of 0.41 pounds per cubic yard. On a 12x12 pattern the powder factor was approximately 0.64 pounds per cubic yard.

Area shots created quite a uniform surface disturbance. At the White site this was primarily a few feet of surface swell with some small depressions also seen. With properly chosen stemming heights cracking and backbreak beyond the blasting area was quite well controlled. By contrast shooting individual rooms left a more variable profile. Also, backbreak was as much as 30 feet behind the blast for individual room shots. However, in later individual room shots this was controlled to less than 15 feet.

Based on the results of shots B6 to B8 the 10x10 foot pattern is smaller than necessary for an area blast in these types of materials. The 12x12 foot pattern provides quite acceptable results. It would be a good pattern for starting a new reclamation project.

The 15x15 foot pattern also produced good results, provided there was adequate powder load. Each hole required 120 pounds of explosives of which 60 pounds was placed in the bottom deck and 30 pounds were placed in the middle and top decks.

The best approach to area blasting on similar sites would be to start with a 12x12 pattern. For a hole with two decks the bottom deck should contain 60 pounds of HANFO, which will give a 4 to 5 foot rise. The upper deck would consist of 30 pounds of ANFO (3 feet). If three decks are needed then each of the upper two decks would contain 30 pounds of ANFO. Initial stemming heights should be about 14 feet.

If results warrant the patterns could be expanded toward 15x15 feet with the same explosive loading. If surface disruption were more than desired stemming heights could be increased to 16 feet.

The area pattern blasts shot at the White site did not suffer from backbreak problems affecting the next shot to the extent that the individual room blasts did. Also surface swell was modest and more uniform than for the individual shots. However, some of the surface disruption of the individual blasts was eliminated when longer stemming lengths were employed.

More importantly the area blasting technique reduced time consuming and costly exploration requirements If a site were completely reclaimed using this technique such costs would be considerably reduced.

Area blasting will be most economical where extraction ratios were high meaning a large proportion of the area was undermined. When recovery was low increasing numbers of holes will be in pillars rather than void and the cost will tend to increase on an extraction basis relative to blasting individual rooms with holes located only above the voids.

9.5 BLAST TIMING

The timing of the blasts was similar to that used at Beulah. A prime consideration was to allow adequate time for each charge to crater toward the void below before the next charge in the hole detonated. Also, charges cratering in succession would provide some additional relief to the side for subsequently detonating charges.

At the White Site all blasts provided 50 ms of delay between individual explosive decks in a hole. To accomplish this every other period down the hole delays were used. These ranged from 100 ms to 200 ms and were number 4, 6 and 8 period delays used in conjunction with detonating cord and slider primers.

To delay between holes noiseless trunkline delays were used. In all cases these were 42 ms surface delays.

The area shots were delayed somewhat differently than the individual room shots. For these blasts multiple holes were detonated per surface delay. This was possible because the distance to buildings was long and vibration was not a major concern.

Shooting multiple holes per delay would add to the particle velocities near the shot. However, the down-the-hole delays result in only one deck firing per blasthole simultaneously. Therefore, this helped to control close-in vibration. Even with considerably more explosive weight detonated per delay there was no evidence of collapse outside the blast area due to high vibration level. Also, the higher vibration levels did not create more surface cracking and disruption than was experienced for the individual room blasts, where holes were independently delayed.

The multiple hole detonations did lead to higher vibration levels at the seismograph location, as reported in Chapter 7. This was not an important concern due to the remote nature of the site. However, it is important to realize that vibration levels could be further reduced by adding more surface delays if one was reclaiming a site near to residences or other buildings.

The combination of 42 ms surface delays and 50 ms between decks down-the-hole worked well. This method is quite satisfactory when shooting individual rooms. If a site is to be reclaimed using area pattern shots then one should give added consideration to the surface delay combinations. For example, if it would be advantageous to detonate the shot on a chevron configuration, then a combination of 42 ms and 17 ms surface delays, or other suitable combination, could be used to delay between chevrons and along each individual chevron. The goal, as before, would be to maximize the ability for overburden to move toward and fill the mined rooms.

9.6 EFFECTS OF WATER IN THE WORKINGS

The White site location afforded the opportunity to blast in some rooms that were water filled. The east side of the property, where the pattern blasts were performed was the area containing water.

It was necessary to use the waterproof heavy ANFO in the bottom deck of the blastholes in these shots, unless the drill hole struck a void at shallow depth. This was a clear example of a property where a waterproof product has important application. The most difficult holes were those that were drilled in pillars as these exhibited the greatest water rise.

Holes that intersected a void could have some water rise depending on water conditions in the works below.

Changes had to be made to the loading procedures in those holes that contained water. After the bottom deck of heavy ANFO was loaded a seismic hole plug was placed just above the water level. Then, one to two feet of cuttings were placed on the plug followed by the second explosive deck, consisting of ANFO. In this manner the ANFO deck was protected against water attack.

Depending on the amount of water present this method could lead to longer spacings between decks than would be optimum. Therefore, another approach would be to load a waterproof product in the second deck as well. Doing so would increase the explosives cost, but would also assure the best placement of the explosive and protection against water attack. However, plugging the hole above the water rise and using ANFO did work at the test location.

Prior to blasting on this site there was some concern as to the effect of water on blasting effectiveness. Observation at the site indicated little difference in results in wet and dry areas. Some water was forced out of the area by venting through unloaded exploration drillholes. There was no slumping observed that could be attributed to slurrying and flow of the blasted overburden. Observation of the site in the summer of 1989 indicated that time did not result in such effects either.

The primary effect of the presence of water then was to increase the time taken to load the shot. The extra time resulted from added difficulty loading the bagged HANFO in wet holes and the time to place the plugs and backfill. Provided proper wet hole loading procedures were used there was not a problem with reduced performance of the explosive.

It was harder to assess what happened to the water upon detonation of the blast and what effect this could have on overall site reclamation, because there is no means of directly studying the situation. At this location only a portion of the workings were water filled and the indication is that displaced water was either vented or absorbed in the workings as a whole. For a site where most or all of the works were water filled it would be wise to monitor for seepage of water from the reclaimed mine. This would be especially important if the mine contained acidic water or water with other contaminants. Provision for the collection and treatment of such water might be required.

The results of this research indicate that such problems are not likely to arise unless the mine as a whole is substantially water filled.

9.7 BLASTING TECHNIQUE VERSUS MINING RECOVERY

In many older mines the amount of coal extracted was low. Thus, as little as twenty-five percent of the area may have actually been mined. In other cases, if there was good stability of the workings, extraction could be considerably higher.

At the White test site the extraction was substantial as can be seen in the mine map in figure 3-10 found in Chapter 3. Interior pillars were often small and were located in a haphazard fashion.

For this type of site area blasting can prove quite economical. The number of holes needed to pattern blast over the area may not be much more than individual room blasthole requirements. For example, area pattern blasting all of the White Site on a 15x15 foot pattern would have required approximately 195 blast holes. This compares with 178 holes actually shot at the White location, of which 95 blastholes were located in individual shots in blasts B1 to B5. Given the reduction in exploration time and effort required overall efficiency and cost would be enhanced using the pattern approach.

The area approach will be more useful in cases where the extraction was substantial because more blastholes will intersect the void and crater down into it. Pillars will be smaller, so that holes intersecting pillars will be closer to a void and more effective.

The area method is sensitive to the pattern dimensions as well. If a 12x12 pattern were necessary at the White test area then 306 holes would be needed. Any savings would then depend on the ability to greatly reduce exploration costs.

When the extraction is low shooting on an individual room basis will reduce the number of production holes and explosives needed to reclaim the site. Also exploration costs could be less because there are fewer workings to find.

Therefore, the choice of method will be related to the degree to which the site was mined. Both methods work and the choice will be related to cost. Since individual sites can vary substantially such an analysis will need to be made based on costs and technical specifics pertaining to the given location.

CHAPTER 10
PROJECTED COSTS FOR PERFORMING AML RECLAMATION BY BLASTING ON A PRODUCTION BASIS

10.1 INTRODUCTION

Cost analyses have been performed to estimate the cost of reclaiming AML sites using blasting methods. The costs are based on performing the work on a production basis and, therefore, do not include costs associated with research components of the current study. Such costs include high-speed camera work, research related field labor, data analysis and report writing. Thus, the costs used are strictly related to blasting on a production basis.

Some advance engineering work is required when blasting old mining sites. However, engineering is needed no matter what mitigation technique is chosen. Therefore, costs for preliminary engineering design and planning are not included in the analysis.

Exploration work is necessary to identify the location and depth of old workings. This need is somewhat specific to the use of blasting to reclaim old works. Also, different blasting methods have different exploration requirements. Therefore, exploration costs have been estimated for these methods, which include vertical opening closure, blasting over individual works and area pattern blasting.

The unit costs used in the analysis are based on the costs experienced for this research. If necessary, modifications have been made to reflect a production rather than research and development orientation.

Table 10-1 lists unit costs for drilling and explosives that have been used in the analysis.

TABLE 10-1: UNIT DRILLING AND EXPLOSIVES COSTS USED IN THE COST ANALYSIS

Item	Beulah Test Site	White Test Site
Drilling	\$0.85/ft.	\$1.00/ft.
ANFO	\$17.15/CWT	\$18.075/CWT
Heavy ANFO	\$36.75/CWT	\$36.82/CWT
Surface Delays (42ms NTD)	\$263.50/C	\$267.50/C
Down-The-Hole Delays (#4-#8)	\$223.50/C	\$225.50/C
One Pound Cast Primers	\$3.57 ea.	\$3.50 ea.
15 Grain Detonating Cord	\$80.50/M ft.	\$81.00/M ft.

The costs differ slightly between sites, but are largely comparable. The drilling cost increase was due to the more remote nature of the site. At Beulah the drilling contractor could leave the site to perform other work and return when required. For the White site the equipment and crew were more than three hundred miles from home base and, as such, were captive to the site.

The ANFO and heavy ANFO costs reflect the cost of purchasing explosives in small lots of bagged powder. The larger the order that can be placed the more favorable the pricing. On a bulk basis the ANFO cost could be reduced by 4 to 5 cents per pound and the HANFO price could similarly be reduced by 10 to 15 cents per pound for the 70/30 emulsion to ANFO mix. However, in AML work small lot pricing will commonly be experienced unless a site is unusually large. Therefore, the pricing in the table ought to be quite realistic for most AML locations.

Table 10-2 lists other equipment and supply needs for the site.

TABLE 10-2: EQUIPMENT AND ACCESSORY COSTS FOR AML BLASTING

Item	Beulah Test Site	White Test Site
Explosive Magazine	$400/mo	$250/mo.
Seismograph	$775/mo	$605/mo
Field Office Trailer	$450/mo	$400/mo
Blasting Equipment	$5/hr	$8/hr
Stakes and Paint	$22.48	$20.45
EPCO Holes Plugs	$345.00	$345.00
Miscellaneous Items	$82.00	$82.00

The field office trailer was necessary for the current work because the research required considerable design and data recording in the field. It also provided storage for seismographs, high-speed camera, film, etc. For a small production project the trailer likely could be eliminated. For a large project, performed over a period of time, the field office facility is needed.

The blasting equipment included the use of a suitably outfitted pickup for loading explosives, blasting machine, blasting galvanometer, lead wire, blasting signs and siren. All equipment complied with blasting regulations.

Table 10-3 lists the labor costs estimated as being appropriate to this work. It is assumed that the drill crew labor cost is included in the drilling footage cost. Experience at the White site indicated that, for larger shots, it may be necessary to have the drill crew assist in blasthole loading.

TABLE 10-3: PROJECTED UNIT LABOR RELATED COSTS FOR AML BLASTING PROJECT

Labor Type	Hourly Costs
Blasting Engineer	$39.75
Blaster	21.75
Blaster Helper	14.50

These rates reflect base salary or wage and all payroll taxes and benefits. It is considered that a three-man crew will be appropriate. It is expected that the blasting engineer will assist in loading holes and other tasks associated with preparing and detonating the blasts. For small, straight forward blasts time requirements for the blasting engineer may well be considerably reduced.

An additional cost is that of transportation and subsistence for persons working on site. This is difficult to estimate because it depends on whether drilling and blasting expertise is available near to the site or whether it requires crews to work at remote locations.

For this cost analysis, travel costs for the Beulah site have been limited to transportation costs only. This reflects the fact that the contractor was located near the site.

For the White site additional travel costs have been assigned. This test area was over three hundred miles from the location of the contractor. Therefore, added costs were necessary.

Thus these analyses portray both the situation where travel is a small item and where it is a larger factor due to remoteness of the site. Specific information on how travel costs were applied is given below.

10.2 BLASTING VERTICAL OPENINGS - BEULAH SITE

10.2.1 Cost Calculations for Vertical Openings

For the closure of vertical openings, eighteen blasts including 199 blastholes have been analyzed. The nature of each blast has been described in Chapter 5.

The cost analysis of closing vertical openings is best performed on a per opening basis. This provides better results than attempting a cost per acre since there could be great variation from site to site in the number of sinkholes found on an acre of mined land.

The costs are based on just the production requirements to mitigate the features. As described in the introduction to this Chapter research and development components of the work are excluded.

Table 10-4 lists the costs for each sinkhole blast at the Beulah Site. These have been itemized in seven categories as shown in the table. Totals and average costs are also provided.

The exploration cost for each vertical opening are those that were listed in Chapter 3. There were no exploration costs for the second shot on a sinkhole, where such blasts were necessary.

The drilling cost for a sinkhole is based on the footage obtained from the field data sheets. The Beulah drilling rate of $0.85 per foot is applied to this footage to obtain the total drilling cost for the vertical opening.

The consumption of explosive by type was maintained for each blast. For some shots there was a part bag of explosive left after loading for which the weight had to be estimated. Therefore, ANFO weights could be off by a maximum of 25 pounds and HANFO weights by as much as 15 pounds. These quantities will not represent significant variations in the cost projections. The unit explosives costs have been listed in table 10-1.

The quantities of primers used was recorded for each blast. So too were the number of down-the-hole and surface delays. Detonating cord consumption was estimated on the total hole footage and the pigtail lengths on surface. These estimates were within twenty-five feet so the maximum cost variation is $2.00 per blast, an insignificant sum.

TABLE 10-4: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH.

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$456.00	$1,639.48
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$380.00	$1,496.74
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$228.00	$859.68
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$304.00	$1,036.18
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$228.00	$579.56
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$532.00	$1,315.27
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$608.00	$1,531.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$304.00	$849.53
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$760.00	$2,745.93
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$228.00	$743.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$608.00	$1,922.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$152.00	$545.79
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$532.00	$1,568.15
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$228.00	$665.37
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$190.00	$641.04
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$456.00	$1,895.72
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$380.00	$1,338.72
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$456.00	$1,988.69
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$7,030.00	$23,363.61
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	390.56	1297.98

The cost listed as accessory and supply cost were those fixed project expenses which were listed in table 10-2. Added to those were transportation costs for getting to and from the site. Total transportation cost was estimated at $972.05.

It is difficult to determine how best to incorporate these site costs. In this case the accessory and supply cost has been divided equally among the blasts at the site. The total estimated cost was $3,782.00 for this category. Divided among the 18 blasts fired at Beulah, the unit cost becomes $210.11 per shot.

This approach is not exact and in reality some variation would occur. However, if the estimate was made on the ratio of holes in a blast to the total number of blastholes, for example, the result would not reflect that some of these costs do not vary. Therefore, applying these charges equally among the blasts was deemed most appropriate.

The labor charges are based on an estimate of the amount of time spent directly preparing to detonate a blast. This varies by the size and complexity of the shot. Blasts with more decks, for example, take longer to load. Blasts having more holes that strike a void below and require a plug also take longer to load, as the plugs must be placed using loading poles. The labor required for research purposes such as design, measurement, analysis and camera work is not included here.

The proposed labor crew is that needed for performing the work on a production basis. It is recommended that the crew include an experienced blasting engineer. This type of blasting is never routine and technical expertise on site is important to good results. In addition, when blasting near residences vibration must be carefully controlled and a suitable technical background is necessary to perform appropriate designs.

In table 10-4 the total cost for each veritical opening is provided. The totals by category are also shown. The average cost for each category and the total effort are also shown in the table.

10.2.2 Cost Analysis for Vertical Openings

Observation of table 10-4 shows that the cost varies considerably from one sinkhole to another. The most costly openings to close are those that include a large depression and an open slope down into the mined room. The most notable examples are VO-7 & 8-N, VO-7 & 8-S, VO-11 and VO-13. Vertical openings of this sort require large shots and are, typically, the type that may require a second shot to close.

Openings that are undercut, providing a profile that is small at surface but large at the base, are the next most costly. These features are difficult to drill around and the collapse is more difficult to control. Openings VO-9 and VO-12 are the examples of this condition.

Finally, other openings, that are smaller and do not have an open ramp to the works, appear to require the least cost to close. Examples of this type of opening are VO-3, VO-6 and VO-10. Costs to mitigate these features are quite economical

Table 10-4 provides average values for each cost category and for the total cost to close the feature. Averages are somewhat misleading due to the great variation in size and form of individual vertical openings. However, based on the sinkholes shot at Beulah, it is estimated that the average cost to mitigate a feature is $1,297.98.

Some of the blasts listed are the second shot on the same opening. The average above is based on the total eighteen blasts. When the costs on the same sinkhole are combined then a total of fourteen features were reclaimed. Applying this number the average cost is $1,668.83 per vertical opening blasted closed.

This second average is likely the more accurate value. It accounts for the periodic need for more than one blast to fully close a single feature. Since this need is likely to occur on any site the average obtained on this basis is more representative. Since the blasts at Beulah covered a wide range of opening types and sizes the value will be good for estimating purposes for any projects in similar materials and depths.

Figure 10-1 is a chart illustrating the category costs by percentage. The category labeled primers on the chart includes surface and down-the-hole delays and detonating cord.

The graph illustrates that, for vertical openings, exploration costs are modest. Since the workings below can often be seen it takes fewer exploration holes to determine the trend of the workings.

Labor is the single largest cost. It is almost double the next highest cost. Therefore, efficient use of labor will be a key to optimizing project costs.

When the two explosive cost percentages are combined the remaining categories represent quite similar portions of the costs. Combined these are 64.6 percent of the total cost to close a sinkhole.

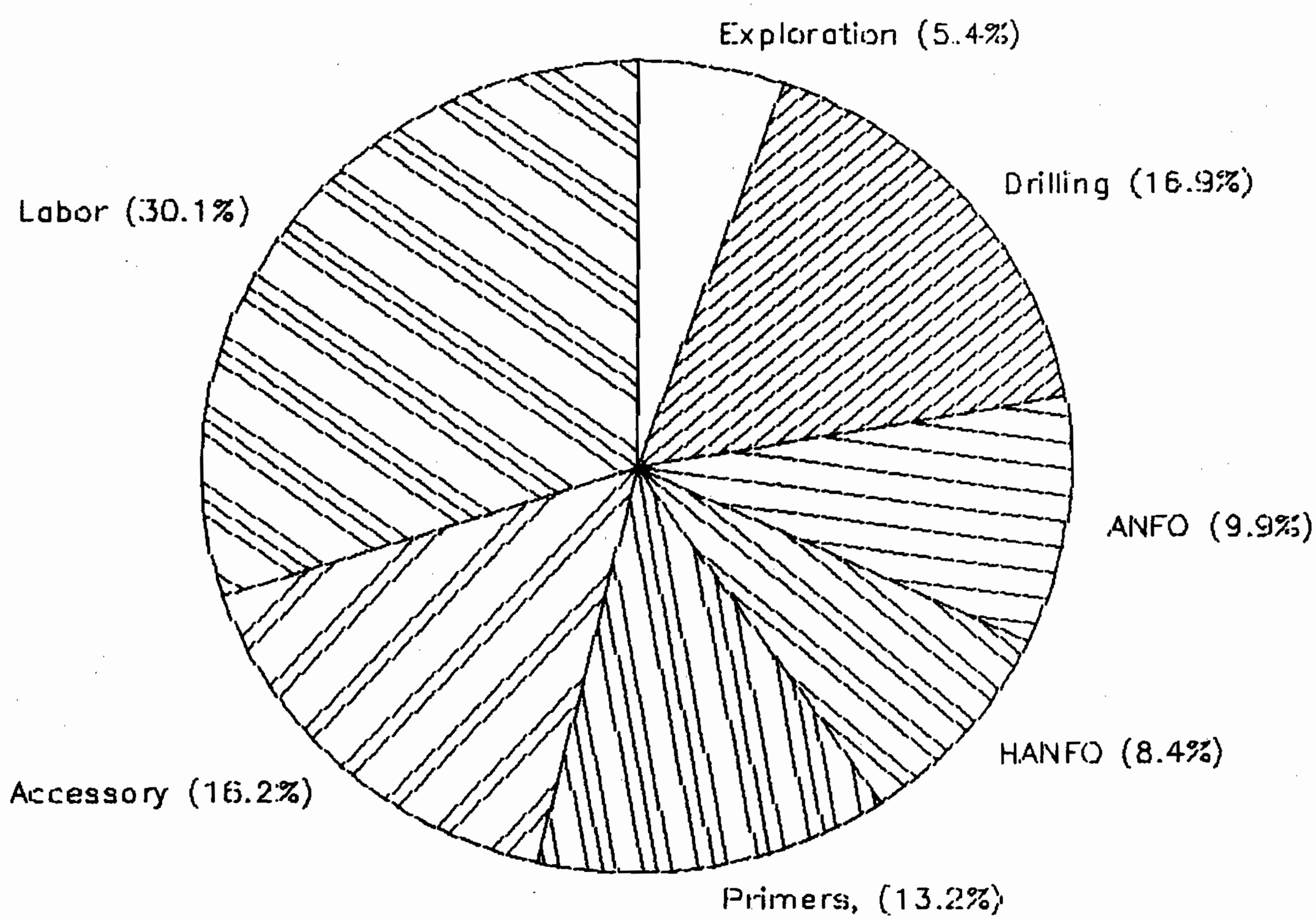
There are few savings to be made in exploration costs. Substituting ANFO for HANFO wherever possible would reduce costs. Using a primer of less than one pound weight could also be a feasible way to reduce cost to some extent.

For these shots a quite favorable drilling cost was obtained. Therefore, this category is likely to be most sensitive to cost increase. However, one sees contract drilling

FIGURE 10-1

ESTIMATED PRODUCTION BLASTING COSTS

Vertical Openings



costs in 6½-inch diameter in granite quarries being performed for \$1.69 to \$2.00 per foot. Therefore, it is unlikely that drilling costs would exceed \$1.50 per foot in this application.

10.2.3 Variations in Costs

Some factors may cause the cost of closing a sinkhole to increase or decrease. The factors which could have the greatest effect are listed below:

- Changes in unit drilling cost
- Use of ANFO for all explosives loading
- Changes in labor cost

The impact of these factors has been analyzed and are discussed below.

10.2.3.1 Changes in Unit Drilling Costs

The rate obtained for drilling for this research project was quite favorable and it is unlikely that a lesser rate could be realized. However, increases would be a possibility, especially when drilling rigs are less available due to high levels of available work. As a result the Beulah blasts have been analysed for a drilling cost of \$1.50 per foot as opposed to the \$0.85 per foot used in the original estimate.

The resulting increase in drilling costs can be seen in table 10-5. The average cost per blast has risen from \$219.02 to \$386.50 an increase of 76 percent.

The total cost per blast has risen from an average of \$1,297.98 to \$1,465.46 which is an increase of \$167.48 or 13 percent. The average, on the basis of fourteen sinkholes blasted is \$1,884.16 as opposed to the \$1,668.83 calculated earlier also a cost increase of 13 percent.

The increase in unit drilling cost results in the drilling category representing 26.4 percent of the total cost. It is equal to the labor cost which now equals 26.7 percent of the total. Other requirements are reduced accordingly.

The analysis does indicate that the cost of reclaiming vertical openings by blasting is sensitive to the drilling cost. If the cost of drilling were to increase to more than \$1.50 per foot drilling would become the greatest single cost involved in mitigating these features.

On the other hand a cost rise to \$1.50 per foot would not preclude the use of blasting to close vertical openings. The total increase is, on average, about \$200 per sinkhole which is manageable. An increase to \$2.50 per foot, which would increase the cost of reclaiming an average feature to \$2,215.45, could

TABLE 10-5: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. DRILLING COST INCREASED TO $1.50

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$547.06	$120.05	$242.55	$214.07	$210.11	$456.00	$1,876.54
VO-2	7	$354.45	$328.13	$42.86	$209.48	$113.90	$210.11	$380.00	$1,638.93
VO-3	5	$96.90	$221.26	$25.73	$99.23	$74.33	$210.11	$228.00	$955.56
VO-4A	8	$38.25	$243.00	$42.88	$176.40	$126.84	$210.11	$304.00	$1,141.48
VO-4A#2	3	$0.00	$90.00	$42.88	$0.00	$47.57	$210.11	$228.00	$618.56
VO-4B	8	$33.15	$336.00	$102.90	$99.23	$147.48	$210.11	$532.00	$1,460.87
VO-5	13	$68.85	$394.50	$102.90	$132.30	$185.31	$210.11	$608.00	$1,701.97
VO-6	6	$93.50	$168.00	$68.60	$0.00	$78.12	$210.11	$304.00	$922.33
VO-7&8-N	28	$66.30	$1,068.00	$377.30	$242.55	$484.47	$210.11	$760.00	$3,208.73
VO-7&8N#2	9	$0.00	$202.50	$102.90	$0.00	$88.04	$210.11	$228.00	$831.55
VO-7&8-S	20	$60.35	$750.00	$282.98	$0.00	$336.54	$210.11	$608.00	$2,247.98
VO-7S#2	4	$0.00	$120.00	$51.45	$0.00	$64.23	$210.11	$152.00	$597.79
VO-9	10	$110.50	$333.00	$68.60	$297.68	$160.56	$210.11	$532.00	$1,712.45
VO-9#2	6	$0.00	$154.50	$68.60	$0.00	$71.11	$210.11	$228.00	$732.32
VO-10	5	$73.10	$123.00	$42.88	$0.00	$55.25	$210.11	$190.00	$694.34
VO-11	22	$55.25	$747.00	$334.43	$77.18	$339.45	$210.11	$456.00	$2,219.42
VO-12	14	$89.25	$447.00	$214.37	$0.00	$191.69	$210.11	$380.00	$1,532.42
VO-13	21	$37.40	$684.00	$214.37	$374.85	$308.36	$210.11	$456.00	$2,285.09
TOTALS	199	$1,263.95	$6,956.95	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$7,030.00	$26,378.29
Average	11.00	70.22	386.50	128.15	108.41	171.52	210.11	390.56	1465.46

begin to make the approach less competitive with other techniques. However, $2.50 per foot would represent a high unit cost in overburden that drills easily. Large diameter blasthole drilling costs in surface coal mines, by contrast, seldom exceed $2.00 per foot.

10.2.3.2 Effect of Explosives Usage

Costs could be reduced if only ANFO were used and the HANFO were eliminated. The savings would be $0.196 per pound. The basic ANFO product has adequate energy output to perform as required except, possibly, in cases where a dangerous or undercut sinkhole requires a greater than usual drill setback from the rim of the vertical opening. Generally, ANFO will be acceptable.

Table 10-6 present the results when all explosive is ANFO. The average cost for explosive drops from $236.56 per feature on average to $178.74 per feature. The total cost per sinkhole decreases from an average of $1,297.98 to $1,240.16 per opening.

The ANFO cost is now 14.4 percent of the total. This compares to a combined percentage of 18.3 percent where a combination of both products was considered (figure 10-1).

This decrease is not particularly significant. The reasons are the HANFO was used only in the bottom deck, it was not used for second shots on an opening and it was not used for three initial blasts on a sinkhole. Therefore, the quantities being replaced are not large and the cost decrease is limited.

The use of HANFO in the bottom deck can guard against water attack. It also adds energy at the toe where burdens may be long. Therefore, replacing the HANFO does not give a sufficient cost decrease, when blasting vertical openings, to warrant the loss of technical advantages.

A more significant situation results if all waterproof HANFO is used. This could occur if a property was particularly wet and there was concern about the use of ANFO, which is very susceptible to water attack.

Table 10-7 contains the data for the situation where the blasts at Beulah are assumed to be performed using only HANFO. In this case the average explosive cost increases to $383.02 an increase of 62 percent over use of the combined products. The average cost per blast rises to $1,444.43 an increase of $146.45 per blast. Explosives are now 26.5 percent of the total cost and are equal to the labor cost involved.

TABLE 10-6: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. ANFO ONLY

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$233.24	$214.07	$210.11	$456.00	$1,510.12
VO-2	7	$354.45	$185.94	$140.62	$113.90	$210.11	$380.00	$1,385.02
VO-3	5	$96.90	$125.38	$72.04	$74.33	$210.11	$228.00	$806.76
VO-4A	8	$38.25	$137.70	$125.20	$126.84	$210.11	$304.00	$942.10
VO-4A#2	3	$0.00	$51.00	$42.88	$47.57	$210.11	$228.00	$579.56
VO-4B	8	$33.15	$190.40	$149.21	$147.48	$210.11	$532.00	$1,262.34
VO-5	13	$68.85	$223.55	$164.64	$185.31	$210.11	$608.00	$1,460.46
VO-6	6	$93.50	$95.20	$68.60	$78.12	$210.11	$304.00	$849.53
VO-7&8-N	28	$66.30	$605.20	$490.49	$484.47	$210.11	$760.00	$2,616.57
VO-7&8N#2	9	$0.00	$114.75	$102.90	$88.04	$210.11	$228.00	$743.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$336.54	$210.11	$608.00	$1,922.98
VO-7S#2	4	$0.00	$68.00	$51.45	$64.23	$210.11	$152.00	$545.79
VO-9	10	$110.50	$188.70	$207.52	$160.56	$210.11	$532.00	$1,409.39
VO-9#2	6	$0.00	$87.55	$68.60	$71.11	$210.11	$228.00	$665.37
VO-10	5	$73.10	$69.70	$42.88	$55.25	$210.11	$190.00	$641.04
VO-11	22	$55.25	$423.30	$370.45	$339.45	$210.11	$456.00	$1,854.55
VO-12	14	$89.25	$253.30	$214.37	$191.69	$210.11	$380.00	$1,338.72
VO-13	21	$37.40	$387.60	$389.30	$308.36	$210.11	$456.00	$1,788.77
TOTALS	199	$1,263.95	$3,942.27	$3,217.36	$3,087.28	$3,781.98	$7,030.00	$22,322.84
Average	11.00	70.22	219.02	178.74	171.52	210.11	390.56	1240.16

TABLE 10-7: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. HANFO ONLY

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$499.80	$214.07	$210.11	$456.00	$1,776.68
VO-2	7	$354.45	$185.94	$301.32	$113.90	$210.11	$380.00	$1,545.72
VO-3	5	$96.90	$125.38	$154.37	$74.33	$210.11	$228.00	$889.09
VO-4A	8	$38.25	$137.70	$268.29	$126.84	$210.11	$304.00	$1,085.19
VO-4A#2	3	$0.00	$51.00	$91.89	$47.57	$210.11	$228.00	$628.56
VO-4B	8	$33.15	$190.40	$319.73	$147.48	$210.11	$532.00	$1,432.87
VO-5	13	$68.85	$223.55	$352.80	$185.31	$210.11	$608.00	$1,648.62
VO-6	6	$93.50	$95.20	$147.00	$78.12	$210.11	$304.00	$927.93
VO-7&8-N	28	$66.30	$605.20	$1,051.05	$484.47	$210.11	$760.00	$3,177.13
VO-7&8N#2	9	$0.00	$114.75	$220.50	$88.04	$210.11	$228.00	$861.40
VO-7&8-S	20	$60.35	$425.00	$606.39	$336.54	$210.11	$608.00	$2,246.39
VO-7S#2	4	$0.00	$68.00	$110.25	$64.23	$210.11	$152.00	$604.59
VO-9	10	$110.50	$188.70	$444.68	$160.56	$210.11	$532.00	$1,646.55
VO-9#2	6	$0.00	$87.55	$147.00	$71.11	$210.11	$228.00	$743.77
VO-10	5	$73.10	$69.70	$91.89	$55.25	$210.11	$190.00	$690.05
VO-11	22	$55.25	$423.30	$793.82	$339.45	$210.11	$456.00	$2,277.92
VO-12	14	$89.25	$253.30	$459.36	$191.69	$210.11	$380.00	$1,583.71
VO-13	21	$37.40	$387.60	$834.21	$308.36	$210.11	$456.00	$2,233.68
TOTALS	199	$1,263.95	$3,942.27	$6,894.34	$3,087.28	$3,781.98	$7,030.00	$25,999.82
Average	11.00	70.22	219.02	383.02	171.52	210.11	390.56	1444.43

The conclusion from this analysis is that the use of heavy ANFO should be limited to only those situations requiring such a product. These include the presence of water or large burdens. However, elimination of all HANFO is not necessary and its use in the bottom decks does have advantages as described above.

10.2.3.3 Effect of Changing Labor Cost

Labor costs could be reduced if the blasting engineer position was eliminated. As stated earlier in the Chapter there are technical reasons why this may not be wise, but the analysis is provided for comparison purposes.

Table 10-8 illustrates the effect of this change. Only a blaster and helper are engaged in the work. The hours have been adjusted to provide the same total manhours as before. Compared to the original case the average cost of labor is reduced from $390.56 per shot to $279.43. The total cost to blast an opening reduces from $1,297.98 to $1,186.85, a $111.13 reduction. The two man crew reduces the labor portion of the cost from 30.1 percent to 23.5 percent.

The other factor that affects the cost is the required time to load and shoot a blast. If actual requirements are greater than estimated above the cost will increase.

To examine the effect of this a twenty-five percent increase in required time has been studied. Table 10-9 presents the results for a three-man crew. In this case the average labor cost to shoot a blast increases to $488.19. The total cost per shot increases to $1,395.62 an increase of $97.64 per shot. The labor cost is then 35 percent of the total.

Table 10-10 provides the results of the same time increase when a two-man crew is utilized. The labor cost per blast is now $349.28 and the average cost of a shot is $1,256.71. The labor cost as a percent of the total is 27.8 percent. Thus, the two-man crew, with a twenty-five percent increase in the time required to perform a shot is about equivalent to the cost of the original three-man crew.

The results of the analyses indicate a range from about a $100 per shot decrease to a $100 per shot increase in cost. The $100 decrease by eliminating the blasting engineer is not likely warranted. Elimination of this position could adversely affect the project and, in total, cause an increase in cost.

The twenty-five percent increase in time is manageable. The increase beyond the estimate would likely have to be fifty percent or more before costs became adverse.

TABLE 10-8: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH. TWO MAN CREW

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$326.25	$1,509.73
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$271.88	$1,388.62
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$163.13	$794.81
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$217.50	$949.68
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$163.13	$514.68
VO-4B	8	$33.15	$190.40	$102.90	$99.29	$147.48	$210.11	$380.63	$1,163.89
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$435.00	$1,358.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$217.50	$763.03
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$543.75	$2,529.68
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$163.13	$678.92
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$435.00	$1,749.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$108.75	$502.54
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$380.63	$1,416.78
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$163.13	$600.49
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$135.94	$586.98
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$326.25	$1,765.97
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$271.88	$1,230.59
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$326.25	$1,858.94
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$5,029.69	$21,363.30
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	279.43	1186.85

TABLE 10-9: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH-THREE MAN CREW-TWENTY FIVE PERCENT TIME INCREASE

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$570.00	$1,753.48
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$475.00	$1,591.74
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$285.00	$916.68
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$380.00	$1,112.18
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$285.00	$636.56
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$665.00	$1,448.27
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$760.00	$1,683.02
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$380.00	$925.53
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$950.00	$2,935.93
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$285.00	$800.80
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$760.00	$2,074.98
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$190.00	$583.79
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$665.00	$1,701.15
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$285.00	$722.37
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$237.50	$688.54
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$570.00	$2,009.72
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$475.00	$1,433.72
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$570.00	$2,102.69
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$8,787.50	$25,121.11
Average	11.00	70.22	219.02	128.15	108.41	171.52	210.11	488.19	1395.62

TABLE 10-10: PROJECTED COSTS FOR PRODUCTION BLASTING OF VERTICAL OPENINGS AT BEULAH-TWO MAN CREW-TWENTY FIVE PERCENT TIME INCREASE

Vertical Opening	Number of Holes	Exploration Costs	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
VO-1	10	$86.70	$310.00	$120.05	$242.55	$214.07	$210.11	$407.81	$1,591.29
VO-2	7	$354.45	$185.94	$42.86	$209.48	$113.90	$210.11	$339.84	$1,456.58
VO-3	5	$96.90	$125.38	$25.73	$99.23	$74.33	$210.11	$203.91	$835.59
VO-4A	8	$38.25	$137.70	$42.88	$176.40	$126.84	$210.11	$271.88	$1,004.06
VO-4A#2	3	$0.00	$51.00	$42.88	$0.00	$47.57	$210.11	$203.91	$555.46
VO-4B	8	$33.15	$190.40	$102.90	$99.23	$147.48	$210.11	$475.78	$1,259.05
VO-5	13	$68.85	$223.55	$102.90	$132.30	$185.31	$210.11	$543.75	$1,466.77
VO-6	6	$93.50	$95.20	$68.60	$0.00	$78.12	$210.11	$271.88	$817.40
VO-7&8-N	28	$66.30	$605.20	$377.30	$242.55	$484.47	$210.11	$679.69	$2,665.62
VO-7&8N#2	9	$0.00	$114.75	$102.90	$0.00	$88.04	$210.11	$203.91	$719.70
VO-7&8-S	20	$60.35	$425.00	$282.98	$0.00	$336.54	$210.11	$543.75	$1,858.73
VO-7S#2	4	$0.00	$68.00	$51.45	$0.00	$64.23	$210.11	$135.94	$529.72
VO-9	10	$110.50	$188.70	$68.60	$297.68	$160.56	$210.11	$475.78	$1,511.93
VO-9#2	6	$0.00	$87.55	$68.60	$0.00	$71.11	$210.11	$203.91	$641.27
VO-10	5	$73.10	$69.70	$42.88	$0.00	$55.25	$210.11	$169.92	$620.96
VO-11	22	$55.25	$423.30	$334.43	$77.18	$339.45	$210.11	$407.81	$1,847.53
VO-12	14	$89.25	$253.30	$214.37	$0.00	$191.69	$210.11	$339.84	$1,298.56
VO-13	21	$37.40	$387.60	$214.37	$374.85	$308.36	$210.11	$407.81	$1,940.50
TOTALS	199	$1,263.95	$3,942.27	$2,306.68	$1,951.45	$3,087.28	$3,781.98	$6,287.11	$22,620.72
Average	11.00	$70.22	$219.02	$128.15	$108.41	$171.52	$210.11	$349.28	$1,256.71

10.2.4 Comparison With Other Methods

The most direct comparison to other means of mitigating vertical openings is with direct fill. Direct fill is a method whereby material from a borrow pit is hauled to and dumped in the open voids. This method is subject to significant cost sensitivities including the haulage distance, the availability of material and the size of the vertical openings.

The direct fill method has been used in North Dakota to reclaim sinkholes in the past. Generally, the conditions of use have been a ready supply of overburden from surface mine spoil piles at no cost, a short haul distance and vertical openings similar to those studied in this research. The cost of filling an individual sinkhole has been estimated to be less than $1,000 per hole.

The cost per hole would quickly rise if fill had to be purchased. Also a long haul for fill would rapidly increase costs. On the other hand blasting costs would not increase as rapidly for remote sites. The primary cost increase would be travel expense and some mobilization cost for drilling. These costs could increase the average cost per sinkhole by $150 for a similar number of blasts as performed at Beulah.

When blasting is used more disturbance around the vertical opening will occur. Therefore, the regrading cost after the void is filled will be greater than for direct fill, if such further reclamation is carried out. In the worse case the cost by blasting could be 50 to 100 percent greater than for direct fill. However, when direct fill is not performed under the most favorable terms, the two methods will quickly become equivalent. In addition, the absence of free borrow material, or a long haul distance could preclude the use of the direct fill method. Blasting, on the other hand, would be precluded more for technical reasons. The most important of these would be openings less than 500 feet from residences or residences more than 500 feet away that are undermined.

10.3 COST ANALYSIS OF BLASTING IRREGULAR MINE WORKINGS

10.3.1 Cost Data for Production Blasting of Irregular Workings

The initial compilation of cost data for the White site has been performed on a per blast basis. The data has been divided into two groups; one for the blasts over individual rooms (B1 to B5) and one for the area pattern blasts (B6 to B8). As before these cost estimates are for production based work and do not include research related tasks required for the current project.

In this case the exploration costs are not computed on a per blast basis. It was found difficult to allocate these to individual blasts in a meaningful way, because exploration studies outside a given blast area often had an impact on the successful establishment of the location of rooms within the blast area. Therefore, the exploration cost is computed separately and then combined with the production costs.

If the entire site were reclaimed by blasting individual rooms then the exploration cost would be that given in table 3-2 in Chapter 3. This amounted to $21,395 for the total area. Even though part of the area was pattern blasted, exploration was thorough throughout so that the mine workings could be properly delineated for research purposes. Therefore, the cost estimate reflects the kind of exploration effort required for shooting individual rooms, where exact knowledge of the room size and trend is necessary.

Pattern blasting would considerably reduce the exploration cost. Some T.V. camera work might be needed to help orient the effort, but would not exceed 3 days work. The engineering time is estimated to reduce by one half. The eight-inch diameter drilling is estimated at 150 feet and the 4- or 6-inch diameter exploration drilling is estimated at 3000 feet. The cost then totals $9,790,000 or less than one-half that required for blasting individual rooms.

Another matter that had to be addressed was how to apportion the accessory and supply cost. In this case it was found that dividing the cost equally among the shots gave unrealistically high cost for the individual room blasts and too low a cost to the pattern blasts, each of which affected a considerably larger area. After several theories were tested it was found best to divide this cost up based on the ratio of the square feet of mined area in the blast to the total square feet mined. This best related the cost to the effort required for each blast. Since the pattern blasts were much larger these absorbed a higher portion of the cost. The square footage was found by planimeter from the mine maps that were prepared. The map was provided earlier as figure 5-1 (page 127).

Table 10-11 provides the estimated production blasting costs for the individual room blasts. Table 10-12 gives the same data for the area pattern blasts.

The drilling cost is based on a $1.00 per foot cost and the total footage required for each shot. Each blasthole was taped and the depth recorded.

The explosive consumption was recorded for each blast by type. The weights were quite accurate, the only error possibility being the estimate of partial bags of powder remaining after a blast.

TABLE 10-11: PROJECTED COSTS FOR PRODUCTION BLASTING OVER MINED ROOMS AT THE WHITE TEST SITE.

Blast Number	Number of Holes	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
B1	12	$243.00	$117.49	$110.46	$151.40	$244.64	$456.00	$1,322.99
B2	11	$289.00	$99.41	$254.05	$181.25	$276.00	$532.00	$1,631.71
B3	13	$304.00	$81.34	$165.69	$200.14	$299.30	$632.00	$1,682.47
B4	5	$122.00	$0.00	$88.38	$52.83	$155.92	$304.00	$723.13
B5	3	$98.00	$18.08	$77.32	$51.11	$97.08	$228.00	$569.59
TOTALS	44	$1,056.00	$316.32	$695.90	$636.72	$1,072.94	$2,152.00	$5,929.88
Average	8.80	$211.20	$63.26	$139.18	$127.34	$214.59	$430.40	$1,185.98

TABLE 10-12: PROJECTED COSTS FOR AREA PATTERN BLASTING AT THE WHITE TEST SITE.

Blast Number	Number of Holes	Drilling Cost	ANFO Costs	HANFO Costs	Primers, Det. cord Delays	Accessory & Supply Cost	Labor Cost	Total Costs
B6	46	$1,425.00	$262.09	$751.12	$741.91	$1,104.29	$945.00	$5,229.41
B7	20	$744.00	$171.71	$320.33	$413.34	$480.61	$532.00	$2,661.99
B8	65	$2,347.00	$894.71	$1,060.42	$1,470.20	$2,979.16	$1,260.00	$10,011.49
TOTALS	131	$4,516.00	$1,328.51	$2,131.87	$2,625.45	$4,564.06	$2,737.00	$17,902.89
Average	43.67	$1,505.33	$442.84	$710.62	$875.15	$1,521.35	$912.33	$5,967.63

Primer and delays counts were kept for each blast. The length of detonating cord was estimated according to the hole depths and the surface pigtail lengths.

The accessory and supply cost calculation is based on the data in table 10-2. The method for apportioning the cost to individual blasts was explained above.

The labor cost is based on the data from table 10-3. The hours spent actually involved in production blasting were estimated and the labor cost applied to these numbers.

The total cost for each blast in tables 10-11 and 10-12 are found on the right-hand side of the table. These are production costs only. At this point the exploration costs are not included.

10.3.2 <u>Analysis of the Cost Data For The White Site</u>

Figures 10-2 and 10-3 show the cost breakdown for blasting above the rooms and pattern blasting respectively. There are differences between the two. When blasting above the rooms the labor cost is the greatest single cost as the process is more time consuming. Other costs are roughly similar when the explosive percentages are combined. This cost breakdown looks similar to that in figure 10-1 for the vertical openings.

When pattern blasting is employed the labor cost percentage is reduced by half. The drilling cost rises substantially and is equal to the accessory and supply cost. The increase in drilling cost is partly due to the depth to the workings increasing toward the east. Also, in pattern blasts up to half the holes were in pillars. These holes were typically drilled deeper so the total drilling footage tended to increase.

The total cost for each blast was determined and was listed in the tables in the previous section. The square feet of mined area has also been determined for each shot, from the maps. It is therefore, possible to determine a cost per square foot of mined area for each shot. An average is then determined.

This data is presented in table 10-13. The top part of the table is for the individual room blasts and the bottom portion is for the area pattern blasts. The table shows that using the pattern blasts is a more economical approach in the case of irregular workings. Blast B8, which was a large pattern blast on a 15x15 foot pattern was especially economical relative to the individual room method.

Taking the total cost and the total square footage mined in each case one finds a cost of \$1.65 per square foot for the individual room blasts and \$1.17 per square foot for the area pattern blasts. The former, applied to the total area of 18,882

FIGURE 10-2

ESTIMATED PRODUCTION BLASTING COSTS

White Site–Above Rooms

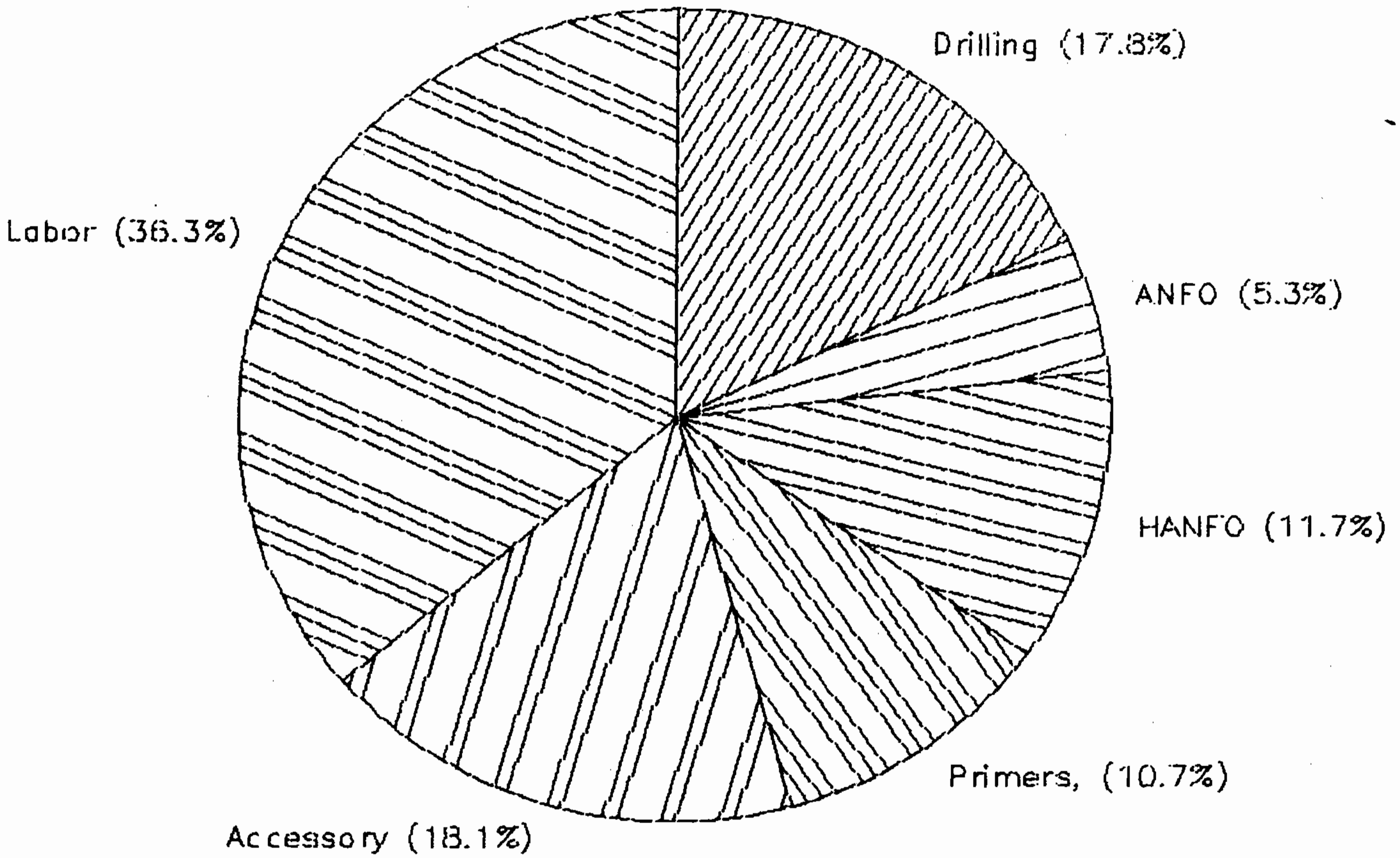


FIGURE 10-3

ESTIMATED PRODUCTION BLASTING COSTS

White Site—Pattern Blasts

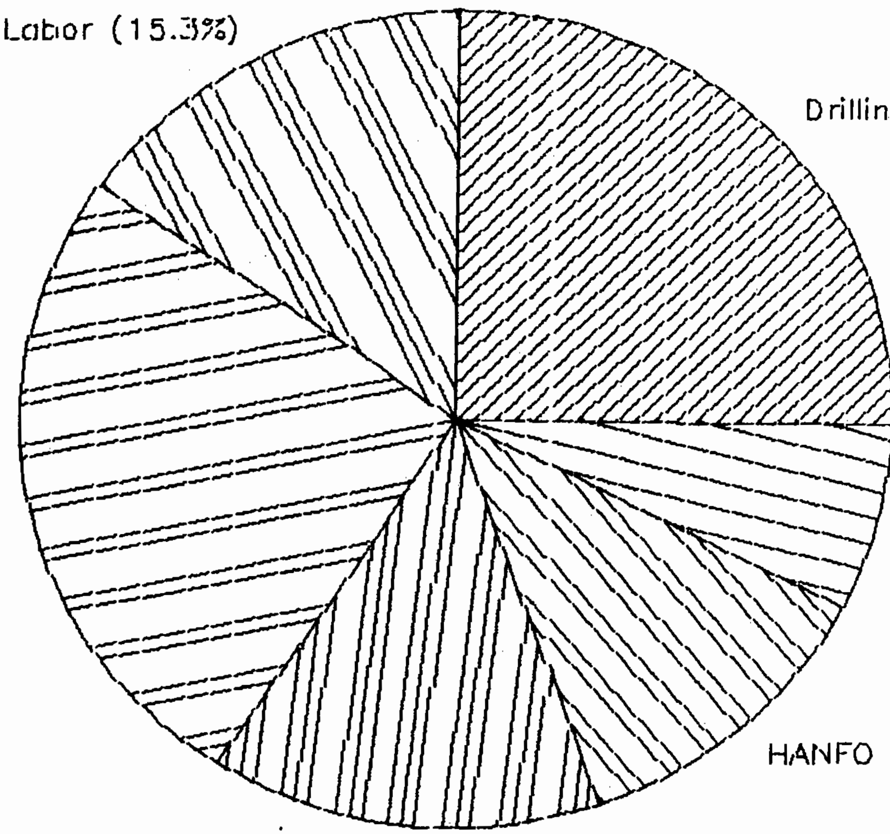


TABLE 10-13: UNIT COST OF RECLAIMING MINE WORKINGS BY BLASTING METHODS

Blast Number	Blast Cost	Mined Area sq. ft.	Cost Per Square Foot
B1	$1,322.99	819	$1.62
B2	$1,631.71	924	$1.77
B3	$1,682.47	1002	$1.68
B4	$723.13	522	$1.39
B5	$569.59	325	$1.75
Totals	$5,929.89	3592	$1.65
Average	$1,185.98	718	
B6	$5,229.41	3697	$1.41
B7	$2,661.99	1609	$1.65
B8	$10,011.49	9984	$1.00
Totals	$17,902.89	15290	$1.17
Average	$5,967.63	5097	

square feet, yields a cost of $31,155. For the pattern blasts the resulting total cost is $22,092. This represents a significant reduction in total cost.

The projected exploration costs for each case were given above. These costs applied to the same square footage of mined area as do the production costs. Therefore the cost per square foot of mined area for exploration when individual room blasting is used is $1.13 per square foot. When a pattern blasting technique is used for the whole area the cost per square foot is $0.52.

The combined exploration and production costs for shooting individual rooms is, therefore, $2.78 per square foot. The combined cost for blasting on an area pattern basis is $1.69 per square foot. For the total 18,882 square feet of mined area this amounts to a cost of $52,492 for shooting individual rooms and $31,910 for blasting by area pattern techniques. Clearly, there are significant savings to be obtained using pattern blasting techniques.

The surface area affected by the blasting was approximately one acre. A planimeter run on the mine map gives an affected area of about 36,000 square feet which is close to one acre. Therefore, the costs quoted above can be considered as applicable on a per acre basis.

These costs make clear that the pattern blasting approach is the better method for mines such as that exemplified by the White site. This is due especially to the substantial variance in exploration costs. However, production costs are also lower on a square footage basis when a 15x15 foot pattern can be used.

10.3.3 <u>Sensitivity Analysis of Costs</u>

Cost factors such as drilling, explosives and labor can be varied as was done for the Beulah test blasts. Some variation in costs will result from such changes. However, one of the more important changes that will affect the cost is the recovery experienced, or, the amount of an acre of land that was undermined.

For the individual room method the internal recovery, that is the ratio of rooms to pillars will also offset the cost since only rooms are blasted. This aspect is less important when pattern blasting as holes are placed without regard to being in room or pillar.

Table 10-14 are estimated costs for different field recovery rates. It can be seen that, as the amount of an acre that was undermined increases, the cost per acre also rises. At the White Site 18,882 square feet were undermined for a recovery of 43 percent of the one acre site. Since field recovery is

TABLE 10-14: ESTIMATED COST FOR PATTERN BLASTING AS A FUNCTION OF FIELD RECOVERY

Field Recovery %	Square Feet Undermined	Cost For Pattern Blasts
25	10890	$18,404.10
30	13068	$22,084.92
35	15246	$25,765.74
40	17424	$29,446.56
45	19602	$33,127.38
50	21780	$36,808.20
55	23958	$40,489.02
60	26136	$44,169.84

unlikely to be less than 25 percent or more than 60 percent,costs per acre should be within the project range.

Table 10-15 shows the different field recoveries and also different internal recoveries. The internal recovery relates to the area of the internal pillars that were left to support the mine during operation. In this analysis recoveries are shown that reflect mining that left from 20 to 55 percent of the mine area as pillars. At the White location internal pillars had an area of 3,712 square feet. The internal recovery was 80 percent which is quite high. For many mines a recovery of 50 to 60 percent, on a square footage basis would be common.

Examining tables 10-14 and 10-15 one observes that the cost of reclaiming an acre is about equal for each field recovery, when the internal recovery is 60 percent. For internal recoveries less than 60 percent individual room blasting should be more economical, even taking into account the high exploration cost.

When the internal recovery is greater than 60 percent pattern blasting becomes increasingly economical. At 80 percent, which was the recovery at the White test site pattern blasting is much less costly than individual room blasting, reflecting the significant differences in exploration cost.

The conclusion then is that there will be an advantage to performing the additional exploration and drilling production blastholes only over the workings when there are few workings to be found. If the area is extensively undermined then pattern blasting, without regard for whether the blastholes are in pillar or room will be more economical than the time consuming effort to exactly locate all the rooms and pillars. This conclusion applies primarily to reclamation of irregular workings where the rooms cannot easily be located.

Another factor which will affect the cost is the depth to the workings. For this field test the average depth was about 35 feet. If the mine is considerably deeper both drilling cost and explosives costs will increase. It is difficult to estimate in advance how much cost will be added. For any given project the design should be developed as recommended in this report. If there is significant difference in drilling depth the added cost can then be developed from the design.

For mines that are at considerable depth the surface collapse and disruption that makes more shallow mines hazardous and destroys land use values is not likely to occur. The swell of collapsing roof material will tend to fill the void before it manifests itself at surface.

For example, consider a mined room having dimensions of 8 feet wide by 6 feet high. Consider also that the material swells by 18 percent as it caves into the workings. Then about 33 feet of height would be required to fill the void. The room

TABLE 10-15: ESTIMATED COSTS FOR RECLAIMING MINED ROOMS USING THE INDIVIDUAL ROOM METHOD FOR DIFFERENT FIELD AND INTERNAL RECOVERY RATES

Field Recovery %	Square Feet Undermined	Cost Per Acre for Different Internal Recoveries							
		Internal Recovery Ratio, %							
		45	50	55	60	65	70	75	80
25	10890	$13,623.39	$15,137.10	$16,650.81	$18,164.52	$19,678.23	$21,191.94	$22,705.65	$24,219.36
30	13068	$16,348.07	$18,164.52	$19,980.97	$21,797.42	$23,613.88	$25,430.33	$27,246.78	$29,063.23
35	15246	$19,072.75	$21,191.94	$23,311.13	$25,430.33	$27,549.52	$29,668.72	$31,787.91	$33,907.10
40	17424	$21,797.42	$24,219.36	$26,641.30	$29,063.23	$31,485.17	$33,907.10	$36,329.04	$38,750.98
45	19602	$24,522.10	$27,246.78	$29,971.46	$32,696.14	$35,420.81	$38,145.49	$40,870.17	$43,594.85
50	21780	$27,246.78	$30,274.20	$33,301.62	$36,329.04	$39,356.46	$42,383.88	$45,411.30	$48,438.72
55	23958	$29,971.46	$33,301.62	$36,631.78	$39,961.94	$43,292.11	$46,622.27	$49,952.43	$53,282.59
60	26136	$32,696.14	$36,329.04	$39,961.94	$43,594.85	$47,227.75	$50,860.66	$54,493.56	$58,126.46

dimensions in this example are typical of older mines, and would be somewhat high where thin seams were mined. It would appear then that overburden heights, where reclamation of this sort is required, would usually not be much over 50 feet. This height of overburden would fill rooms of 8x9 feet or equivalent. However, for those fewer mines with large rooms it can take deep cover to close the void before surface expression is seen. For example rooms 20 feet wide by 10 feet high require caving of 139 feet of material to fill the void when an 18 percent swell is assumed. This is why such large sinkholes can appear at Beulah even in deeper cover.

Since most mines to be reclaimed are likely to have openings of 8x9 feet or less the costs from this research will apply to most cases. However, one should be aware of and take into account sensitivity to the depth to the workings.

10.3.4 Comparison With Other Methods

Prior estimates have been made by the State of North Dakota concerning the cost of AML reclamation by other methods. In some cases these methods have been used to reclaim old works locally. The costs for methods of complete reclamation of AML sites are given in table 10-16.

TABLE 10-16: PROJECTED COSTS FOR VARIOUS RECLAMATION METHODS FOR UNDERGROUND MINES

Method	Per Acre Costs
Daylighting - without coal removal	
(a) Shallow Mine	\$10,000 - \$20,000
(b) Deep Mine	\$100,000 - \$200,000
Remote Backfill	\$59,000 - \$100,000
Dynamic Consolidation (Shallow Mines)	\$10,000 - \$70,000

Examining these methods shows that blasting costs are within the same range and the method is acceptable on a cost basis. Blasting appears to be considerably cheaper than remote backfill, another frequently used method. Daylighting shallow mines may be less costly, depending on the extent of undermining involved but costs accelerate rapidly as the overburden depth increases. Dynamic consolidation has similar costs to blasting.

As tables 10-14 and 10-15 show, blasting may be especially attractive on a cost basis when recoveries are low. Cost attractiveness is further enhanced when, as at the White site it is possible to blast with minimal surface disruption and therefore minimum post-blasting regrading requirement.

CHAPTER 11: CONCLUSIONS

As a result of this research, in which the filling of individual vertical openings and the closure of erratic, undocumented underground workings has been studied, the following conclusions have been developed.

11.1 GENERAL CONCLUSIONS

The following general conclusions, concerning the project as a whole have been made:

1. Blasting can be used to successfully close individual vertical openings. A method employing cratering charges in decked blastholes works well. The cratering charges are designed to displace to the void. Adequate swell is generated in the material to fill the sinkhole. Examination of the site a year after blasting did not indicate much settlement of the blasted material.

2. Blasting can be employed to close irregular mine workings for which there is little documentation in the form of mine maps or surveys. Two methods are available to blast in these workings. One is to blast the individual rooms with charges placed directly over the opening, designed to crater down into the mined room. The second approach is to area pattern blast without regard for whether the blastholes are drilled in room or pillar. Charges are designed to crater down into the workings or to the side depending on whether the hole is over the room or drilled into the pillar.

3. For irregular workings with few pillars the pattern blasting approach has been indicated to be most effective. For mines with a low recovery and large or many internal pillars shooting in the individual rooms is likely best.

4. Blasting individual sinkholes is somewhat more costly than direct fill methods when optimum conditions are available for direct fill. However, as haul distances increase for the fill, or a source of free or very inexpensive fill is not available, blasting soon becomes as or more economical than direct filling with borrowed material.

5. Blasting closed irregular workings is cost competitive with other methods. It appears to be significantly less costly than remote fill (hydraulic or pneumatic), an often used method. It is competitive with daylighting and with dynamic consolidation techniques.

6. The current work has been in overburden depths of a maximum of forty feet. However, using four explosive decks and a similar loading arrangement blasting could be conducted in fifty to fifty-five feet of cover. It is considered that four

decks is about the maximum that can easily be placed and timed.

7. For deeper cover it can still be possible to blast the workings closed without requiring more than four explosive decks. Consider that, for eighteen percent swell a 10x6 foot void can be filled with thirty-three feet of blasted overburden. Similarly an 10x10 foot void could be filled by fifty-five feet of overburden. Therefore, in seventy-five feet of cover only the bottom thirty-five to fifty-five feet of cover would need to contain explosives, in these examples, to adequately fill the void. It would appear that quite deep workings could be shot by determining how much material actually needs to be broken to fill the void. An associated advantage with blasting deep works would be that the project could be completed with virtually no surface disturbance. This method will not likely be successful when the rooms are very large.

8. Blasting closed vertical shafts would require the same principles as blasting vertical openings. The blasts should be designed to cast material into the void. The scaled depth of burial and radius requirements would be in the same range as for the vertical openings. For deeper shafts in more remote areas longer linear charges could be used designed with square root scaling. In this case the scaled burdens should be in the range of 2.0 to 2.5 $ft/lb^{1/2}$ and the scaled radius should be about 2.0 $ft/lb^{1/2}$. More than one row of holes may be required. This can be calculated by calculating the volume of void to fill and comparing this to the volume of rock. A main concern with shafts will be the effect of any shaft support systems in place and shaft hardware if any. The effect of these items should be given careful consideration. It is concluded that much of the data obtained in this study for vertical openings may also apply to shafts.

11.2 CONCLUSIONS CONCERNING EXPLORATION NEEDS

The amount of exploration required varies depending on the type of blasting and the type of site. The following conclusions have been arrived at from the current research:

1. The least amount of exploration is required when vertical openings are being blasted closed. The nature of these features provides much visual information not available for other types of sites.

2. When blasting sinkholes exploration drilling costs averaged $79 per opening. The cost varied from $30.60 to $354.45 for an opening. Some cost is also required for engineering time to lay out holes and interpret results. This cost would range from $40 to $80 per sinkhole.

3. Exploration needs are much greater when the need is to collapse irregular, ill defined underground workings. The method whereby the rooms are collapsed using blastholes located

directly over the workings is most costly because the location of the rooms must be known exactly. For the White Site, having an extent of one acre, the exploration cost, under these circumstances was approximately $22,000.

4. When the method of area blasting with a regular pattern is used the exploration cost is estimated to reduce considerably. The workings do not need to be defined so exactly in this case. For the White Site exploration costs of $9,790 are estimated for the one acre extent.

5. The use of the down hole T.V. camera system developed by L.C. Hanson, Inc. was very helpful in aiding the attempt to locate the rooms and reduced the number of drill holes needed to establish the hole locations. The camera does not work, however, where the workings are water filled.

6. Borehole T.V. camera work was useful for assessing the results of blasting. The instrument could be used to look back at blasted areas to examine the closure obtained. Both the Hanson device and a more sophisticated commercial instrument from the State of North Dakota were used at the White site.

11.3 FIELD PROCEDURES AND TECHNIQUES

A variety of field procedures were studied. The conclusions developed are listed below.

11.3.1 <u>Vertical Openings</u>

1. The blasting of vertical openings does not result in the unwanted collapse of immediately adjacent rooms. In shooting fourteen openings using eighteen blasts there were no cases, where the detonation caused nearby collapse. A key to successfully avoiding unwanted disruption is the use of a method which keeps explosive charges of low weight and provides for independent detonation of successive decks, thereby minimizing vibrational effects.

2. Examination of the test area a year after blasting shows two incidents where a new sinkhole had opened up near one that was blasted. It is difficult to say how much the blasting affected the nearby area. However, there is some indication that blasting may affect nearby, unstable areas some time after blasting.

3. In general the vertical openings fill well upon blasting. Smaller, circular sinkholes provide the best result. Longer openings with an open ramp into the workings may require two blasts to fill. The biggest problem is the sloping ramp which, in some cases, ratholes. It has also been found that using two rows of holes can provide success with only one shot required.

4. Undercut vertical openings, with a small opening at surface and a large void at the base can also require two shots to fill. It is difficult to position the drill as one would wish due to safety needs. Two openings of this type (VO-9 and VO-12) were blasted. These gave the least satisfactory overall results of the vertical openings tested.

5. The decked loading system was quite successful for vertical openings. Good movement into the opening was obtained. Powder loads were controlled at a reasonable weight. The explosive decks successfully cratered toward the central opening.

6. Where a sinkhole had a sloping ramp, open to the room below, it was necessary to drill holes along both sides of the ramp. The blastholes were increased in depth as the ramp extended deeper. Different numbers of decks had to be used as the hole depths changed. This method worked well for sloping ramps.

7. It was important to fill out the blast summary sheets while the blast was being drilled and prepared for loading. Also, taking time to prepare the explosive loading charts prior to loading the blast was essential to good success. Decked loading of holes requires careful placement of explosives. Also the hole depth and location of any void below is important to designing the hole loads. Therefore, any project of this type should use recording procedures like those described in the report for best results.

8. Drill cuttings provided adequate service for deck stemming and for stemming the top of the hole. Since drill cuttings are the only stemming material likely to be present on site the use of this material is dictated by availability and costs. Problems should not arise unless a site is very wet, when drill cuttings may provide poor stemming material.

9. For the vertical openings the use of 42 ms delays on surface works well. Nonel noiseless trunkline delays were used. These were easy to connect up and helped control noise generated by blasting.

10. Down-the-hole delays of periods 4 to 8 (100 to 200 ms) were used with good success. The delays were Nonel HD Primadets used with one pound slider primers. Generally, every other period was used giving a 50 ms delay between decks. In some cases 25 ms delays were used between upper decks. Either approach worked. The use of 50 ms between decks is recommended as usually giving the best result.

11. There was no problem manoeuvering equipment around the site. Care did have to be taken, when positioning the drill on a blasthole, to insure the equipment was safely located. Undercut sinkholes were the most difficult features in terms of being able to position the blastholes where needed and still be able to position the drill safely. When necessary holes were placed in less than optimum locations to insure safe operation around undercut vertical openings.

12. The use of a combination of ANFO and a waterproof heavy ANFO (70/30) proved quite satisfactory. The heavy ANFO was useful in the bottom deck for protection against any water accumulation and for added energy. Only ANFO was needed in upper decks.

13. Second blasts on a feature were loaded with all ANFO. So were four primary blasts. Results were quite satisfactory. Therefore, the use of all ANFO in most cases is acceptable and will reduce costs. However, in blastholes that do not intersect a void the use of HANFO is recommended to guard against water attack.

11.3.2 Collapsing Irregular Underground Mine Workings

1. These abandoned mine features can be reclaimed using either a method that places each blasthole directly over the workings to be collapsed or by placing holes on a regular pattern, such that some holes intersect the void while others are drilled in pillars.

2. Irregular workings at the White Test Site tended to have large intersects and small, variable pillars. The result was that upon firing the shot there tended to be considerable surface disruption and collapse beyond the blast area. This made it difficult to properly position blastholes for the next shot, adjacent to the blasted area. The potential then exists for unwanted voids to remain.

3. The blasted area profile could be controlled by using more stemming to limit surface displacement. At the same time full collapse of the workings was achieved. It was found that twelve to sixteen feet of stemming could be used successfully. When twenty-five feet of stemming was used, in one case, the void was not filled but moved closer to the surface. In general, for a 6-inch diameter blasthole, not more than sixteen feet of stemming should be used.

4. It was also helpful to attempt to finish the blast at a point where there were pillars and/or the intersections were narrow. The mine would be more stable at such locations and less likely to generate much unwanted collapse beyond the blast.

5. The deck loading system worked well in these cases. For holes over the void the successive decks cratered down into the voids well. For holes in pillars the shot cratered off the

side, toward the nearby void. Cratering action would be more exact for holes directly over the rooms but designing for cratering charges worked well in either case.

6. The use of seismic plugs to seal the bottom of the hole before loading was successful. There were no incidents of loaded blastholes slumping due to failure of the hole plug. Tying the plug off at surface using baler twine is recommended to insure explosive and primers are not lost into the room below due to plug failure. Loading poles are required to force the plug down to the bottom of the hole, so these should be available.

7. When using the area pattern blasting technique the attempt should be made to place as many of the holes in voids as possible. The more holes penetrating voids the better the performance. At the White location about fifty percent of the blastholes typically struck void.

8. Throughout most of the White test site the surface alteration from blasting was modest. Typically the change ranged from three feet of swell to three feet of surface depression. Swell was more common than slumping. This reflected the small voids usually found below the shot. The small change in topography meant that subsequent regrading needs would be small.

9. The water contained in the workings was not a major problem. The primary result was to increase time requirements for loading the holes with explosives. Upon shooting the blast there was some venting of water through open boreholes, but there was no seepage of water from the workings.

10. The presence of water made the use of heavy ANFO necessary in the bottom decks of holes that contained water. This site was representative of explosive selection requirements where the workings contain water.

11. The method of placing a hole plug above the water, followed by one to two feet of stemming and then placing the first ANFO deck worked well to guard the ANFO against water attack. An alternative, which would speed loading but also increase explosive costs, would be to use heavy ANFO in the second deck as well. Doing so would help to meet design explosive placement criteria.

12. The first blast at the White site closed that adit and main entry into the workings. The adit was thoroughly closed by drilling holes on the slopes around it and throwing material into the trench in front of the adit. It was further closed by holes which collapsed the roof of the adit as it extended into the workings. It is concluded that adits, driven into the side of a hill can be successfully closed by this technique.

11.4 HIGH-SPEED CAMERA STUDIES

A high-speed camera was used to film most of the blasts at both sites. The following conclusions have been reached.

1. The high-speed camera is a good tool for examining the results of AML work. It is quite important to use the camera for research and development work, but not as important for production projects. It would be wise to film a few initial blasts to assist in optimizing the approach. Video taping the shots with standard video camera is recommended as a tool to assist in assessing the blasts, that will provide good information for production work at considerably less cost.

2. The surface delays at Beulah had a mean firing time of 42.1 ms. At the White site the mean time was 35.4 ms. These delays had a nominal firing time of 42 ms. The standard deviations indicated significant variation in firing times.

3. The computation in surface delay times led to the conclusion that some delays between successively firing decks would be less than 8 ms and overlaps in firing could occur. However, this problem was not major.

11.5 VIBRATION CONTROL

Each blast was monitored with a seismograph. Both ground vibration and airblast were monitored. The following conclusions were reached.

1. At the Beulah site vibration levels were low. The particle velocity versus scaled distance plot shows that the particle velocity at the upper limit line would be 0.1 ins/second for a scaled distance of 72 $ft/lb^{1/2}$. Particle velocity of this level would have no damage potential and would eliminate citizen concern about blasting.

2. At the White test site a scaled distance of 105 $ft/lb^{1/2}$ gives a peak particle velocity of 0.1 inches per second. A scaled distance of 70 $ft/lb^{1/2}$ yields a particle velocity, at the upper limit line, of 0.2 ins/sec. Therefore, vibration could again be controlled to eliminate both damage and most human response.

3. Based on the results obtained it is recommended that, for AML blasting, the scaled distance to a house be maintained at 70 $ft/lb^{1/2}$ or greater. Where possible increasing the scaled distance to 100 $ft/lb^{1/2}$ will be even better and should eliminate virtually all complaints.

4. No complaints about blasting were received at either site during this research. This indicates that with the methods used human response was not a problem.

5. It has been concluded that, if the houses are not undermined blasting can be used to within 500 feet of housing safely and without damage. This conclusion is further consistent with the fact that much close-in blasting is performed throughout the country for construction purposes.

6. Control of vibration is also important, when shooting vertical openings, for avoiding unwanted collapse of adjacent workings. The deck loaded, independently delayed system used helped control close-in vibration and avoided unwanted failures.

7. Airblast can also be controlled to quite acceptable levels in AML blasting.

8. A cube root scaled distance of not less than 100 $ft/lb^{1/3}$ is generally recommended for the explosive charges to avoid airblast.

9. The top explosive deck should be adequately buried to avoid bursting of gases through the top and excessive airblast. The depth of burial of the upper deck, to the top surface, should not be less than 3.0 $ft/lb^{1/3}$.

10. All surface delay elements and detonating cord pigtails should be buried with cuttings to reduce noise. The use of Nonel, or a similar product, on surface will also greatly reduce noise from detonation of surface accessories.

11. Airblast can be controlled to the extent that blasting within 500 feet of houses can be performed without adverse affects.

11.6 CONCLUSIONS FOR BLAST DESIGN

The blasting data gathered at the sites has been analyzed. The following conclusions have been made concerning blast design.

1. For vertical openings the scaled depth of burial to void should be in the range of 2.0 to 2.6 $ft/lb^{1/3}$. This is consistent with what was expected. Generally the SDOB should be less in the bottom deck and higher in the upper decks.

2. The scaled radius should be in the range of 1.75 to 1.91 $ft/lb^{1/3}$. This represents a spacing consistent with the observed fact that, for cratering, a spacing between holes of 1.5 times the depth of burial often works well.

3. The scaled depth of burial to the top surface, for the upper deck, should be greater than the scaled depth of burial to the void for sinkhole blasting. This insures movement toward the void. At Beulah the SDOB to the top surface was 3.1 to 3.2 $ft/lb^{1/3}$ which was acceptable.

4. Given these cratering results a pattern around vertical openings that has a 10 foot burden to the void and a 12 foot spacing between holes should perform well. This pattern is for a 6-inch hole diameter. Other hole diameters will give different patterns which can be determined using the relationships above.

5. Because vertical openings are irregular features and because adequate safety must be ensured holes cannot always be placed on the desired pattern. The goal would be to place the holes as close to design as possible. Also, placing the decks in the hole must account for the hole length. Therefore, charges may have to be placed at locations other than optimum. However, the charges should be located as near to the preferred location as possible.

6. Using these techniques and relationships it is possible to fill vertical openings to near the original topography. Also the majority of topsoil can be held near the top of the blasted fill to assist revegetation.

7. For blasting in irregular openings at the White site a scaled depth of burial of 1.65 $ft/lb^{1/3}$ performed appropriately. For the upper decks a SDOB of 2.0 $ft/lb^{1/3}$ was effective.

8. When using the approach of blasting individual rooms a scaled radius in the range of 1.50 to 1.60 $ft/lb^{1/3}$ was acceptable.

9. When shooting area pattern blasts scaled radii in the range of 1.92 to 2.40 $ft/lb^{1/3}$ worked out quite well. The largest scaled radius were obtained in the 15x15 foot patterns.

10. Greater stemming lengths reduce the surface disruption and cracking. A scaled depth of burial of 5.00 $ft/lb^{1/3}$ will be adequate to avoid most disruption without leaving voids close below surface. In a 6-inch hole this corresponds to 14 to 16 feet of stemming when ANFO is used.

11. Surface disruption control through added stemming heights works best when the workings are small relative to the depth of overburden above.

12. When blasting irregular, poorly documented underground rooms the area pattern method will often be the best approach. Exploration time and cost is reduced. Production costs may also be reduced.

11.7 COSTS FOR PERFORMING AML BLASTING ON A PRODUCTION BASIS

The cost of performing AML blasting has been studied. The costs developed are for performing the work on a production basis. It is assumed that research and development has been adequate to allow the work to be performed as a standard procedure. The following conclusions have been reached.

1. On average the cost to close a vertical opening is $1,668.83. This includes the need to perform a second shot on a sinkhole in some cases.

2. Costs vary for vertical openings as drilling, labor or explosive selection requirements vary. However, for reasonable variations in these cost components, the change in cost is manageable.

3. The cost of blasting vertical openings is similar to the cost of direct filling these features. The success of direct fill is dependent on a source of fill which is close by. Blasting is not as affected by such site factors. Blasting is competitive, then, with the most common other method.

4. When blasting irregular workings, to collapse the rooms, the cost of shooting individual rooms with holes directly over the workings is estimated to be $2.78 per square foot of mined void.

5. When area pattern blasting is used to collapse these workings then the cost is $1.69 per square foot of mined void.

6. The cost is sensitive to mining recoveries. For pattern blasting the projected cost per acre is $18,404 when 25% of the area was undermined. If 60% were undermined the cost is $44,170.

7. For individual rooms blasting the least cost occurs when the field recovery is 25% and the internal recovery (accounting for pillars) is 45% and is $13,623. When the field recovery is 60% and the internal recovery is 80% the cost is $58,126 which is equal to the low end cost quoted for the remote fill method.

8. At the White test site the field recovery was 43% on the one acre site. The internal recovery was 80%. The cost to reclaim by pattern blasting is projected to be $31,910. The cost to reclaim by individual room blasting is estimated to be $52,492. Therefore, the pattern blasting method would be more attractive for a site such as this.

9. For any field recovery the cost of either method appears about equal for a 60% internal recovery. For greater than 60% internal recovery pattern blasting should be better. When less than 60% individual room blasting is increasingly more economic.

10. The cost is competitive with other methods. It can be substantially lower than remote backfill and competitive with daylighting and dynamic consolidation. Blasting is probably the method with the broadest scope and is usable where one or more of the other methods is not suitable.

11.8 PRE- AND POST-BLASTING REQUIREMENTS

Depending on the AML site being reclaimed there may be requirements for topsoil and grading before and after blasting. In this regard the following conclusions have been reached, based on observations made during the current research.

1. For both the blasting of vertical openings and the collapse of underground workings surface disruption in terms of surface swell or depression can be kept to a minimum. This excludes cases where large rooms are present under shallow overburden. In those cases surface collapse can be substantial.

2. When vertical openings are blasted there tend to be significant open cracks left in the soil around the opening. These result from venting of explosion gases through the soil and from doming of the surface during the blast which causes the formation of tension cracks in the soil. Since doming or surface swell can radiate out a significant distance from the blasthole cracking can occur ten or more feet from the perimeter of the blast.

3. The presence of these cracks can be reduced by using more stemming, however, this has to be weighed against the need to displace material into the sinkhole.

4. If the vertical opening is in a field which is cultivated then regrading after blasting is likely to be necessary. Also, it will be appropriate to remove topsoil from around the opening before blasting. This can then be replaced after to enhance the growth potential. Such topsoil removal and subsequent regrading could cost $300 to $600 per sinkhole.

5. For vertical openings in other settings such work may not be necessary. Wildlife habitat for example can benefit from some variations in topography. However, if the area is open to public access at least sufficient regrading should be done to remove the cracks formed around the sinkhole during the blast.

6. The White Site, which involved the successful total collapse of the workings, showed that such blasting can be

performed with minimal surface disruption. This site, which was in a cultivated field, could easily be regraded to the original contour. Topsoil had been saved and could then be respread. For this type of site some regrading should be anticipated if cropland is involved.

7. Experimentation at the White Site showed that the workings could be collapsed with very little if any surface disturbance. This was achieved by increasing the stemming heights. The approach was successful because the workings area below was small in relation to the overburden depth. Thus the void could be filled without necessarily breaking overburden all the way to surface. For sites of this sort it can be concluded that much of the area can be blasted with very little surface disruption. The only exception would be areas of very large intersections, where more volume is needed to fill the void.

8. The conclusion is then that for sites like the White test area the work could be performed with a minimum of pre- and post-blasting reclamation.

9. For a site such as the White location it is estimated that removing topsoil, regrading and replacing topsoil can be achieved for approximately $2,500 per acre.

11.9 RECOMMENDATIONS

Based on the research completed the following recommendations are made:

1. Blasting should be considered as a viable method for reclaiming abandoned mine lands. When selecting a procedure for reclaiming these sites blasting ought to be examined along with other methods and a selection made based on which approach best suits the needs of the particular project. Blasting appears to be one of the most broadly applicable methods for reclaiming features from vertical openings to complete collapse of workings to the closure of shafts and adits.

2. The research has been conducted in overburden typical of the Northern Plains consisting of overconsolidated clays with interspersed, intermittent layers of weak sandstones and claystones. It is recommended that sites be attempted with considerably different strata to further establish crater relationships and operating procedures.

3. The T.V. camera system was very useful in dry workings for assisting in the location of the rooms and for viewing the result of the blast. The system used was not commercially available but was inexpensive and easily employed. It is recommended that more work be done with such systems to improve lighting and viewing distance, raising and lowering and monitoring. The goal should be to provide a good system for

under $5,000. Such a system would have high value in AML work beyond just the uses in blasting.

4. Extensive research and development has now been completed using blasting to reclaim old works. It is recommended that an engineering manual and training program be prepared that would benefit state agencies and contractors involved in AML work.

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APPENDIX A

BLAST SUMMARY SHEETS AND TIE-IN DIAGRAMS FOR THE BEULAH SITE

AML PROJECT - BLAST SUMMARY

BLAST #: VO-1

LOCATION: N-14

DATE 8/3/88

HOLE DIAM 6"

NO. HOLES 11

EXPLOSIVES CONSUMPTION

ANFO 14 BAGS = 700 LBS SLURRY 22 BAGS = 660 LBS

BOOSTERS 27 42 M/S DELAYS 11

DELAYS #4 11 #5 ____ #6 11 #7 ____ #8 5 #9 ____

SEISMOGRAPH: ____ LOCATION: n/a PPV: ____ IN/SEC

CAMERA USED: n/a TARGET DELAY #: ____ FPS: ____

det. Cord 350' Elect. E.B. Cap 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1			40	40	42		42	2		
2				40(fill)						in pillar - backfill to 40'
3			34½	34	36½		36½	2		
4		25		24	28		28	3		in Crosscut Caved further to 21' (8/3/88)
5			25	24	28		28	3		in Crosscut
6					19		19	0		in pillar - South Side
7				19(fill)	19½		19½	0		" " " "
8				19(fill)	19½		19½	0		" " " "
9				28(fill)	47		47	0		in pillar close to X-cut need backfi
10			42	40(fill)						Backfill - wet
11			42	42	46		46	4(?)		Void hard to measure

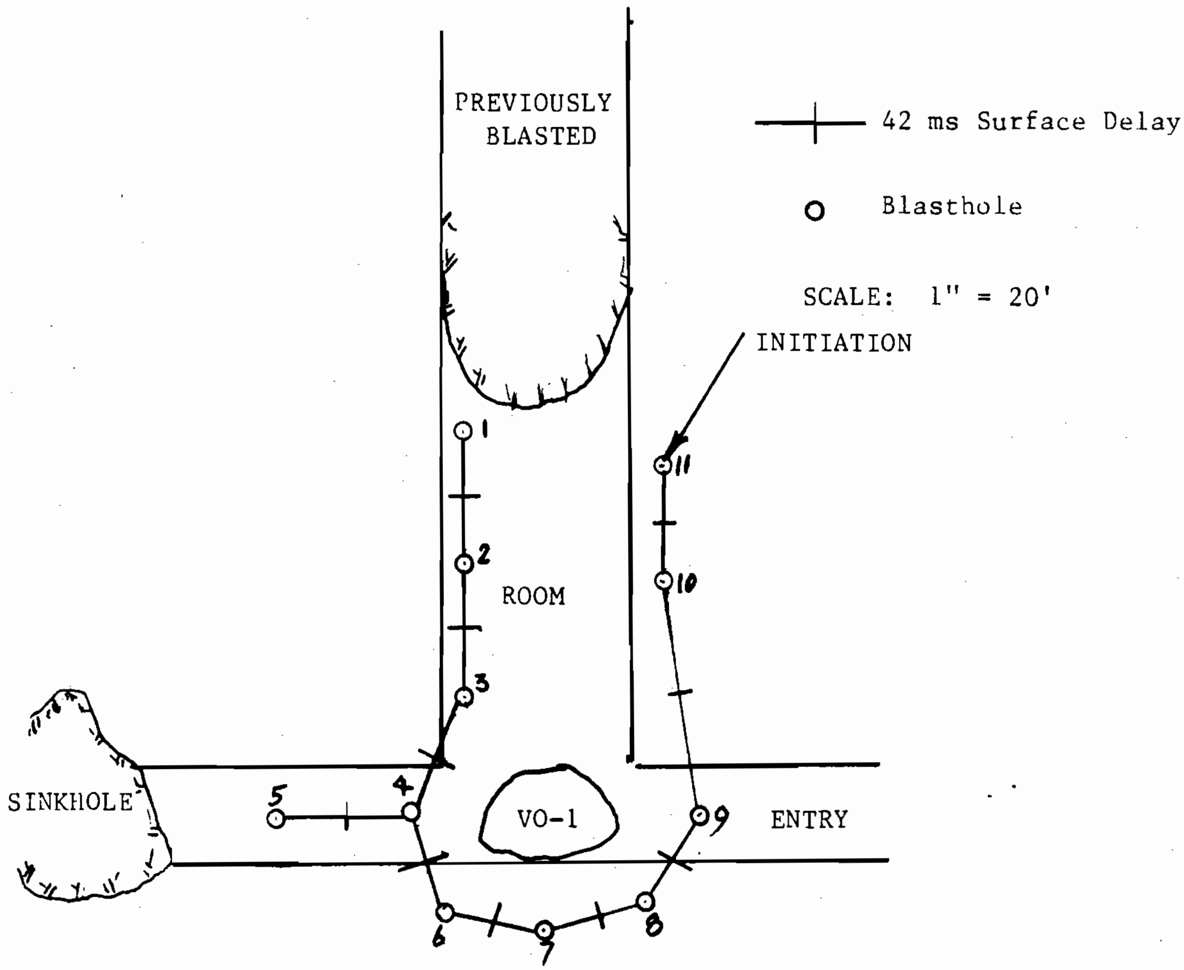

FIGURE A-1: BLAST LAYOUT AND TIE-IN FOR BLAST VO-1

AML PROJECT - BLAST SUMMARY

BLAST #: VO-2

LOCATION: S-2

DATE 8/5/88

HOLE DIAM 6"

NO. HOLES 7

EXPLOSIVES CONSUMPTION

ANFO 5 BAGS = 250 LBS   SLURRY 19 BAGS = 570 LBS

BOOSTERS 14   42 M/S DELAYS 7

DELAYS #4 7 #5 ___ #6 7 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: ___ LOCATION: n/a   PPV: ___ IN/SEC

CAMERA USED: n/a TARGET DELAY #: ___ FPS: ___

det. cord: 175'   Elect E.B. cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1					18			0		Over pillar - no void
2					18½			0		over pillar - no void
3					23½			0		" " " "
4		28			26½		31	2		Void indistinct - hard to measure
5				28 (fill)	45		?	?		Loose material
6		30			28¼		32	2		
7	54	59		28 (fill)	59	5	60			Distinct void either side

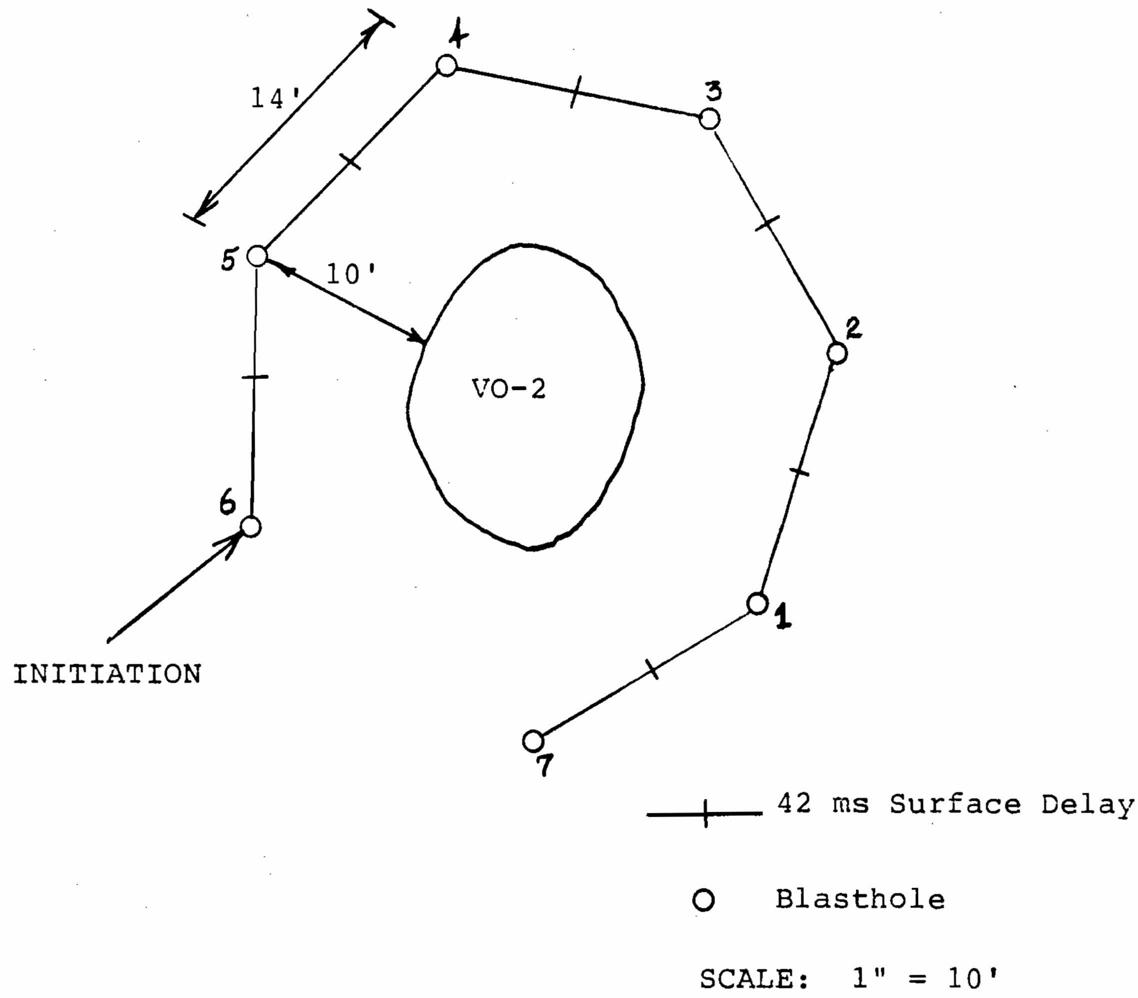

FIGURE A-2: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-2

AML PROJECT - BLAST SUMMARY

BLAST #: V0-3

LOCATION: Test Area #2

DATE 8/8/88

HOLE DIAM 6

NO. HOLES 5

EXPLOSIVES CONSUMPTION

ANFO 3 BAGS = 150 LBS SLURRY 9 BAGS = 270 LBS

BOOSTERS 9 42 M/S DELAYS 5

DELAYS #4 4 #5 ___ #6 5 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 500' NW of Blast PPV: 0.02 IN/SEC 108dB

CAMERA USED: No TARGET DELAY #: ___ FPS: ___

FIRED: 11:15 MDT overcast, 65°F, Wind SW 0-5mph, humid

det cord: 110' Elect. E.B. cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG C/L FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1					23¼		23¼	0		
2					24		24	0		
3		15		15	17¾					Struck void
4		11			9½					Struck void-Short do not load
5					24		24	0		
6					24½		24½	0		Load if #4A is not good.
4A					23		23	0		11 ft. from V.O.

#3 Loaded top deck only. Void @ 12' plugged @ 11' Loaded @ 10½ to 8½ feet with ANFO

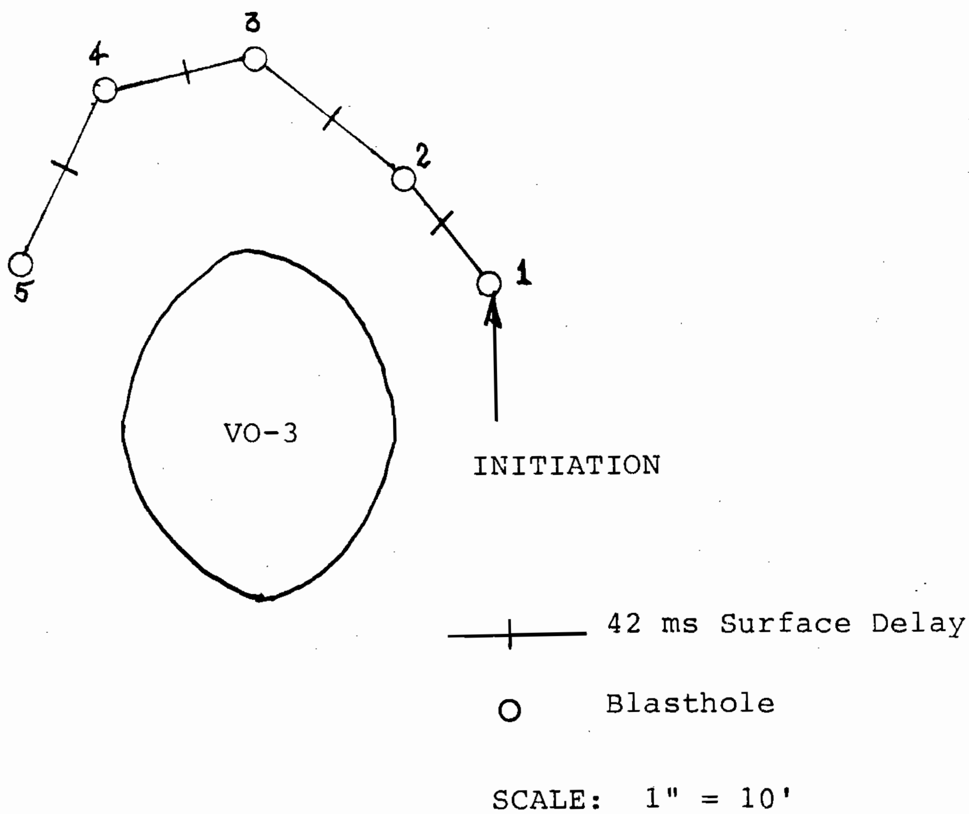

FIGURE A-3: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-3

AML PROJECT - BLAST SUMMARY

BLAST #: V0-4A

LOCATION: Test Area #2

DATE 8/8/88

HOLE DIAM 6

NO. HOLES 8

EXPLOSIVES CONSUMPTION

ANFO 5 BAGS = 250 LBS SLURRY 16 BAGS = 480 LBS

BOOSTERS 16 42 M/S DELAYS 8

DELAYS #4 8 #5 ___ #6 8 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 500' NW. of blast PPV: NO RDG. IN/SEC

CAMERA USED: No TARGET DELAY #: ____ FPS: ____

TIME 11:58 MDT Wind South 10-15mph, clear, 75°F

detonating cord: 160' Elect E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR Fill To	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				18	18			0		
2				18	18			0		16½' - Dirt fell in from Collar
3				24	25½			0		Fill to 24'
4		24	24	20	24		25	1		plug & Fill to 20' (18½')
5				18	18			0		
6				20	20			0		
7				18	20			0		Fill to 18'
8				18	19			0		Fill to 18'

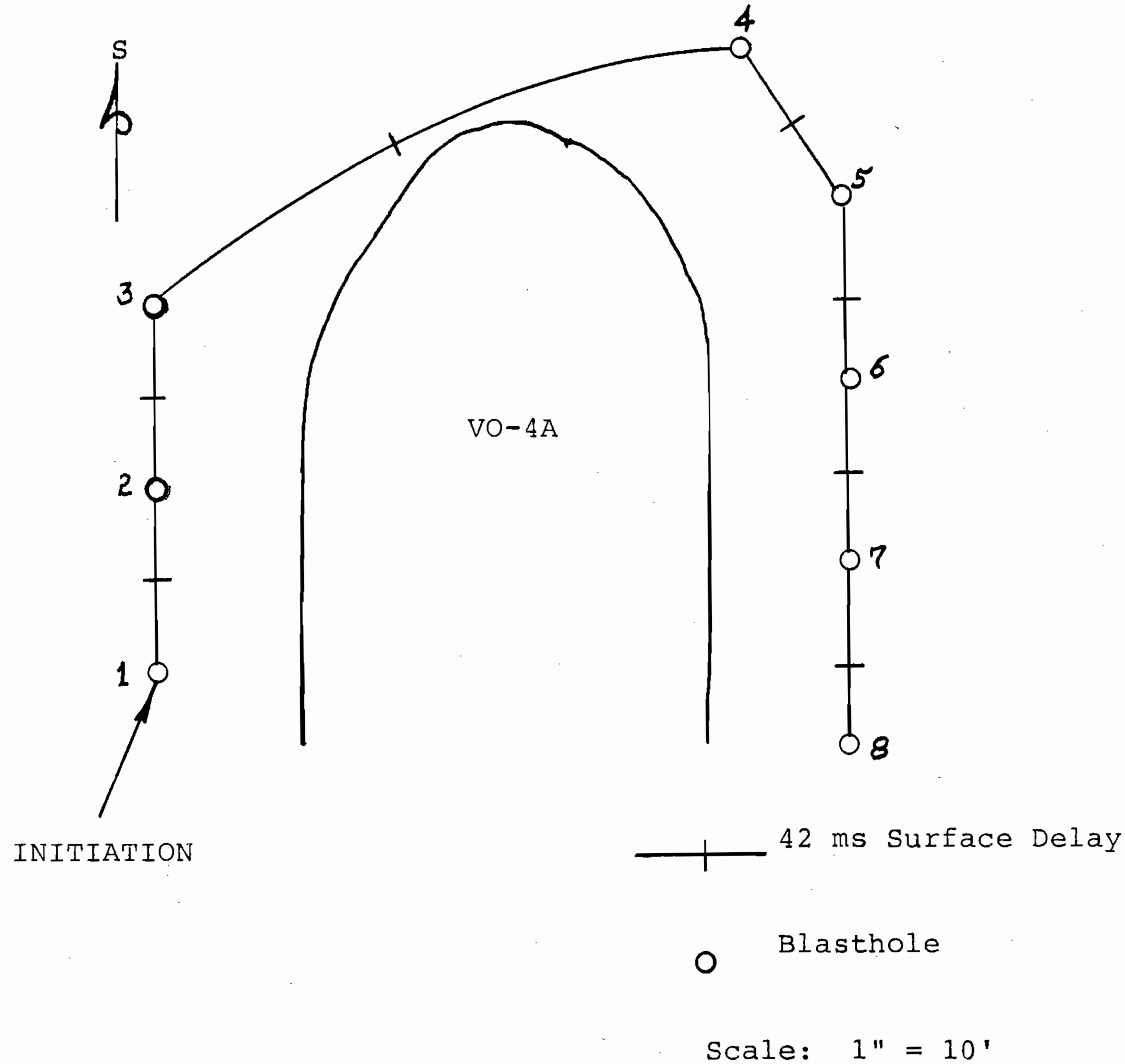

FIGURE A-4: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-4A

AML PROJECT - BLAST SUMMARY

BLAST #: V0-9A#2

LOCATION: Test Area #2

DATE 8/17/88

HOLE DIAM 6

NO. HOLES 3

EXPLOSIVES CONSUMPTION

ANFO 5 BAGS = 250 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 6 42 M/S DELAYS 3

DELAYS #4 3 #5 3 #6 ___ #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 313 ft. East of V0-9A#2 PPV: 0.03 IN/SEC

CAMERA USED: Yes TARGET DELAY #: ____ FPS: ____ FAILED TO START

TIME FIRED: 9:25 MST Partly Cloudy, 77°F WIND: North 10-12 mph

det. cord: 60' Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or fill To	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				17 ½	20			0		Second shot to fill 12'
2				18	20			0		deep rathole on south end
3				18	20			0		of first shot. Filled well.

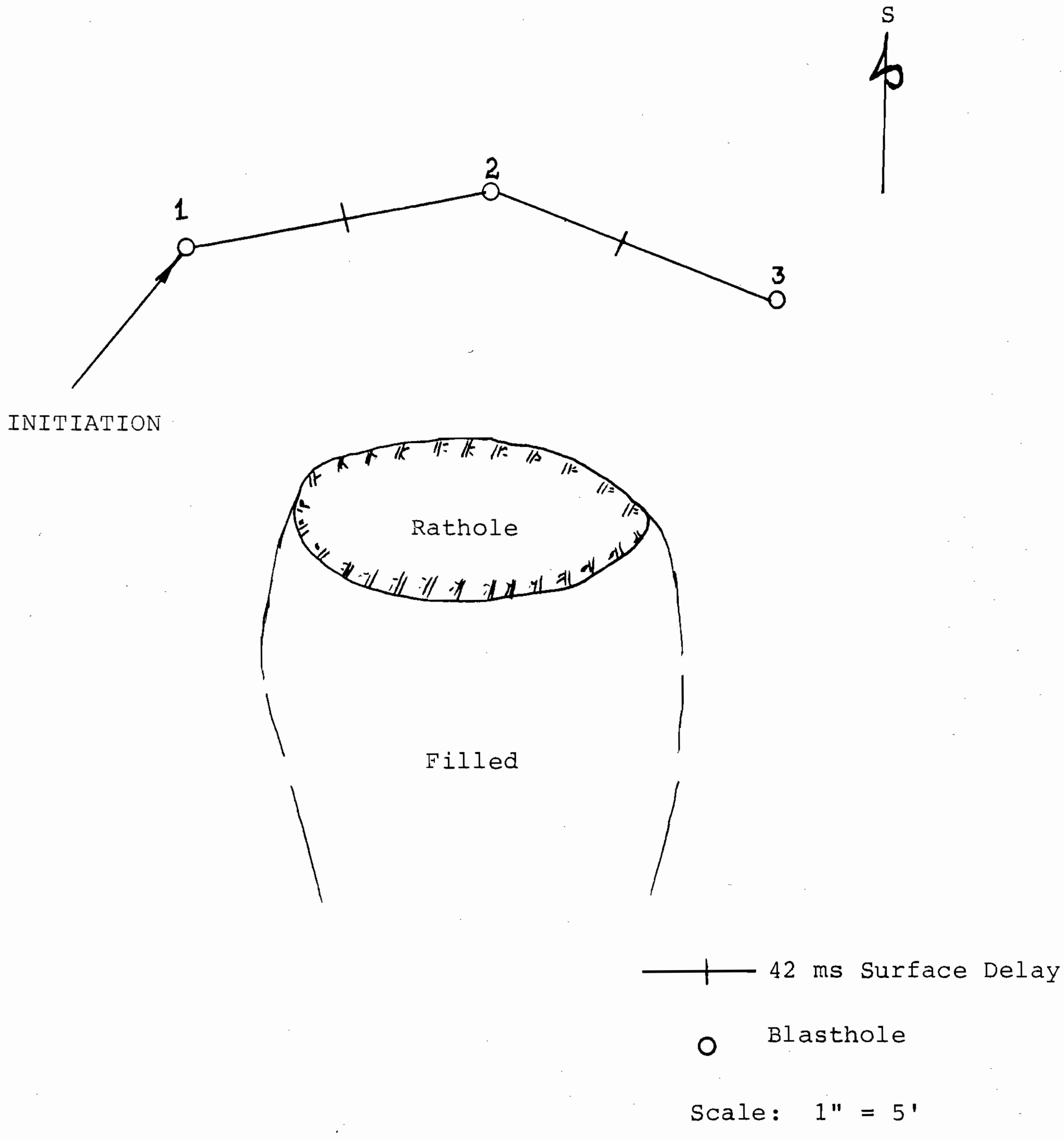

FIGURE A-5: BLASTHOLE LAYOUT AND TIE-IN
FOR BLAST VO-4A #2

AML PROJECT - BLAST SUMMARY

BLAST #: VO-4B

LOCATION: Test Area #2

DATE 8/17/88

HOLE DIAM 6

NO. HOLES 8

EXPLOSIVES COMSUMPTION

ANFO 12 BAGS = 600 LBS SLURRY 9 BAGS = 270 LBS

BOOSTERS 19 42 M/S DELAYS 8

DELAYS #4 8 #5 7 #6 4 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 322 ft S.E. of VO-4B PPV: 0.06 IN/SEC

CAMERA USED: Yes TARGET DELAY #: ____ FPS: ____ didn't start

TIME fired: 16:01 MDT, Partly Cloudy, 77°F, Wind north 10-15 mph.

detonating Cord: 200' Electric E.B. cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or Fill To	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				24	24			0		
2				18	20			3		Void. Plug @18' & Fill to 15'
3				30	30			0		
4				34	35			0		Fill to 34'
5				34	37			0		Fill to 34'
6				30	30½			0		Fill to 30'
7				24	24			0		
8				24	24'10"			0		Fill to 24'

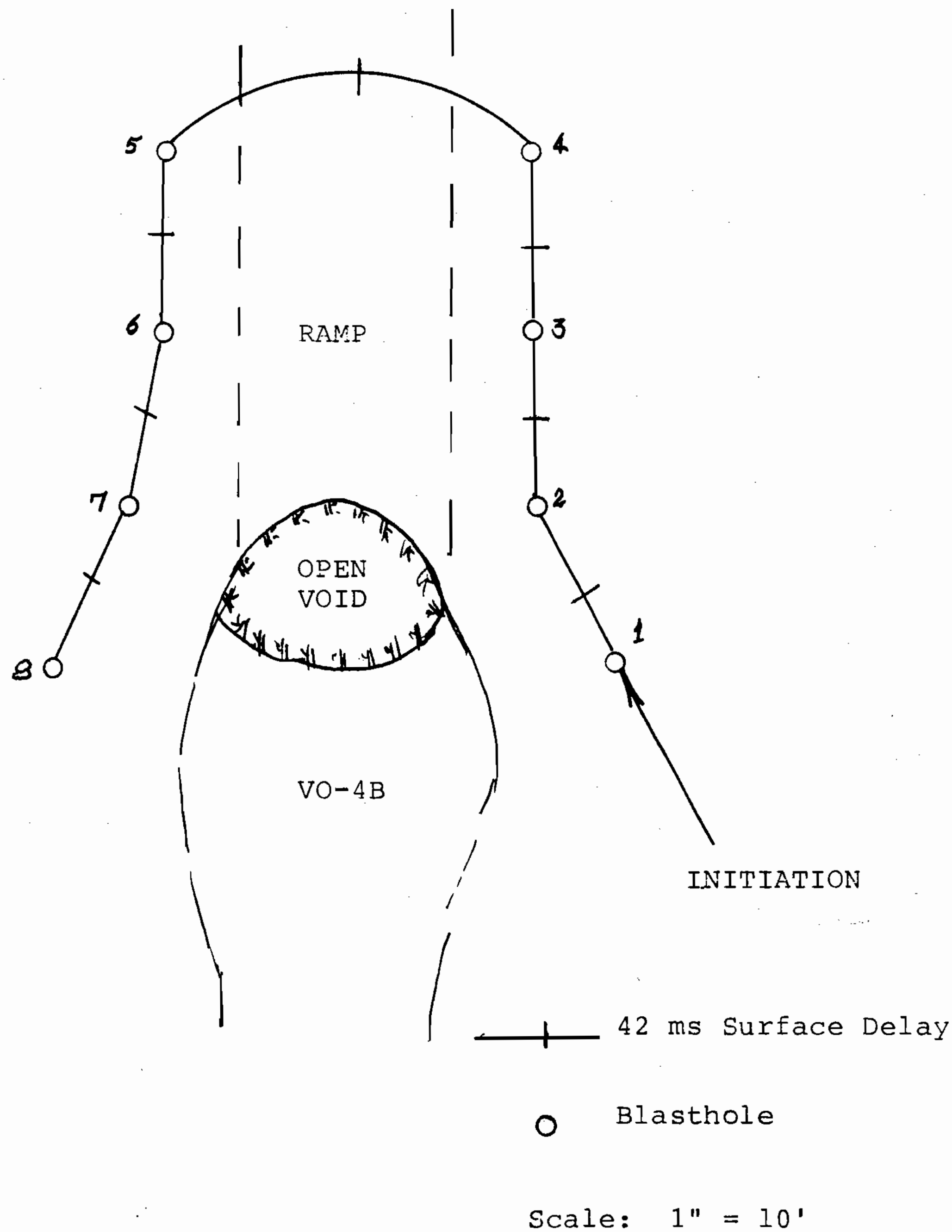

FIGURE A-6: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-4B

AML PROJECT - BLAST SUMMARY

BLAST #: VO-5

LOCATION: Test Area #2

DATE 8/16/88

HOLE DIAM 6"

NO. HOLES 13

EXPLOSIVES CONSUMPTION

ANFO 12 BAGS = 600 LBS SLURRY 12 BAGS = 360 LBS

BOOSTERS 22 42 M/S DELAYS 13

DELAYS #4 13 #5 ___ #6 9 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 300 ft. NW of VO-5 PPV: 0.45 IN/SEC

CAMERA USED: No TARGET DELAY #: ____ FPS: ____

Time Fired: 3:15 MDT Clear, 100°F

Elect E.B. cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or Fill To	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				18	25					Fill to 18 ft.
2				18	19			2?		Void? @ 18: Plug @ 18' Fill to 15'
3				23	23					
4				18	19					
5				17	17½					
6				23	23½					
7				28	29					Fill to 26'
8				20	21		29	8		Plug @ 20': Fill to 16'
9				16	17		20	3		Plug @ 16': Fill to 14'
10				18	18					
11				18	18					
12				23	23					
13				18	18					

#2 Void at 14': Plugged @ 13' and loaded from 12' to 9' with ANFO. (8/16/88)

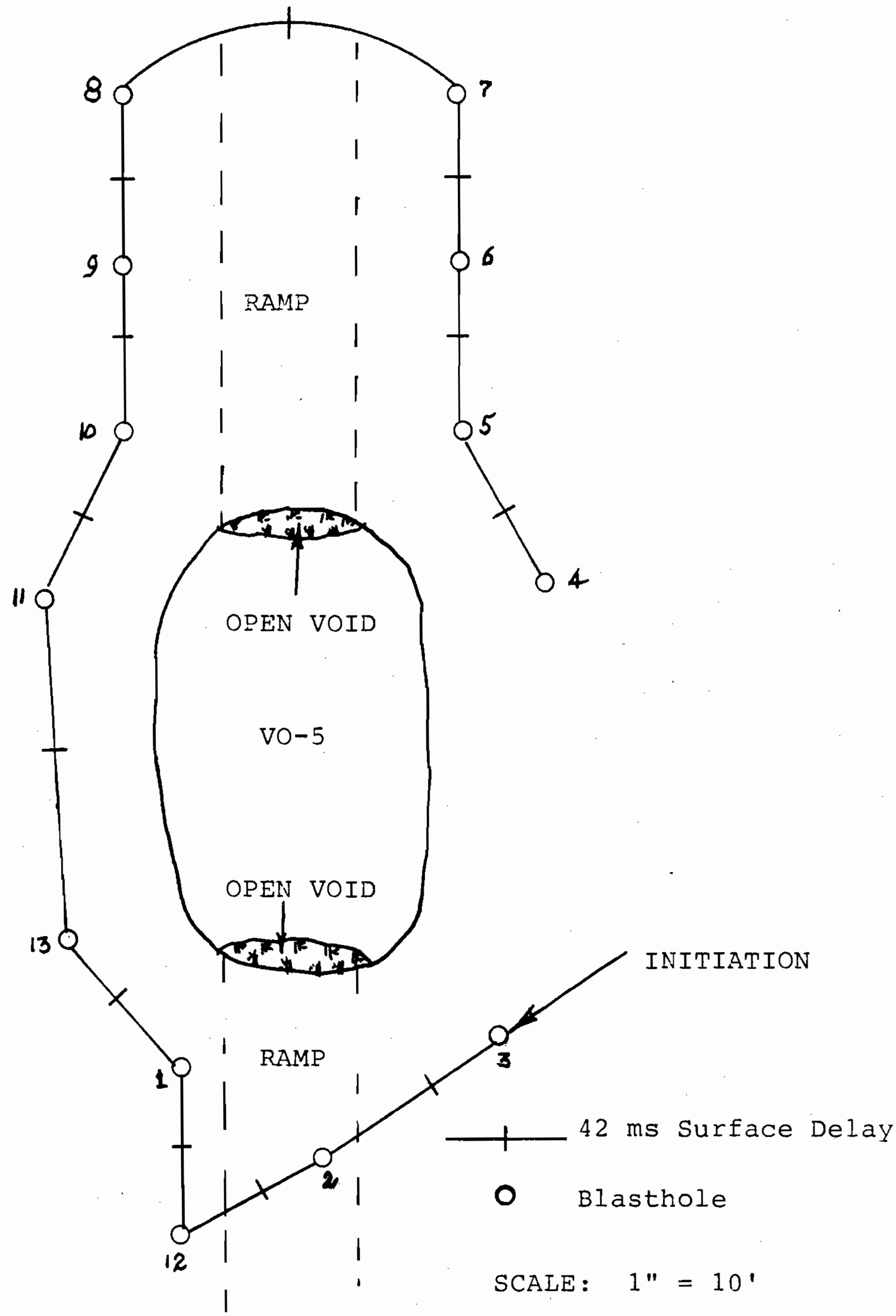

FIGURE A-7: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-5

AML PROJECT - BLAST SUMMARY

BLAST #: VO-6

LOCATION: Test Area #2

DATE 8/22/88

HOLE DIAM 6"

NO. HOLES 6

EXPLOSIVES CONSUMPTION

ANFO 8 BAGS = 400 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 9 42 M/S DELAYS 6

DELAYS #4 6 #5 ___ #6 3 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 406 ft. S.E. of VO-6 PPV: .02 IN/SEC

CAMERA USED: Yes TARGET DELAY #: n/a FPS: 500 f.stop: 7 Zoom Rdg: 120
Location: 450' NE. Focus: ∞ EV. rdg: 14 Az: 210° Vert Angle: 2°
Partly Cloudy, Wind West 25-30 mph

Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or Fill TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				14	14					
2				15	16					Fill to 15'
3				17	17					
4		22		18	20½		25	4½		Void @ 20½. Plug @ 20 & Fill to
5				18	18					
6				15	17½					Fill to 15'

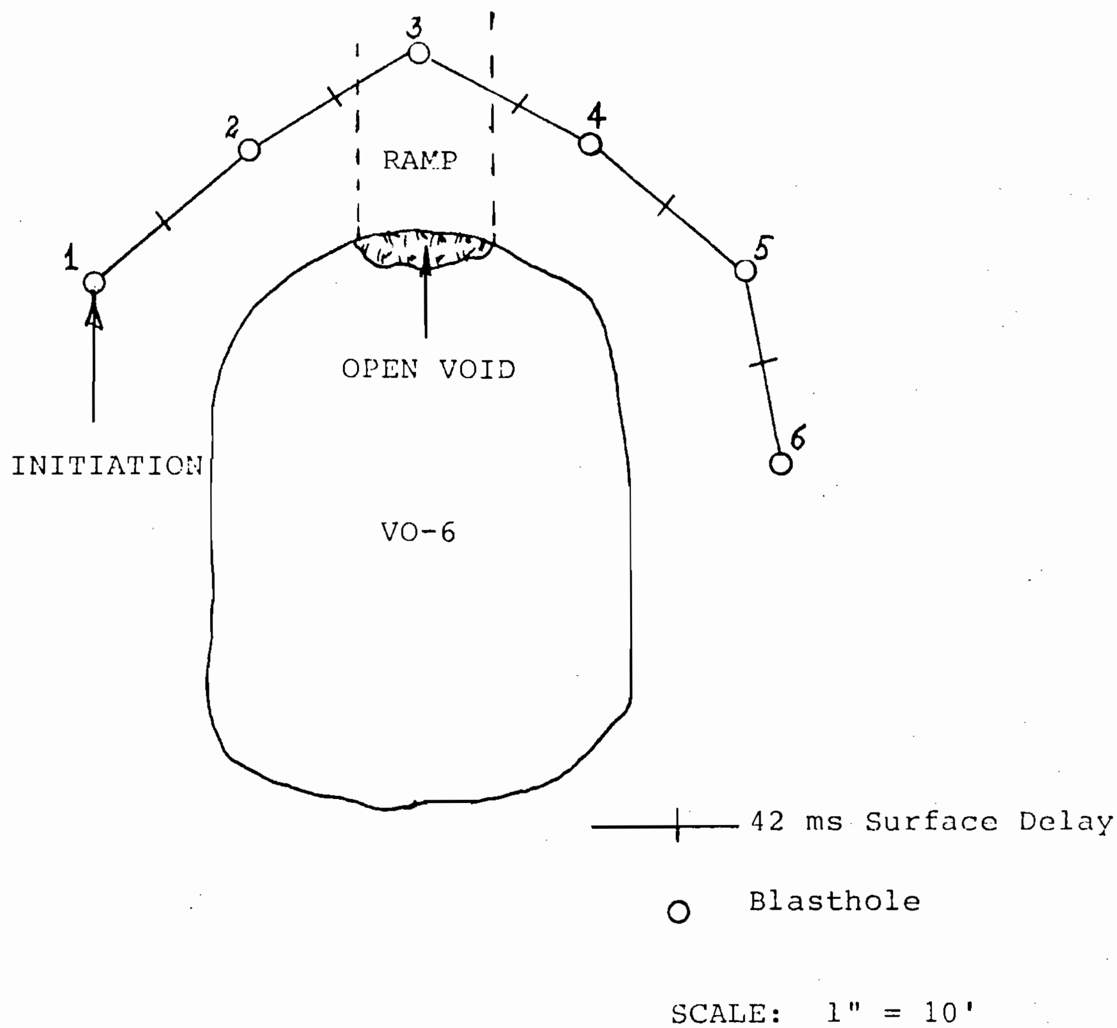

FIGURE A-8: BLAST LAYOUT AND TIE-IN
FOR BLAST VO-6

AML PROJECT - BLAST SUMMARY

BLAST #: VO-7N

LOCATION: Test Area #2

DATE 8/19/88

HOLE DIAM 6"

NO. HOLES 14

EXPLOSIVES CONSUMPTION

ANFO 44 BAGS = 2200 LBS SLURRY 22 BAGS = 660 LBS

BOOSTERS 60 42 M/S DELAYS 28

DELAYS #4 28 #5 ___ #6 22 #7 ___ #8 10 #9 ___

SEISMOGRAPH: Yes LOCATION: 500' NW of VO-7E8-N PPV: 0.02 IN/SEC

CAMERA USED: Yes TARGET DELAY #: n/a FPS: 500 f-stop: 8 focus: ∞

Location: ESE of VO-7&8N lens Setting 60mm vert Angle: -3° Az: 285° E.K 14.5

det Cord: 2000' Elect E.B. Cap: 1 Time Fired 4:20 MDT

#	TOC	VOID DRILLED	VOID MEAS	PLUG or Fill TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
13				15	15			0		
14				14	14½			0		Fill to 14'
15				15	18			0		Fill to 15'
16				18	18½			0		Fill to 18'
17				24	24			0		
18				25	26			0		Fill to 25'
19				30	30			0		
20				30	31½			0		Fill to 30'
21				35	36			0	34'-37'	Fill to 35'
22				35	35½			0	14'-16'	Fill to 35'
23				30	30			0	18'-19'	
24				30	30			0	15'-17'	
25				25	26			0		Fill to 25'
26				24	24½			0		Fill to 24'
27				18	18½			0		Fill to 18'

AML PROJECT - BLAST SUMMARY

BLAST #: VO-8N

LOCATION: TEST AREA #2

DATE 8/19/88

HOLE DIAM 6"

NO. HOLES 14

SEE VO-7N FOR DETAILS FIRED TOGETHER

EXPLOSIVES CONSUMPTION

ANFO ____ BAGS = ______ LBS SLURRY ____ BAGS = _______ LBS

BOOSTERS _______ 42 M/S DELAYS _____

DELAYS #4 ____ #5 ____ #6 ____ #7 ____ #8 ____ #9 ____

SEISMOGRAPH: ____ LOCATION: ____________ PPV: ______ IN/SE

CAMERA USED: _____ TARGET DELAY #: _____ FPS: _____

#	TOC	VOID DRILLED	VOID MEAS	PLUG or Fill to	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
3				15	17					Fill to 15'
4				15	16					" " "
5				15	16					" " "
6				20	20					
7				25	25				7'-9'	
8				23	23½					Fill to 23'
9				25	30		36½	6½'	10'-12'	Void @30'. Plug & Fill to 25'
10				29	35		45	10'	9'-11'	Void @35'. Plug & Fill to 30'
11				35	35				14'-17'	
12				29	29				12'-15'	
13				27	27½				11'-16'	Fill to 27'
14				25	28				11'-13'	Fill to 25'
15				18	18½					Fill to 18'
16				14	14½					Fill to 14'

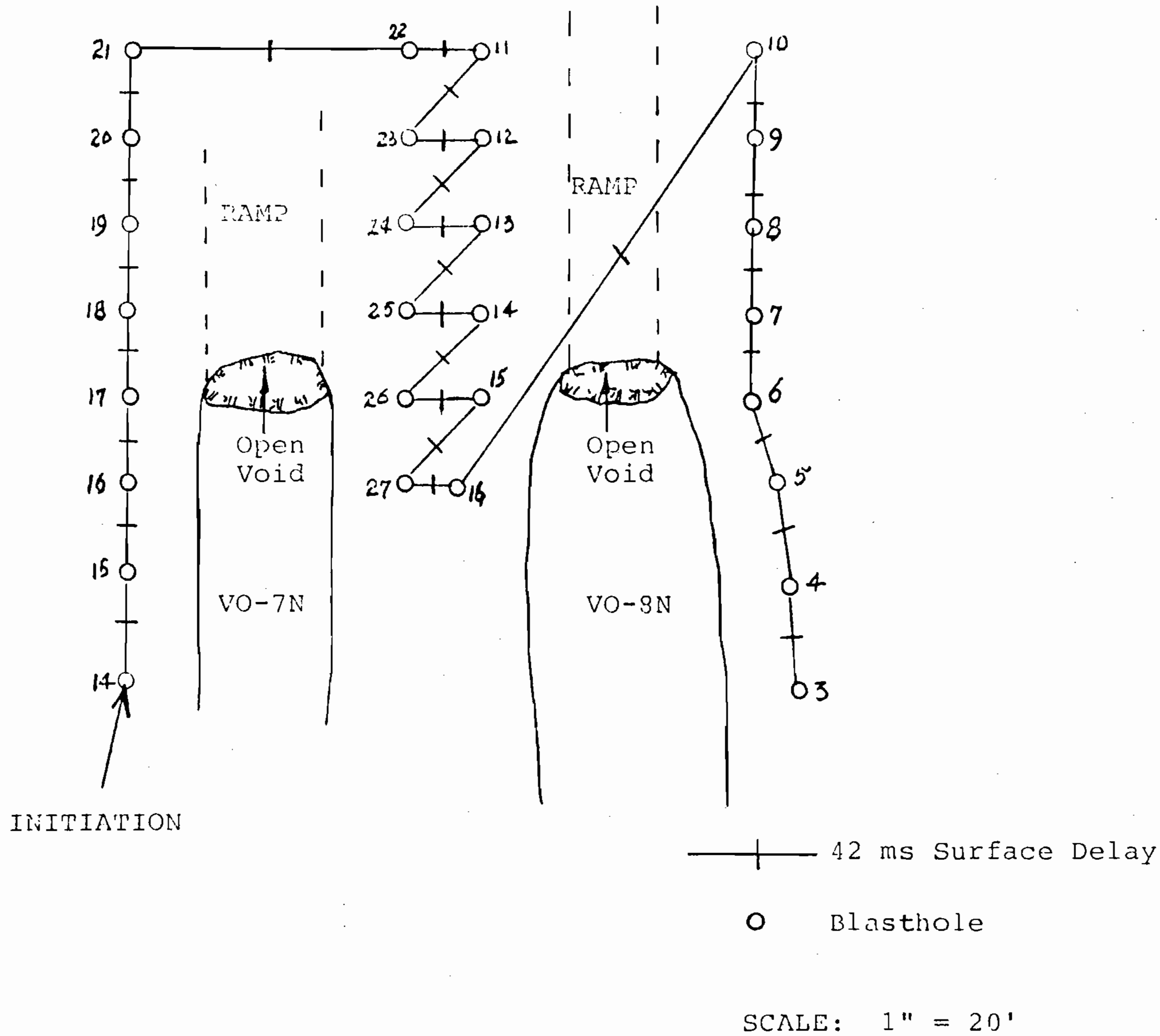

FIGURE A-9: BLAST LAYOUT AND TIE-IN FOR BLAST VO-7 & 8 NORTH

AML PROJECT - BLAST SUMMARY

BLAST #: VO-7&8-N#2

LOCATION: TEST AREA #2

DATE 8/25/88

HOLE DIAM 6"

NO. HOLES 9

EXPLOSIVES CONSUMPTION

ANFO 12 BAGS = 600 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 9 42 M/S DELAYS 9

DELAYS #4 ___ #5 ___ #6 ___ #7 ___ #8 ___ #9 9

SEISMOGRAPH: YES LOCATION: 500' W Between 7&8N#2 & 7S#2 PPV: 0.02 IN/SEC

CAMERA USED: YES TARGET DELAY #: — FPS: 500 f-stop: 5.0 focus: 45
Lens Setting 80mm Vert angle: -1/2° Az: 53° EV: 13 Distance 300' SW.
TIME FIRED: 11:32 MDT Clear, 77°F Wind N.W. 10 15mph

Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR Fill To	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15	17 1/2					Backfill to 15'
2				15	18 1/2					" " "
3				15	16 1/2					" " "
4				15	15					
5				15	17					Backfill to 15'
6				15	17					" " "
7				15	16 1/2					" " "
8				15	16 1/2					" " "
9				15	16					" " "

Second shot to fill potholes on north end of VO-7N & VO-8N. For VO-7 hole was 40' E to W and 21' N to S and 15' deep. ON VO-8N hole was ?5' E to W and 23' N to S and 14' deep. Shot filled holes quite well.

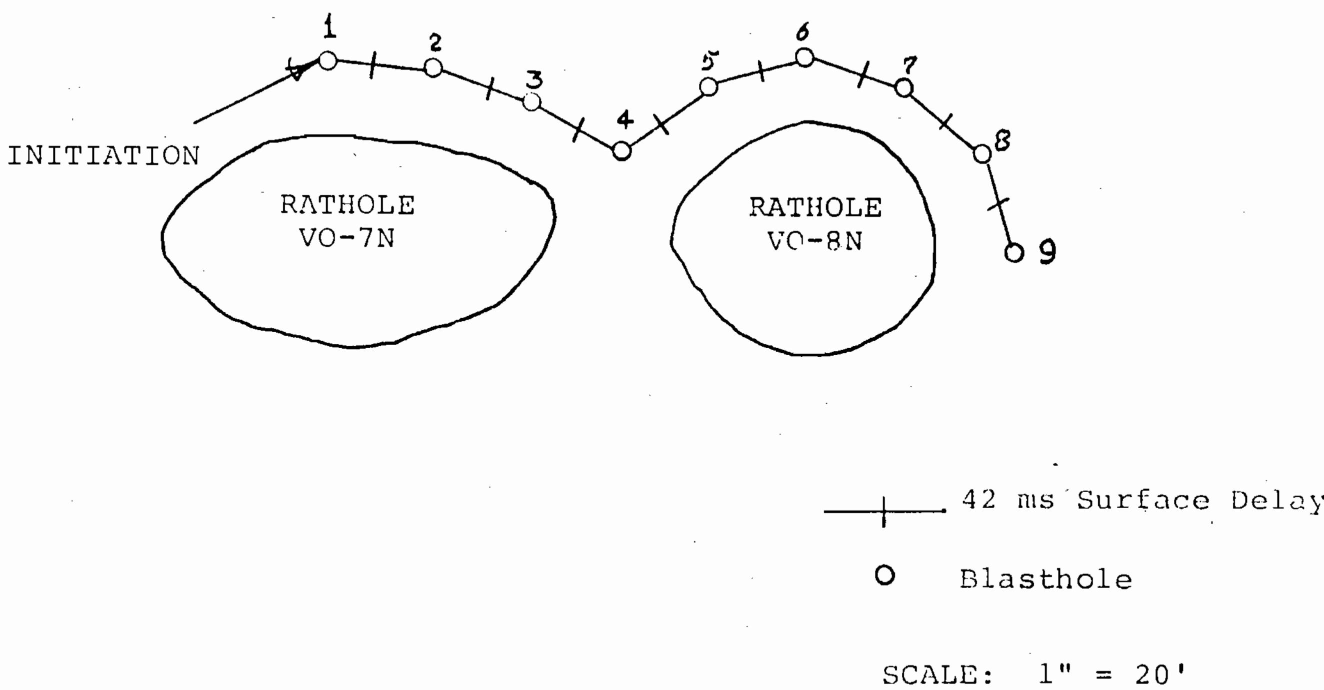

FIGURE A-10: BLAST LAYOUT AND TIE-IN FOR BLAST VO-7 & 8N #2

AML PROJECT - BLAST SUMMARY

BLAST #: VO-7 South

LOCATION: Test Area #2

DATE 8/23/88

HOLE DIAM 6"

NO. HOLES 13

EXPLOSIVES CONSUMPTION

ANFO 33 BAGS = 1650 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 42 42 M/S DELAYS 19

DELAYS #4 19 #5 ___ #6 16 #7 7 #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 500' WNW Between Shot and Sign yard PPV: 0.02 IN/SEC

CAMERA USED: Yes TARGET DELAY #: n/a FPS: 500 f-stop: 7 focus: ∞

Location: 300' NE of Blast lens Setting: 40mm vert. angle: -5° Az: 240° EV: 14

Time Fired 2:50 MDT Clear, 81°F, wind: 20-25 mph NW.

Elect. E.B. cup: 1

#	TOC	VOID DRILLED	~~VOID~~ DEPTH MEAS AFTER BLAST	PLUG or Fill to	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1			15½	15	16			0		Fill to 15'
2			18½	18	19			0		Fill to 18'
3			23	23	24½			0		
4			24	24	25			0		
5			29	29	30			0		
6			32	30	32			0		Fill to 30'
7			29½	29	30			0		Fill to 29'
8			29½	29	30			0		Fill to 29'
9			23	23	24			0		
10			26½	25	27			0		Fill to 25'
11			17	17	18			0		
12			13	13	14			0		
13			0	—	15			0		Lost Hole

Depth of Holes Measured After Blast VO-7 & 8 North was shot. All holes were Drilled Before To Insure Could get Drill Properly Located.

AML PROJECT - BLAST SUMMARY

BLAST #: VO-8 South

LOCATION: Test Area #2

DATE 8/23/88

HOLE DIAM 6"

NO. HOLES 8

SEE VO-7S FOR DEPTHS HOLES SHOT TOGETHER

EXPLOSIVES CONSUMPTION

ANFO ____ BAGS = ______ LBS SLURRY ____ BAGS = _______ LBS

BOOSTERS _______ 42 M/S DELAYS _____

DELAYS #4 ___ #5 ___ #6 ___ #7 ___ #8 ___ #9 ___

SEISMOGRAPH: ____ LOCATION: ______________ PPV: ______ IN/SEC

CAMERA USED: _____ TARGET DELAY #: _____ FPS: _____

#	TOC	VOID DRILLED	~~VOID~~ DEPTH MEAS AFTER BLAST	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1			0	0	13			0		REDRILL - COULDN'T DO. LOST HOLE
2			0	0	16½			0		" " " " "
17			18½	18	19½			0		Fill to 18'
18			22½	22	23½			0		Fill to 22'
19			25	25	26			0		
20			28	28	29½			0		
21			28½	28	29½			0		Fill to 28'
22			29	29	30			0		

See VO-7-South for Comments

Holes 1 & 2 lost due to blast on north end.

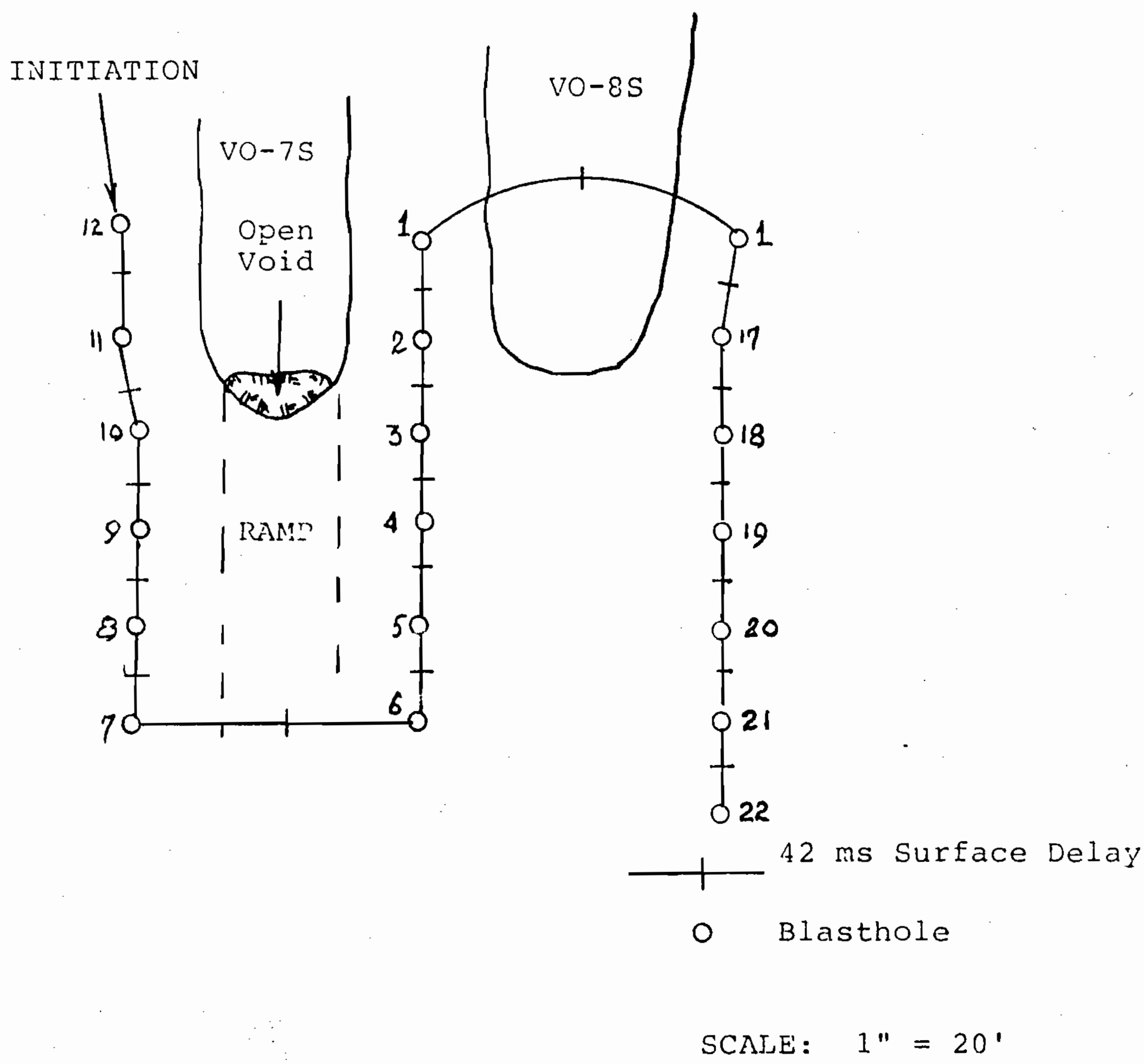

FIGURE A-11: BLAST LAYOUT AND TIE-IN FOR BLAST VO-7 & 8 SOUTH

AML PROJECT - BLAST SUMMARY

BLAST #: VO-75 #2

LOCATION: TEST AREA #2

DATE 8-25-88

HOLE DIAM 6"

NO. HOLES 4

EXPLOSIVES CONSUMPTION

ANFO 6 BAGS = 300 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 8 42 M/S DELAYS 4

DELAYS #4 4 #5 4 #6 ___ #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 500' W. Between 7&8 N#2 and 75#2 PPV: 0.02 IN/SEC

CAMERA USED: YES TARGET DELAY #: ____ FPS: 500 SEE VO-7&8 N#2 for details

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR Fill TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				17	17½					Fill to 17'
2				17	17					Fill to 17'
3				19	19½					Fill to 19'
4				19	19					

Two holes #5 & #6 were staked out but couldn't be drilled due to disruption from first blast.

Rathole was 25' from N to S and 8' deep at South end.

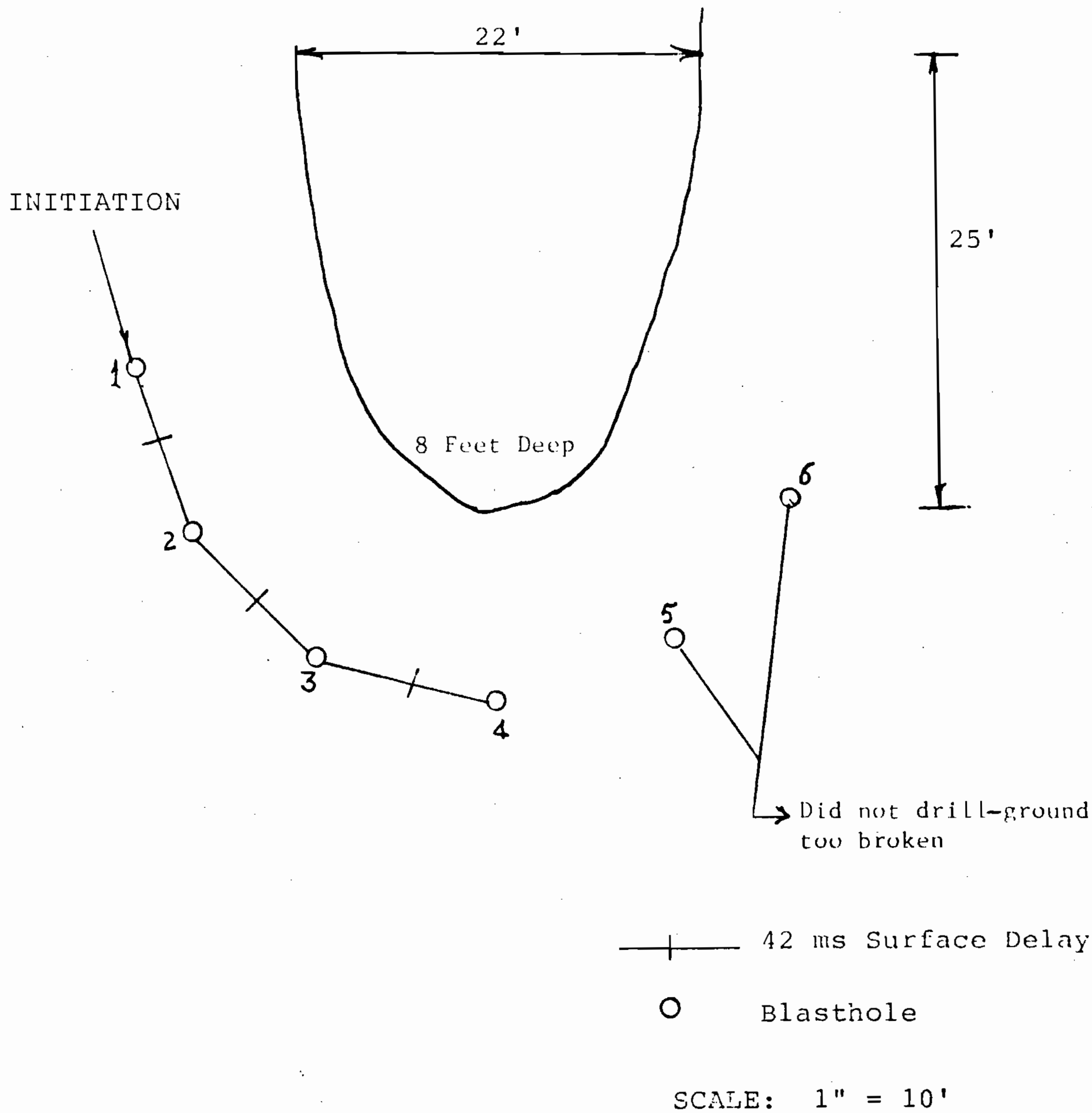

FIGURE A-12: BLAST LAYOUT AND TIE-IN FOR BLAST VO-7S #2

AML PROJECT - BLAST SUMMARY

BLAST #: VO-9

LOCATION: TEST AREA #2

DATE 8/18/88

HOLE DIAM 6"

NO. HOLES 10

EXPLOSIVES CONSUMPTION

ANFO 8 BAGS = 400 LBS SLURRY 27 BAGS = 810 LBS

BOOSTERS 20 42 M/S DELAYS 10

DELAYS #4 10 #5 ___ #6 10 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 311' East of VO-9 PPV: 0.04 IN/SEC

CAMERA USED: YES TARGET DELAY #: n/a FPS: 500 f-stop: 8 focus: ∞

Lens Setting: 70mm EV. 14.5 Location: 311' East

TIME FIRED: 4:18 MDT Clear, 80°F, calm

det. Cord: 225' Elect E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				20	24		27	3		Plug @ 24'. Fill to 20'
2				19	19			0		
3				20	20			0		
4				25	25½			0		Fill to 25'
5				20	20			0		
6				19	19			0		
7				19	19			0		
8				19	25½			?		Void @ 27' Plugged at 25' instead of @ 23'. Fill to 19
9				25	25			0		
10				25	25			0		

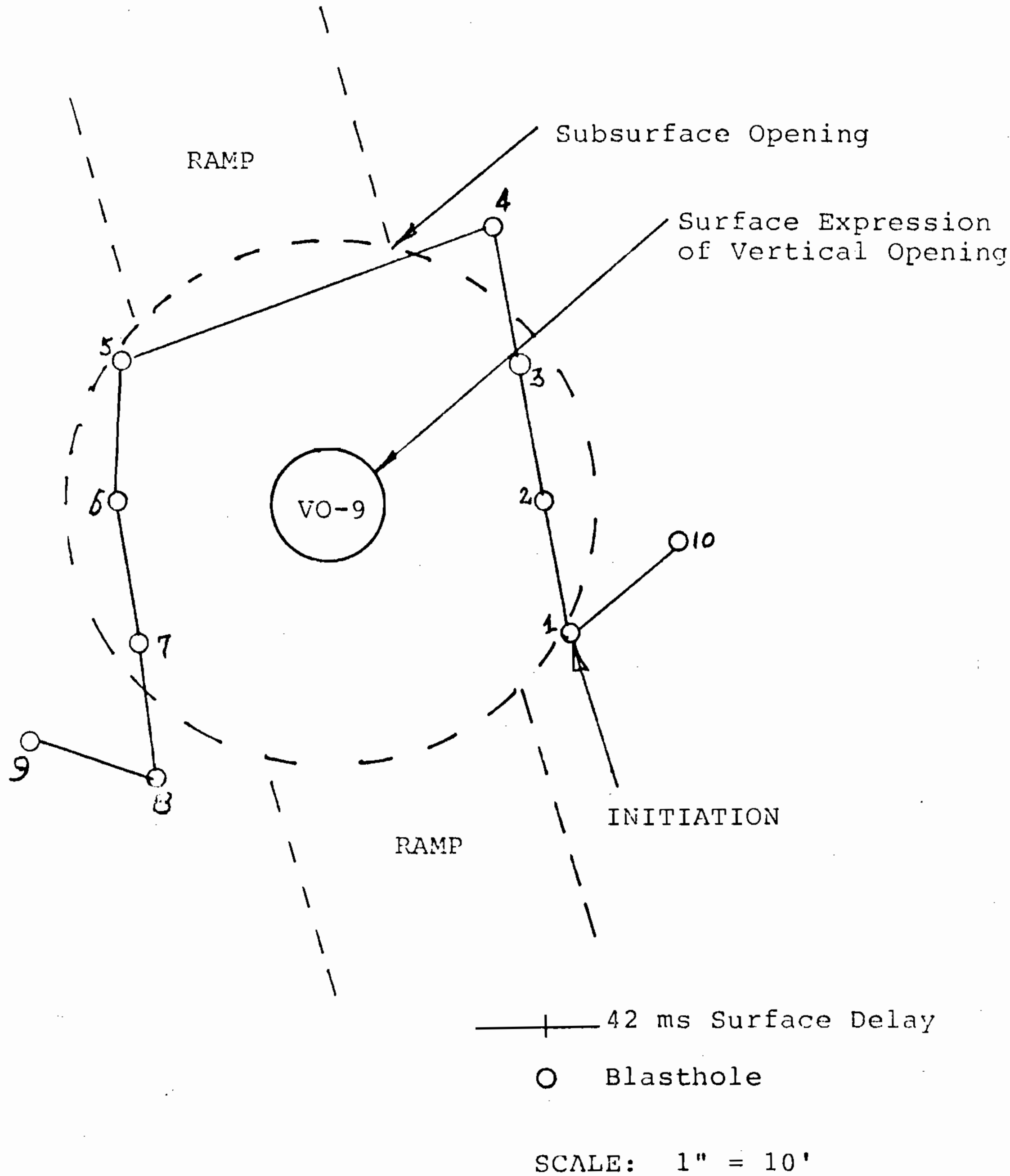

FIGURE A-13: BLAST LAYOUT AND TIE-IN FOR BLAST VO-9

AML PROJECT - BLAST SUMMARY

BLAST #: VO-9#2

LOCATION: TEST AREA #2

DATE 8-24-88

HOLE DIAM 6"

NO. HOLES 6
2 at North End
4 at South End

EXPLOSIVES CONSUMPTION

ANFO 8 BAGS = 400 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 8 42 M/S DELAYS 6

DELAYS #4 6 #5 ___ #6 2 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 450' NW of VO-9#2 BETWEEN SHOT AND SHERMAN YARD PPV: 0.04 IN/SEC

CAMERA USED: NO TARGET DELAY #: ____ FPS: ____

Time Fired: 6:13 MDT Clear, 76°F, NW wind 15-20MPH

Electric E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15'	17 1/2			0		Fill to 15'
2				15	16 1/2			0		Fill to 15'
3				15	16 1/2			0		Fill to 15'
4				19	19			0		
5				19	19 1/2			0		Fill to 19'
6				15	15 1/2			0		Fill to 15'

SECOND SHOT TO CLOSE an open rathole at south end and a closed rathole at north end. Hole at South is 22' deep and at north is 11'

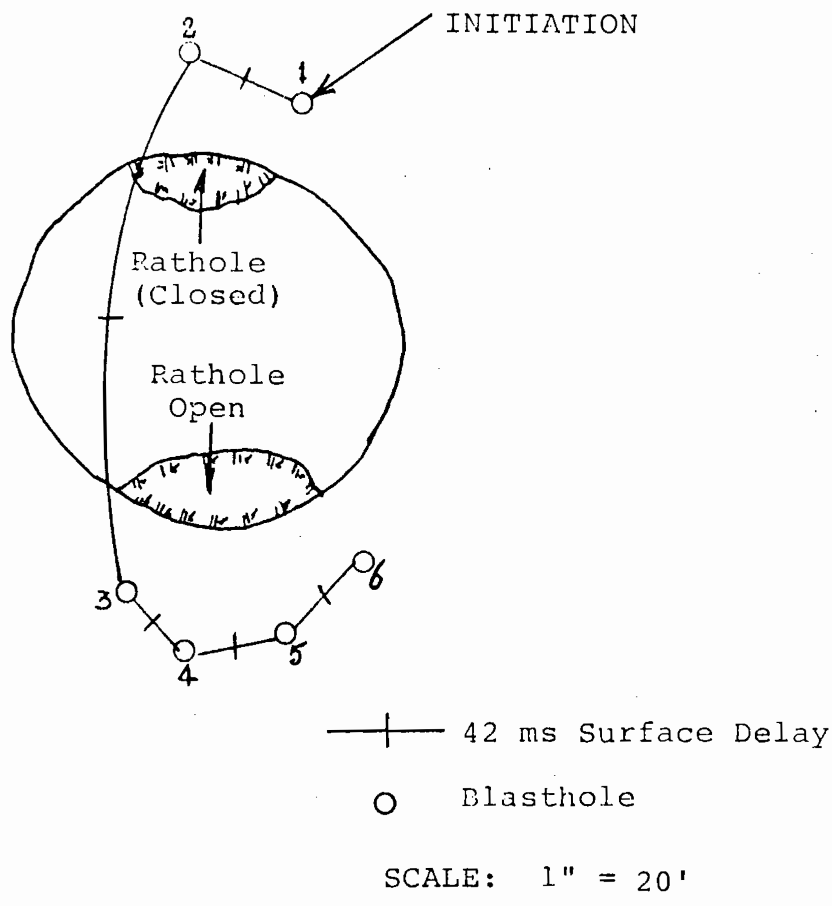

FIGURE A-14: BLAST LAYOUT AND TIE-IN FOR BLAST VO-9 #2

AML PROJECT - BLAST SUMMARY

BLAST #: VO-10

LOCATION: TEST AREA #2

DATE 8/16/88

HOLE DIAM 6"

NO. HOLES 5

EXPLOSIVES CONSUMPTION

ANFO 5 BAGS = 250 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 6 42 M/S DELAYS 5

DELAYS #4 5 #5 1 #6 ___ #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: Did not record PPV: ___ IN/SEC

CAMERA USED: No TARGET DELAY #: ___ FPS: ___

TIME FIRED 10:15 MDT

ELECT. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15'	15			0		
2				13'	15 1/2			3		Void. Plug & Fill to 13'
3				11'	14			4		" " " " to 11'. Stem 7'10
4					18			0		
5				14	14'9"			0		Fill to 14'

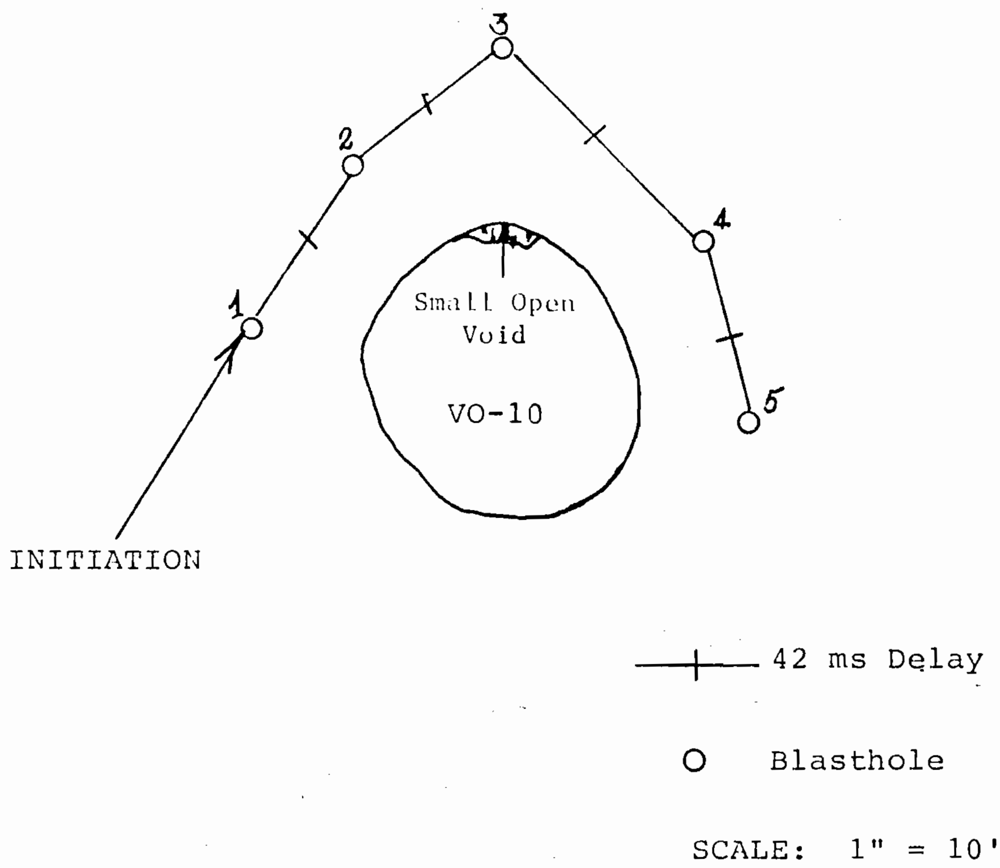

FIGURE A-15: BLAST LAYOUT AND TIE-IN FOR BLAST VO-10

AML PROJECT - BLAST SUMMARY

BLAST #: VO-11

LOCATION: TBST Area #3

DATE: 8/30/88

HOLE DIAM: 6"

NO. HOLES: 22

EXPLOSIVES CONSUMPTION

ANFO 39 BAGS = 1950 LBS   SLURRY 7 BAGS = 210 LBS

BOOSTERS 41   42 M/S DELAYS 22

DELAYS #4 16 #5 ___ #6 16 #7 9 #8 ___ #9 ___

SEISMOGRAPH: yes LOCATION: 300' S.E. of VO-11   PPV: 0.05 IN/SEC

CAMERA USED: YES TARGET DELAY #: — FPS: 500 f-stop: 8 focus: ∞

Location: 300' SE of VO-11 Lens Ldg: 50mm EV: 15 AZ: 325 Vert angle: -3°

Elect E.B. Cap: 1   Time Fired: 12:44 MDT   HALE   Wind SSE 20-25 mph

#	TOC	VOID DRILLED	VOID MEAS	PLUG or FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15	17			0	10'-13'	Fill to 15'
2				15	17			0		" " "
3				15	16			0		" " "
4				20	20			0		
5				19	19			0		
6				24	24½			0		Fill to 24'
7				19	19			0		
8				25	26			0		Fill to 25'
9				25	26			0		" " "
10				25	26			0		" " "
11				25	26			0		" " "
12				28	33			1		Plug @32'. Fill to 28'
13				25	26			0		Fill to 25'
14				28	33			2		Plug @32'. Fill to 28'
15				30	35			2		Plug @34' Fill to 30'
16				25	27			0		Fill to 25'
17				24	25			0		Fill to 24'
18				19	19			0		
19				19	19			0		
20				15	17			0		Fill to 15'
21				15	16			0		" " "
22				15	16					" " "

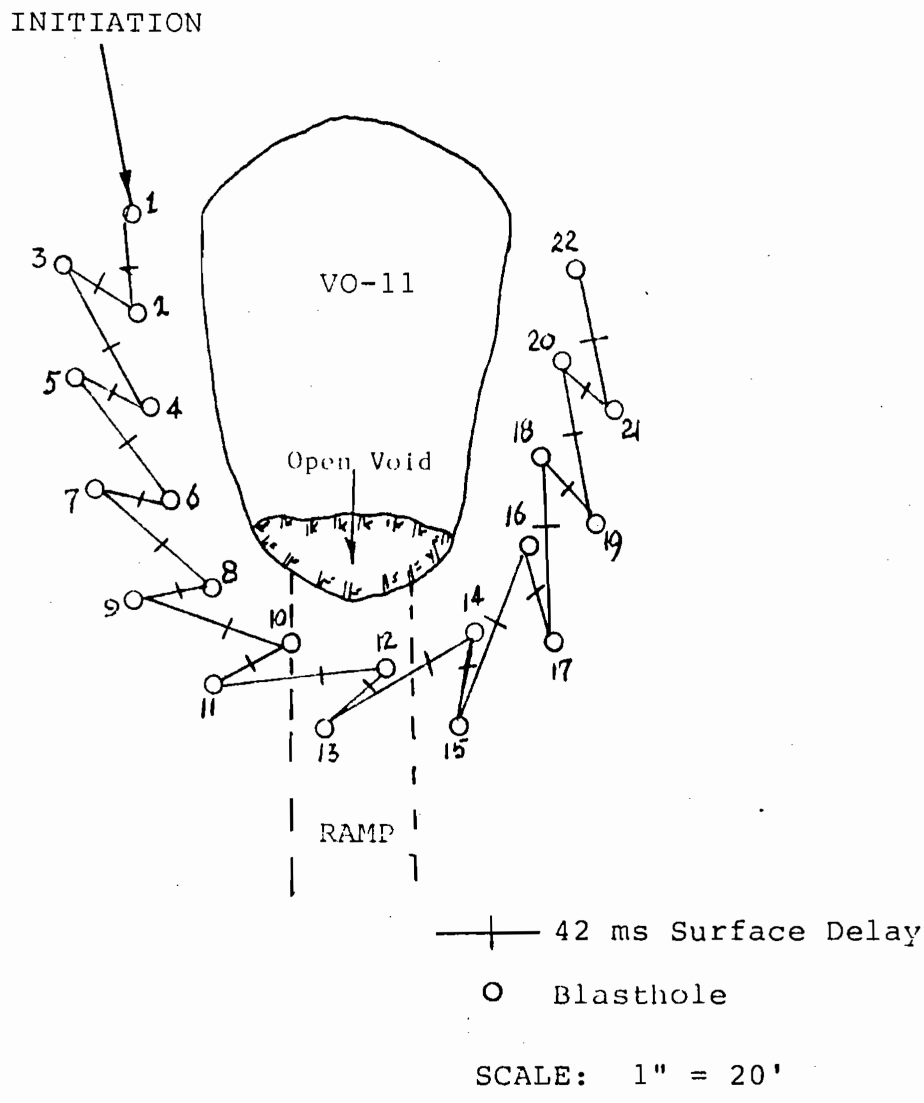

FIGURE A-16: BLAST LAYOUT AND TIE-IN FOR BLAST VO-11

AML PROJECT - BLAST SUMMARY

BLAST #: VO-12

LOCATION: TEST AREA #3

DATE 8/31/88

HOLE DIAM 6"

NO. HOLES 14

EXPLOSIVES CONSUMPTION

ANFO 25 BAGS = 1250 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 24 42 M/S DELAYS 13

DELAYS #4 11 #5 ___ #6 11 #7 2 #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 300' SW OF VO-12 PPV: 0.02 IN/SEC

CAMERA USED: YES TARGET DELAY #: 1/6 FPS: 350 f-stop: 2.8 focus: ∞
LOCATION: 300' SW Lens Setting: 70MM AZ: 90° Vert angle: -3° EV: 12.25

Elect E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15	16½			0		Fill to 15'
2				19	19			0		
3				25	26			0		Fill to 25'
4				18'	23		25	2		Plug @22' - actual @ 18'
5				20'	25			2		Plug @~~24~~': Fill to 20'
6				19'	22 ~~23½~~		23½	1½		Plug @ 21': Fill to 19'
7				24	24			0		
8				17	21			0		Plug @18': Fill to 17'
9				23	23½			0		Fill to 23'
10				25	25½			0		Fill to 25'
11				24	24			0		~~Fill to 24'~~
12				25	25½			0		Fill to 25'
13				15	17			0		Fill to 15'
14				0	14½			4		VOID AT 8': DO NOT LOAD

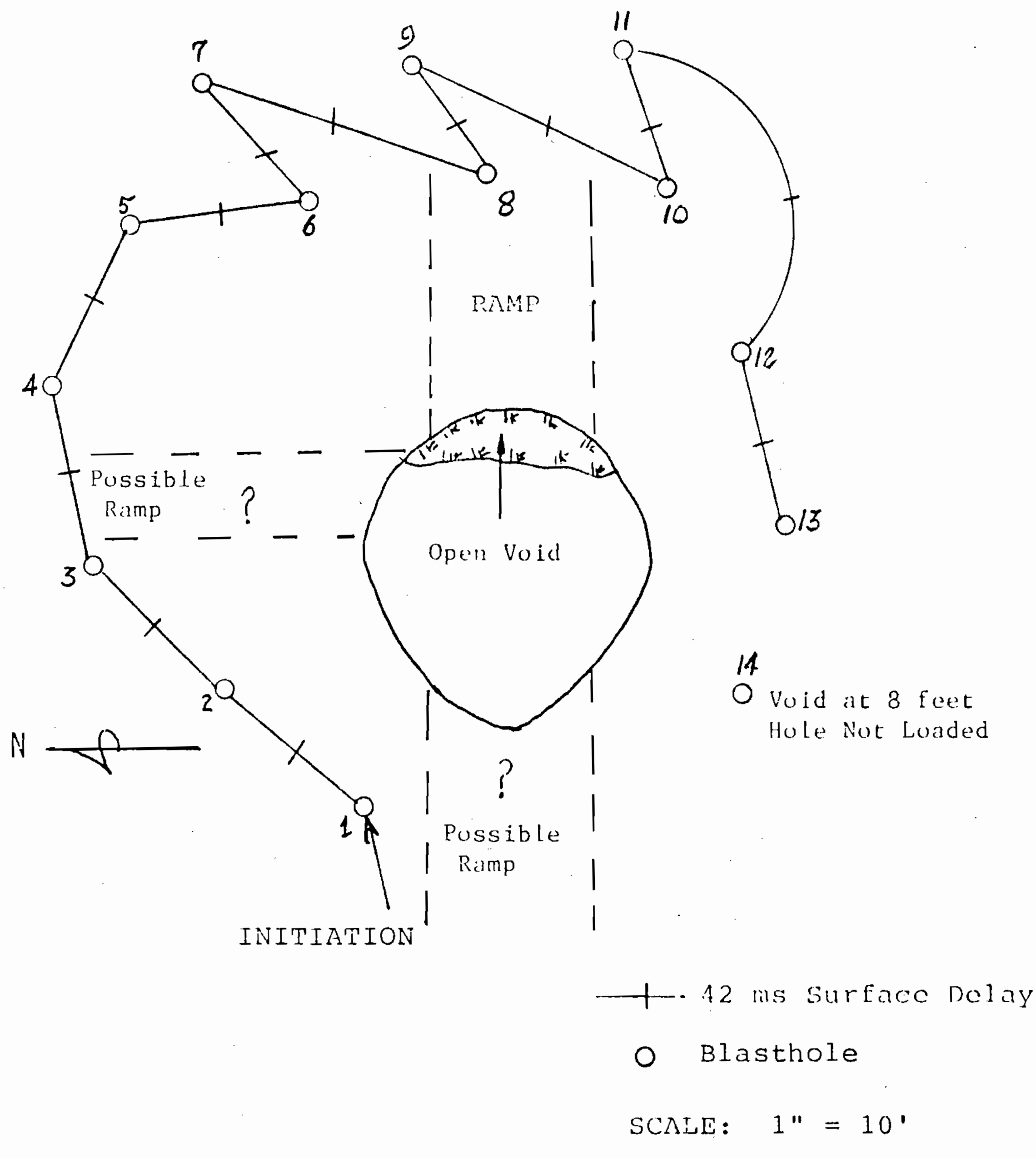

FIGURE A-17: BLAST LAYOUT AND TIE-IN FOR BLAST VO-12

AML PROJECT - BLAST SUMMARY

BLAST #: VO-13

LOCATION: TEST Area #3

DATE 8/31/88

HOLE DIAM 6"

NO. HOLES 21

EXPLOSIVES CONSUMPTION

ANFO 25 BAGS = 1250 LBS SLURRY 34 BAGS = 1020 LBS

BOOSTERS 37 42 M/S DELAYS 21

DELAYS #4 11 #5 ___ #6 11 #7 15 #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 450' SE OF VO-13 PPV: 0.04 IN/SEC

CAMERA USED: YES TARGET DELAY #: N/A FPS: 250 Lens Setting: 9mm focus: ∞

AZ: 85° VERT ANGLE: -3° LOCATION: W. of VO-13 EV: 11

Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG OR FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15	15½			0		ALL ANFO
2				15	15½			0		" "
3				15	16½			0		" "
4				15	16½			0		" "
5				15	16½			0		" "
6				19	19			0		" "
7				24	24			0		" "
8				24	24			0		" "
9				30	31			0		ALL HANFO
10				35	36			0		" " IN Bottom 2 dec
11				30	35		40	5		Plug @ 35'. Fill to 30' all H
12				35	35			0		HANFO IN Bottom 2 deck
13				30	30			0		" " " " "
14				24	24			0		HANFO IN BOTTOM deck
15				25	25			0		" " " "
16				18	18½			0		" " " "
17				15	15½			0		ALL ANFO
18				15	15½			0		" "
19				15	16			0		" "
20				15	15½			0		" "
21				15	15½			0		" "

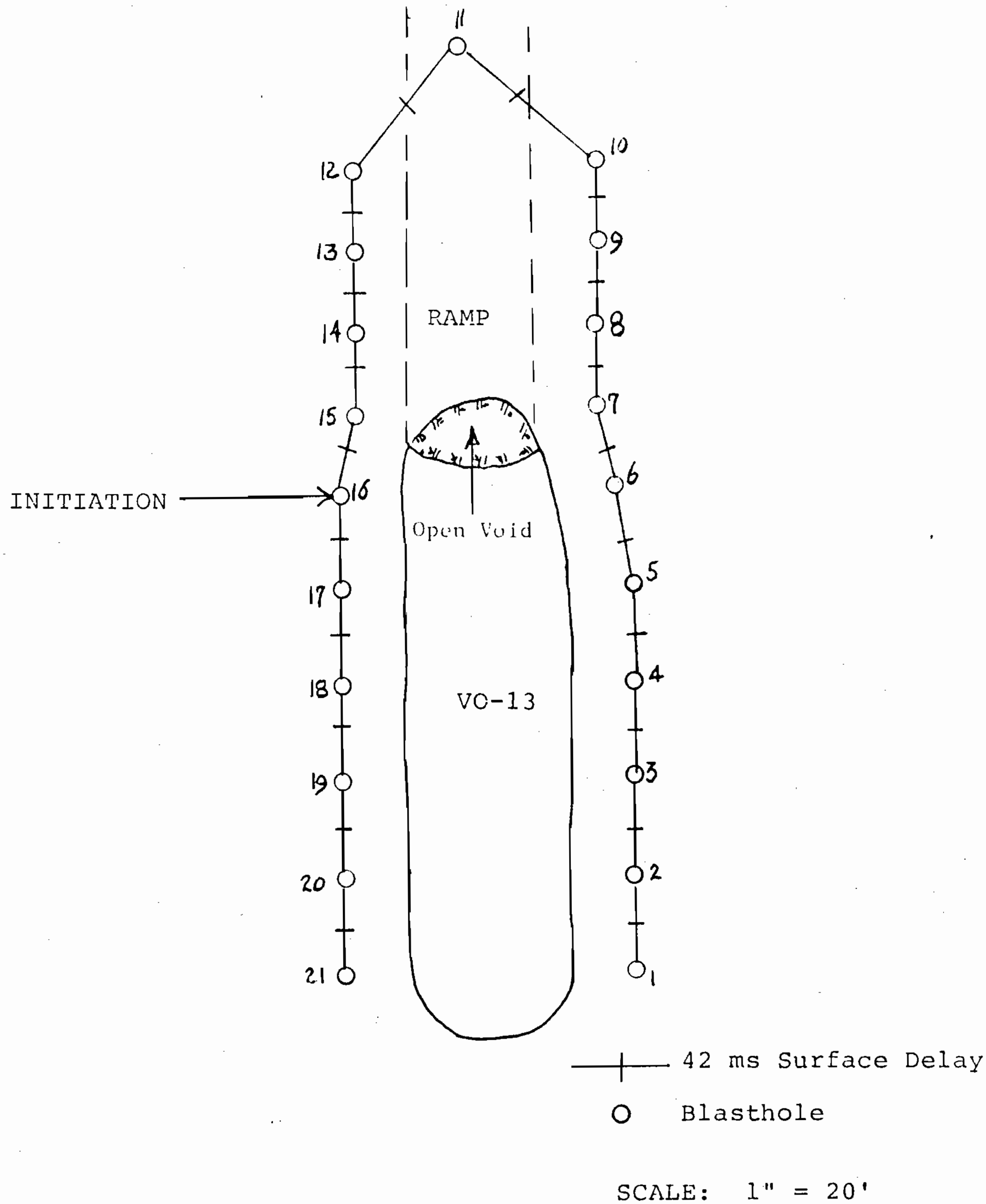

FIGURE A-18: BLAST LAYOUT AND TIE-IN FOR BLAST VO-13

APPENDIX B

BLAST SUMMARY SHEETS AND TIE-IN DIAGRAMS FOR THE WHITE SITE

AML PROJECT - BLAST SUMMARY

BLAST #: B1

LOCATION: WHITE

DATE: 9-12-86

HOLE DIAM: 6"

NO. HOLES: 12 (1-12)

EXPLOSIVES COMSUMPTION

ANFO 13 BAGS = 650 LBS   SLURRY 10 BAGS = 300 LBS

BOOSTERS 17   42 M/S DELAYS 12

DELAYS #4 12 #5 ___ #6 5 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 465' North of B1 PPV: 0.16 IN/SEC

CAMERA USED: YES TARGET DELAY #: - FPS: 400 f-stop: 4 focus: ∞

LOCATION: 465' N of B1 LENS SETTING: 70MM AZ: 195° VERT. ANGLE: -3° EV: 14 FILM: 160ASA

Elect. E.B. cup: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG or FILL TO	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
1				15	19		19			Fill to 15'
2				15	19		19			Fill to 15'
3		17		14	17		19½	2½		broke thru' at 16'
4				15	19½		19½		15'	Fill to 15'
5				15	19		19		15	Fill to 15'
6		18	17	16	18		21½	3½		Plug @ 16'
7		20	19	18	20	1	23½	3½		plug @ 18'
8		21	20	19	21		25	4		plug @ 19'
9		22	23	21	23		28	5		plug @ 21'
10		21	22	20	22		26½	5½		plug @ 20'
11		21	22	20	22	1	27½	6½		plug @ 20'
12		27	23	22	23	4	26½	3½		Void difficult to measure. Lost plug at 23'. Had to pull up.

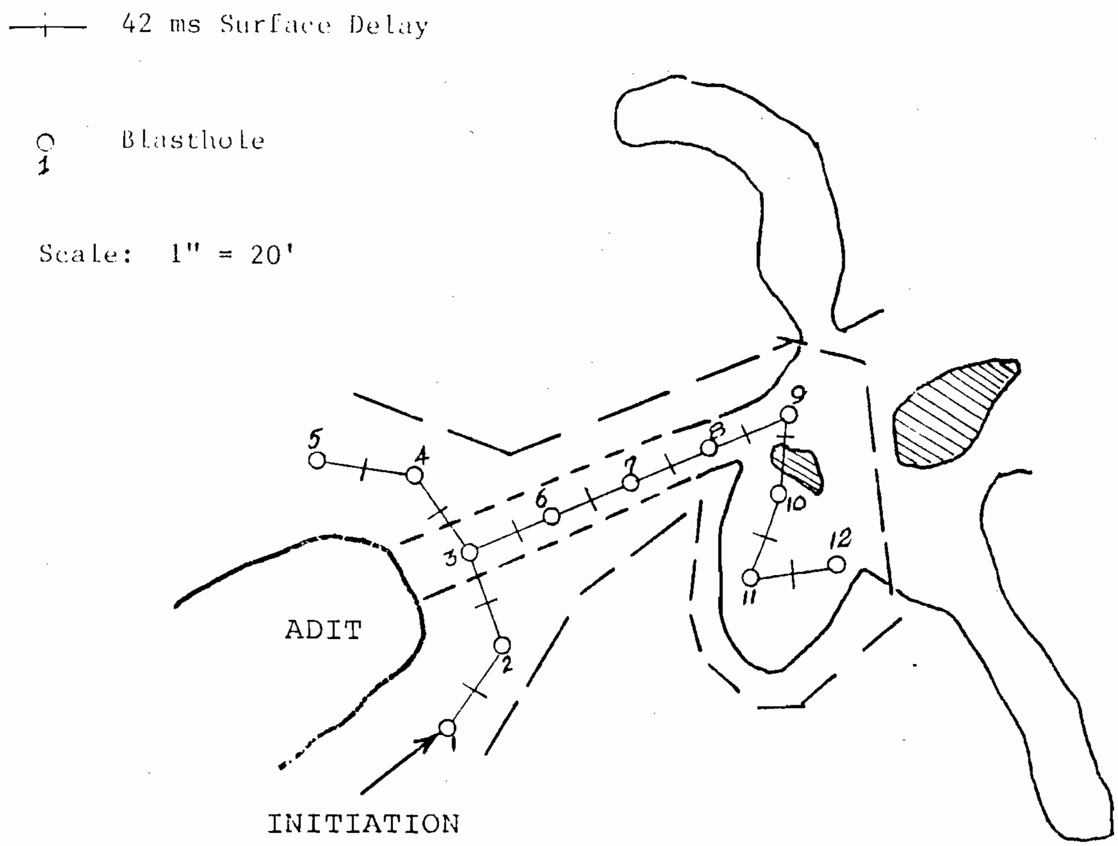

FIGURE B-1: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM FOR BLAST B-1

AML PROJECT - BLAST SUMMARY

BLAST #: B2

LOCATION: White

DATE 9-13-88

HOLE DIAM 6"

NO. HOLES 11 (13-23)

EXPLOSIVES COMSUMPTION

ANFO 11 BAGS = 550 LBS SLURRY 23 BAGS = 690 LBS

BOOSTERS 22 42 M/S DELAYS 11

DELAYS #4 11 #5 ___ #6 11 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 465' North of B2 PPV: .13 IN/SEC 119dB

CAMERA USED: Yes TARGET DELAY #: 1/4 FPS: 400 f stop: 5 focus: ∞

Lens Setting: 120MM AZ: 187° vert angle: -3° EV: 14 Location: 465' N of B2.

Elect E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
13		27	25½	25	26		29	3½		Fill to 22'
14		27	24	23	26		28	4		Fill to 22'
15		26	24	23	25		27	3		Fill to 22'
16		27	23½	23	26		25½	2		22' Plug actual Fill to 21'
17		27	25	24	26		29	4		Fill to 22'
18		26	23½	23	25		27½	4		Fill to 22'
19		25	22½	22	24		27½	5		Fill to 20'
20		NV	NV	–	28		28	NV		Fill to 24'
21			26	25	26		31	5		Fill to 22'
22	26			25	25		35	NV		9' Coal
23			24½	23	23		28	4½		Fill to 22'

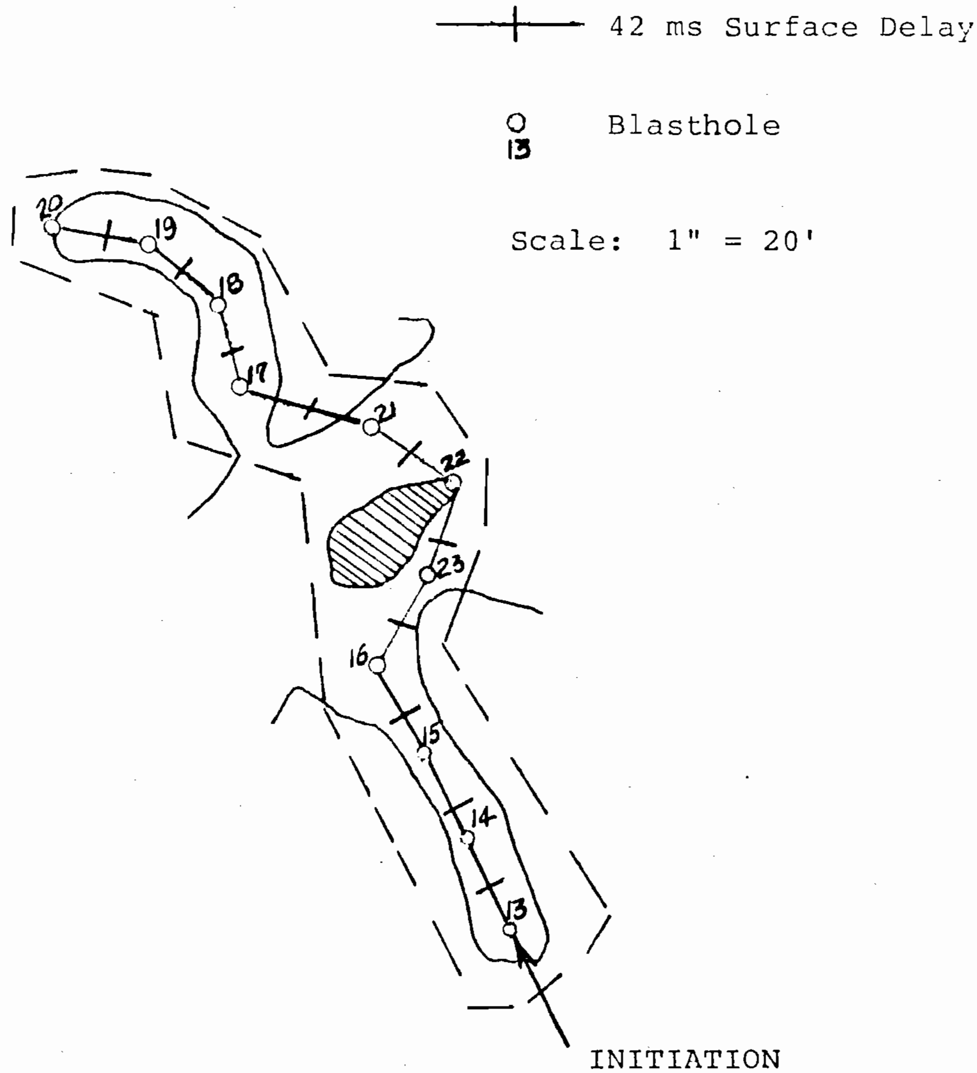

FIGURE B-2: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM FOR BLAST B-2

AML PROJECT - BLAST SUMMARY

BLAST #: B3

LOCATION: WHITE

DATE: 9-15-88

HOLE DIAM: 6"

NO. HOLES: 13 (25-39)

EXPLOSIVES CONSUMPTION

ANFO 9 BAGS = 450 LBS SLURRY 15 BAGS = 450 LBS

BOOSTERS 24 42 M/S DELAYS 13

DELAYS #4 13 #5 ___ #6 11 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 460' North of B3 PPV: 0.06 IN/SEC 12808

CAMERA USED: Yes TARGET DELAY #: ___ FPS: 400 f-stop: 5 focus: ∞

Location: 450' N. of B3 Az: 180° Vert angle: -1° EV: 14 Lens setting: 90 mm

Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH Back filled to	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
25			NV	28	27		28			
26			29	28	27		32	3		
27			31	30	29		33	2		
28		Lost		–	–		–	–		Lost as result of B2 blast
29		Lost		–	–		–	–		" " " " " "
30		29	27	26	25		28½	1½		Rubble and Loose Mat'l
31		28	27	26 (24)	23		28½	1½		" " " "
32		27	24(!)	23 (22)	21		27½	5½!		" " " "
33				30						" " " "
34		32	Filled	31 (26½)	25		32½	6		" " "
35		28	26	25	24		29½	3½		Plugged @ 20' - Couldn't get plug
36		?	22	21	20		23	1		Lost circulation @ 26'
37		15	14	13	12		19	5		
38		17	16½	15	14		20½	4		
39		21	21	20	19		25½	4½		

Hole 24 not loaded because it struck void in a newly discovered room

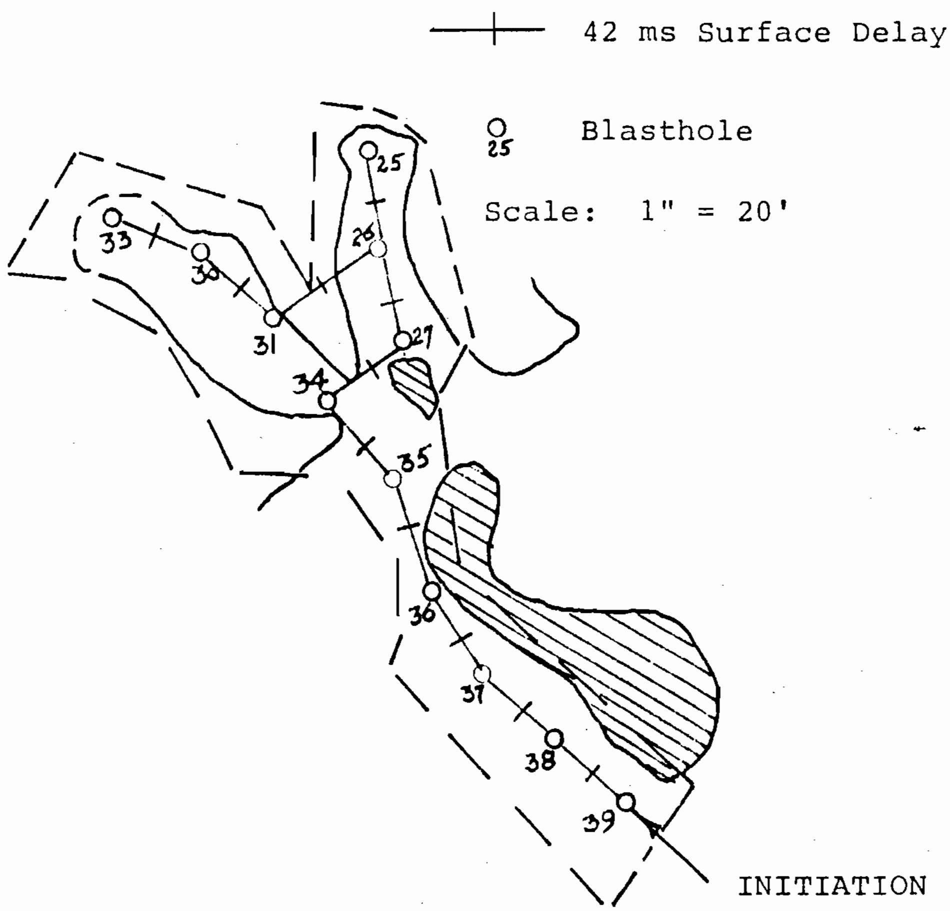

FIGURE B-3: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM FOR BLAST B-3

AML PROJECT - BLAST SUMMARY

BLAST #: 84

LOCATION: WHITE

DATE 9-16-88

HOLE DIAM 6"

NO. HOLES 5 (40- 44)

EXPLOSIVES CONSUMPTION

ANFO 0 BAGS = 0 LBS SLURRY 8 BAGS = 240 LBS

BOOSTERS 5 42 M/S DELAYS 5

DELAYS #4 5 #5 ___ #6 ___ #7 ___ #8 ___ #9 ___

SEISMOGRAPH: Yes LOCATION: 450' North of B4 PPV: .10 IN/SEC

CAMERA USED: Yes TARGET DELAY #: n/a FPS: 400 f-stop: 5 focus: ∞

Location: 450' North of B4. AZ: 170° vert angle: -1° lens setting: 95mm EV: 14

Elect E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH After fill added	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
40		27	26	25	22		29½	3½		Loose Mat'l 28/38. Plug @ 25. Fill
41		27	24½	24	22		28	3½		Plug @ 24'. Fill to 22'
42		27	26	24	22		29½	3½		Plug @ 24'. Fill to 22'
43		24	22½	22	19		27	4½		
44		24	22	22	19		26	4		

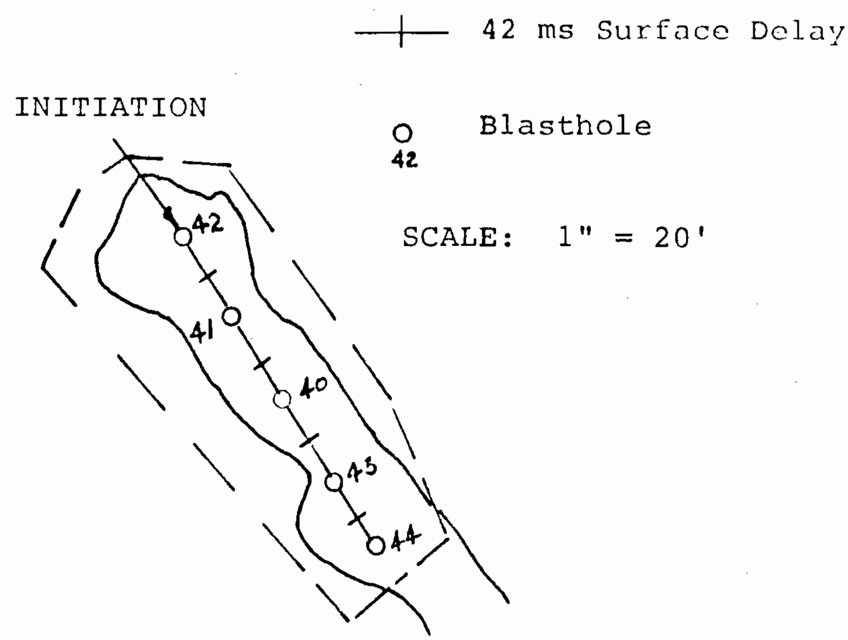

FIGURE B-4: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM
FOR BLAST B-4

AML PROJECT - BLAST SUMMARY

BLAST #: B5

LOCATION: WHITE

DATE 9/17/88

HOLE DIAM 6"

NO. HOLES 3 (45-47)

EXPLOSIVES CONSUMPTION

ANFO 2 BAGS = 100 LBS SLURRY 7 BAGS = 210 LBS

BOOSTERS 6 42 M/S DELAYS 3

DELAYS #4 3 #5 ___ #6 3 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 270 N.E. of B5 PPV: .64 IN/SEC

CAMERA USED: NO. TARGET DELAY #: ____ FPS: ____

Elect. E.B. Cap: 1

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after fill	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
45	31½	32½	31½	31	28	1	35	3½		Plug @ 31'. Fill to 28'
46	32	32½	32	31	28	.6	35.6	3		Loose Material at 38'
47	31	33½	33½	32	29	2.6	36.6	3		Plug @ 32' Fill to 29'

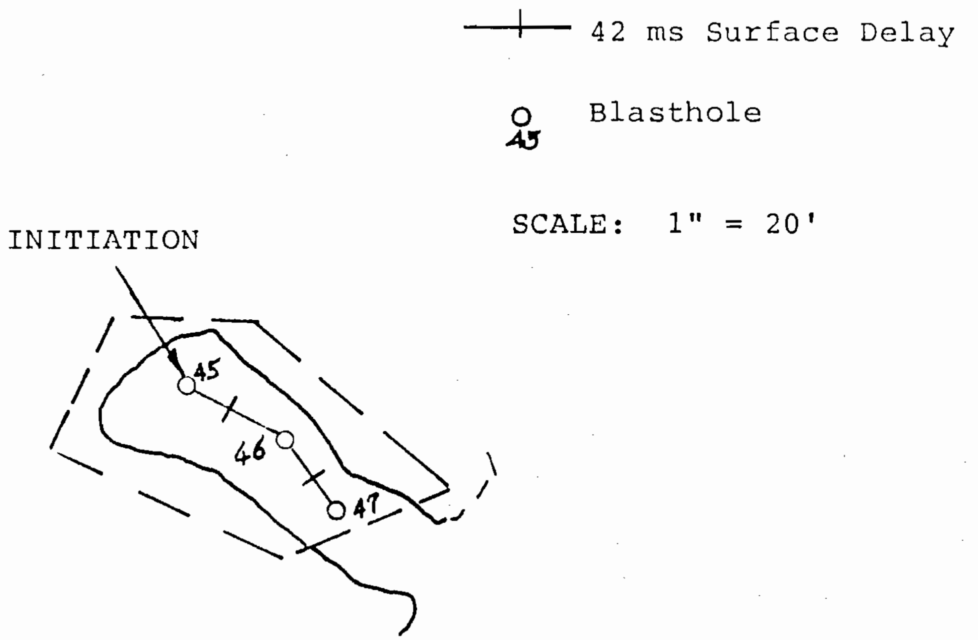

FIGURE B-5: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM
FOR BLAST B-5

A M L P R O J E C T - B L A S T   S U M M A R Y

BLAST #: B6

LOCATION: WHITE

DATE 9/19/88

HOLE DIAM 6"

NO. HOLES 46
(48 - 93)

EXPLOSIVES CONSUMPTION

ANFO 29 BAGS = 1450 LBS   SLURRY 68 BAGS = 2040 LBS

BOOSTERS 88   42 M/S DELAYS 42

DELAYS #4 46 #5 ___ #6 42 #7 ___ #8 ___ #9 ___

SEISMOGRAPH: YES LOCATION: 270' NW of B6   PPV: .68 IN/SEC

CAMERA USED: YES TARGET DELAY #: n/a FPS: 400 f-stop: 4 focus: 60
LOCATION: 270' NW. AZI 200° VERT Angle -2° Lens Setting 40mm EV:13 1/2

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after fill added	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
48		32		31	27		31			
49		17		16	12		20	3		
50		17		16	12		19	2		
51		10		9	8 1/2		12	3		
52	30				31	3	33			
53	31				35	6	37			
54		NV			26		28			Loose material
55	30				34	6	36			
56	32				36	6	38			Loose material
57	33	NV			37	6	39			
58	32				37	7	39			
59	32	36		35	31	5	37	1		
60	33	34		33	29		34	1(?)		
61	29				29		32			Lost Circ. 34'
62	29 1/2 (?)	NV			19		21			
63	31 1/2	32		31	27		34	2		
64	33	34		33	29		36	2		
65		24		23	19		25	1		
66	31	36		35	31		38	2		
67	32 1/2	NV			37	6 1/2	39			
68	33 1/2	36		35	31	2 1/2	36	?		
69	31	NV			36	7	38			
70	32	33		32	28		33			
? 71	32 1/2	35		33	29	1 1/2	35			
72		NV			30		32			
? 73	32	33		31	27		32			
74	31	NV			34	5	36			
75		22		21	17		27	5		
76	22 1/2	33		31	28		33			
77	30				35	7	37			
78	31 1/2	32		31	27	2 1/2	34	2		
79	32	NV			37	7	39	[illegible]		
80		30 1/2		29	25		33	2 1/2		
81	31	31 1/2		30	26		34	2 1/2		
82	32				35		37			Lost Circulation @ 34'
83		29		28	24		31	2		
84	30 1/2	32		31	27		35	3		

AML PROJECT - BLAST SUMMARY

BLAST #: 86 Sheet#2

LOCATION: WHITE

DATE 9/19/88

HOLE DIAM 6"

NO. HOLES 46
(48-93)

EXPLOSIVES CONSUMPTION

ANFO ____ BAGS = ______ LBS SLURRY ____ BAGS = _______ LBS

BOOSTERS _______ 42 M/S DELAYS _____

DELAYS #4 ___ #5 ___ #6 ___ #7 ___ #8 ___ #9 ____

SEISMOGRAPH: ____ LOCATION: ______________ PPV: ______ IN/SEC

CAMERA USED: _____ TARGET DELAY #: _____ FPS: _____

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after fill added	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
85		32		31	27		33	1		
86		29		28	24		31	2		
87		27		26	22		30	3		
88		NV			21		21			
89		31		30	26					
90		15		14	12					
91		29		28	24					
92		26	22	28	16					
93		NV		35	32		35			

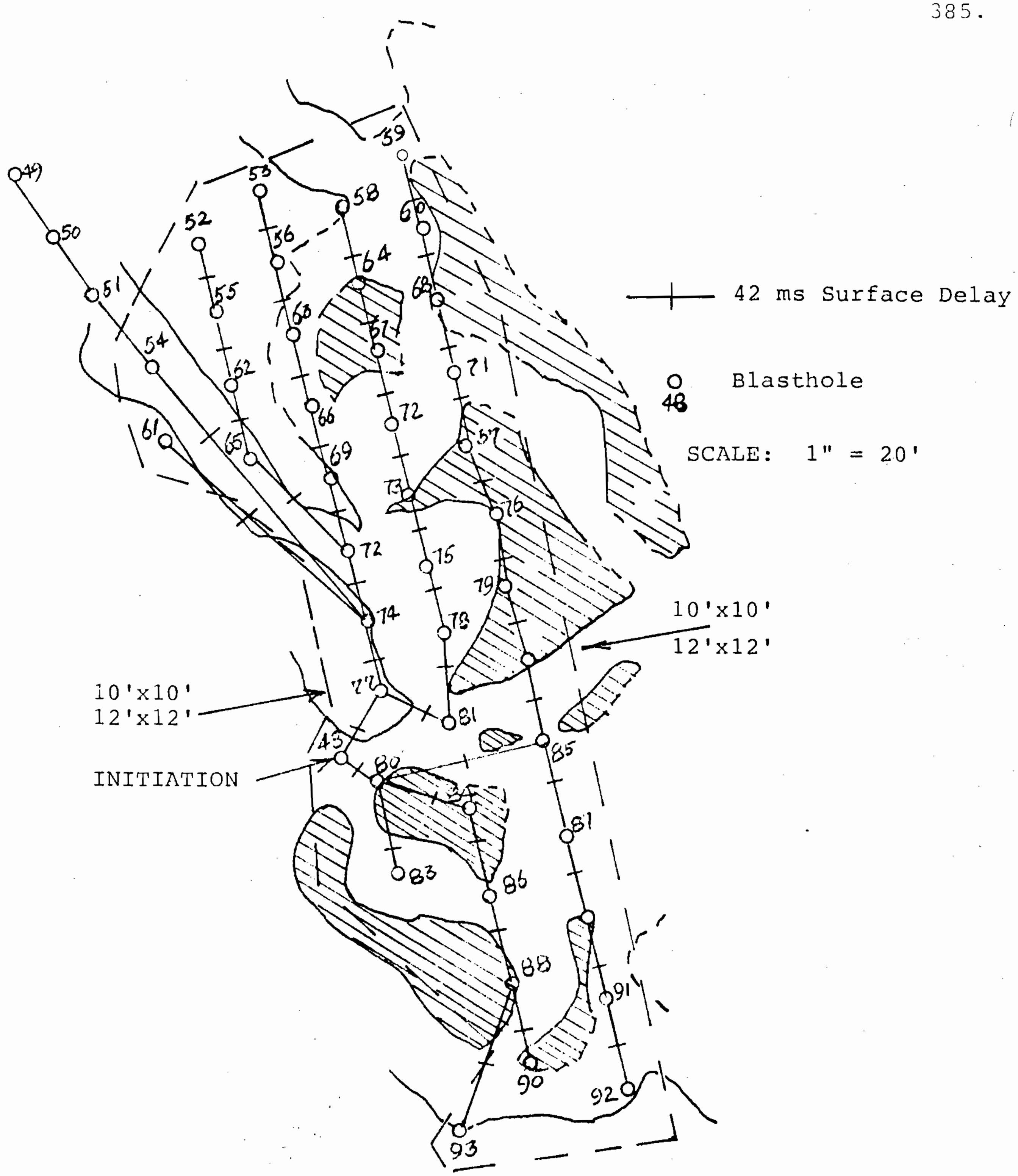

FIGURE B-6: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM
FOR BLAST B-6

AML PROJECT - BLAST SUMMARY

BLAST #: 87

LOCATION: WHITE

DATE 9/22/88

HOLE DIAM 6"

NO. HOLES 20 (94-113)

EXPLOSIVES CONSUMPTION

ANFO 19 BAGS = 950 LBS SLURRY 29 BAGS = 870 LBS

BOOSTERS 58 42 M/S DELAYS 6

DELAYS #4 19 #5 ___ #6 19 #7 ___ #8 20 #9 ___

SEISMOGRAPH: YES LOCATION: 270' NW OF 87 PPV: 1.04 IN/SEC

CAMERA USED: YES TARGET DELAY #: 4/6 FPS: 400 f-stop: 5.6 focus 40

LOCATION: 270' NW AZ: 200° Vert Angle: -1° Lens Setting: Zoom EV: 14½

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after fill added	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
94	36				35	2	38			8' water
95	34				33	5	39			6.2' water
96	34				38	4	38			2.6' water
97					39		39			4' water
98					37		37			3' water
99					35		38			5' water
100	33				37	5½	39½			3.3' water
101	34	36		35	34	5	39½	3		3.3' water
102		31		29	27		35	4		loose mat'l
103		17		16	14		17	—		
104	34				34	4	38			8' water
105	33					5	39			10" water
106	31	35		34	30		36	1		
107	34				38½	5	39			3½' water
108	36				38	2	38			2'9" water
109	35				39		39			1' water
110		36		30	29		36	—		1' water
111	34				38½		39			3'8" water
112	34				39		39			3'5" water
113		34		31	29		35	1		½' water

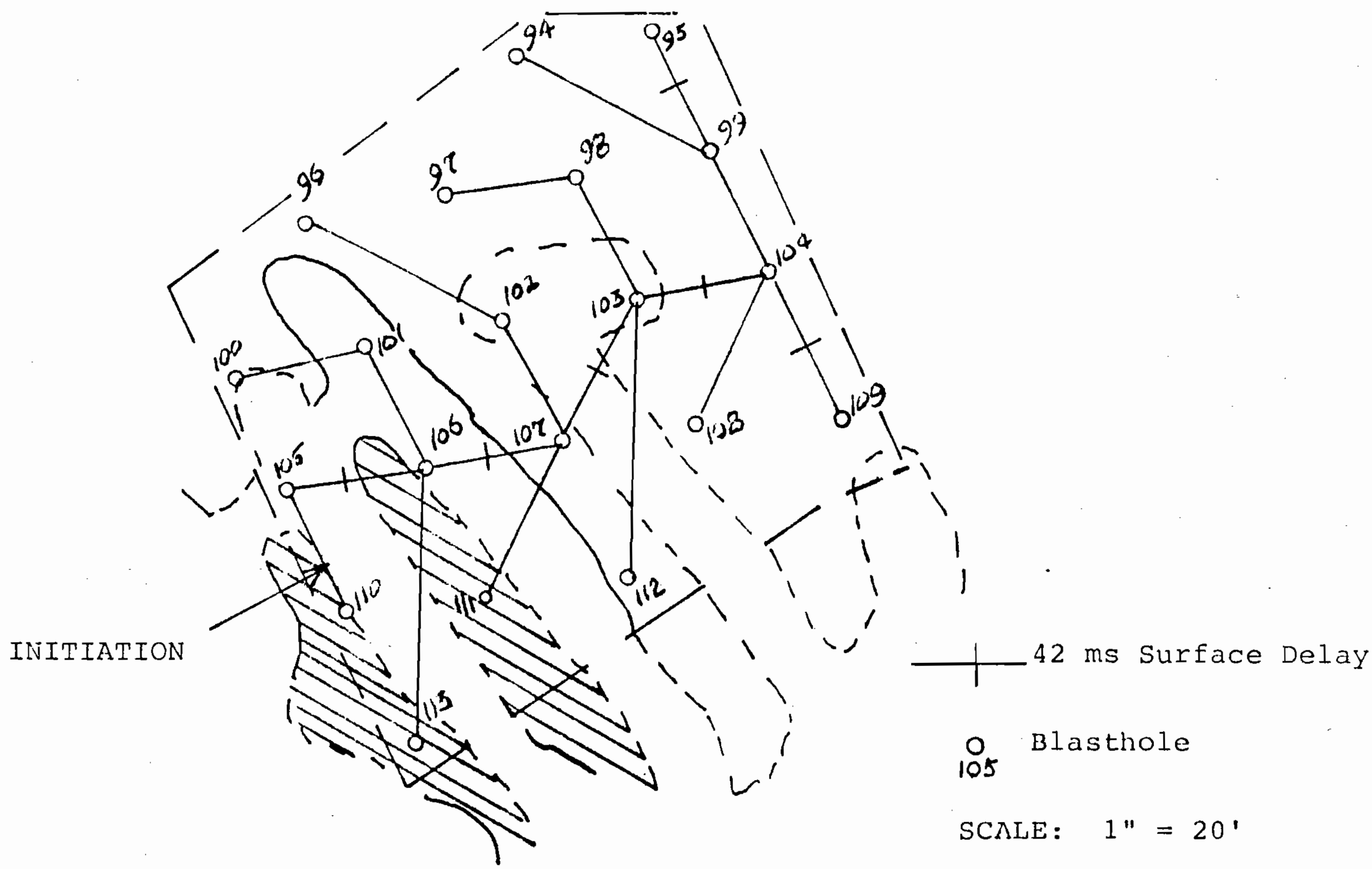

FIGURE B-7: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM FOR 15x15 FOOT PATTERN BLAST B-7

AML PROJECT - BLAST SUMMARY

BLAST #: B8

LOCATION: WHITE

DATE 9-24-88

HOLE DIAM 6"

NO. HOLES 65
(114-177)

EXPLOSIVES CONSUMPTION

ANFO 99 BAGS = 4950 LBS SLURRY 96 BAGS = 2880 LBS

BOOSTERS 190 42 M/S DELAYS 50

DELAYS #4 65 #5 64 #6 64 #7 __ #8 61 #9 __

SEISMOGRAPH: YES LOCATION: 270' NE OF B8 PPV: .78 IN/SEC

CAMERA USED: YES TARGET DELAY #: N/A FPS: 250 f-stop: 6.0 focus: ∞

Location: 270' NE AZ: 197° VERT ANGLE -1° lens Setting: 35mm EV: 13

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after filling	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
114	35	36			31		41	5		2' water
115	34½	NV			39		41			
116		36?			29		41	5		Filled in @ 11' Moved to Expl. 6.6 6' EAST
117		24		23	19½		24½	½		
118		NV			33		37			6' water
119	35'9"	NV					41			
120	34½	36			31		38	2		38'-41' loose Mat'l
121	34½	36			31		39	3		39'-41' " "
122	34	35			29		38	3		38'-41' " "
123	34'9"	36½			31		38	1½		
124	35	NV			39		39			
125	35	37½			32		41	3½		
126	33½	NV			39		41			
127	34'9"	NV			39		41			
128	35	36			31		41	5		
129	35½	37			32		41	4		
130	35	NV			40		39			
131	35	NV			28		39			
132	34	36			30	2	41	5		
133	34	NV			39	7	41			
134	35				39	6	41			
135	35	35½			30	½	41	5½		
136					20		41			27'-41' loose Mat'l
137	33	36			36	3	41	1		37'-41' " "
138		33			30		41	3		36'-41' " "
139		33			28		41	2		35'-41' " "
140	35	36			30	1	41	½		36½-41' " "
141	35	39			33	4	41	2		
142		27			22		41	3		30'-41' " "
143		19			14		21	2		
144		32			26		41	6		38'-41' loose Mat'l
145		31½			24½		41	5½		37'-41' " "
146	34				39	7	41			
147	35	37			32	2	41	1		38'-41' " "
148	35				39	6	41			
149		23			17½		41	2		25'-41' " "
150	35					6	41			

AML PROJECT - BLAST SUMMARY

BLAST #: B8-Sheet#2

LOCATION: White

DATE 9/24/88

HOLE DIAM 6"

NO. HOLES 65

(114-177)

EXPLOSIVES CONSUMPTION

ANFO ____ BAGS = ______ LBS SLURRY ____ BAGS = _______ LBS

BOOSTERS _______ 42 M/S DELAYS _____

DELAYS #4 ____ #5 ____ #6 ____ #7 ____ #8 ____ #9 ____

SEISMOGRAPH: ____ LOCATION: ________________ PPV: _____ IN/SEC

CAMERA USED: _____ TARGET DELAY #: _____ FPS: _____

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after filling	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
151	30½	31½				1	41	3½		35'-41' Loose Material
152	30				30	8	38			
153	32	34			28	2	41	4		38'-41' " "
154	32	32½			28	½	41	3½		36'-41' " "
155	32				35	7	41			39'-41' Clay
156		32			39		41	2		34'-41' Loose Mat'l
157	33				37	8	41			
158	35				31½	6	41			
159	33				39	8	41			
160		10			8		21	1		11'-21' Loose Mat'l
161	33				36	8	41			
162	33				29	8	41			
163	33				39	8	41			
164		32			28		41	3		35'-41' Loose Material
165	35					6	41			
166	33					8	41			
167		33			28		41	2		35'-41' " "
168	31				37½	7	41			38'-41' Clay
169	30					7	37			
170	31				35	7	41			38'-41' "
171	30				38	8	41			38'-41' "
172		27			35½		41	3		30'-41' Loose Mat'l.
173		29			23		41	9		38'-41' " "
174	36				38	5	41			
175	36				38	5	41			
176	36				39	5	41			
177	34				38	7	41			

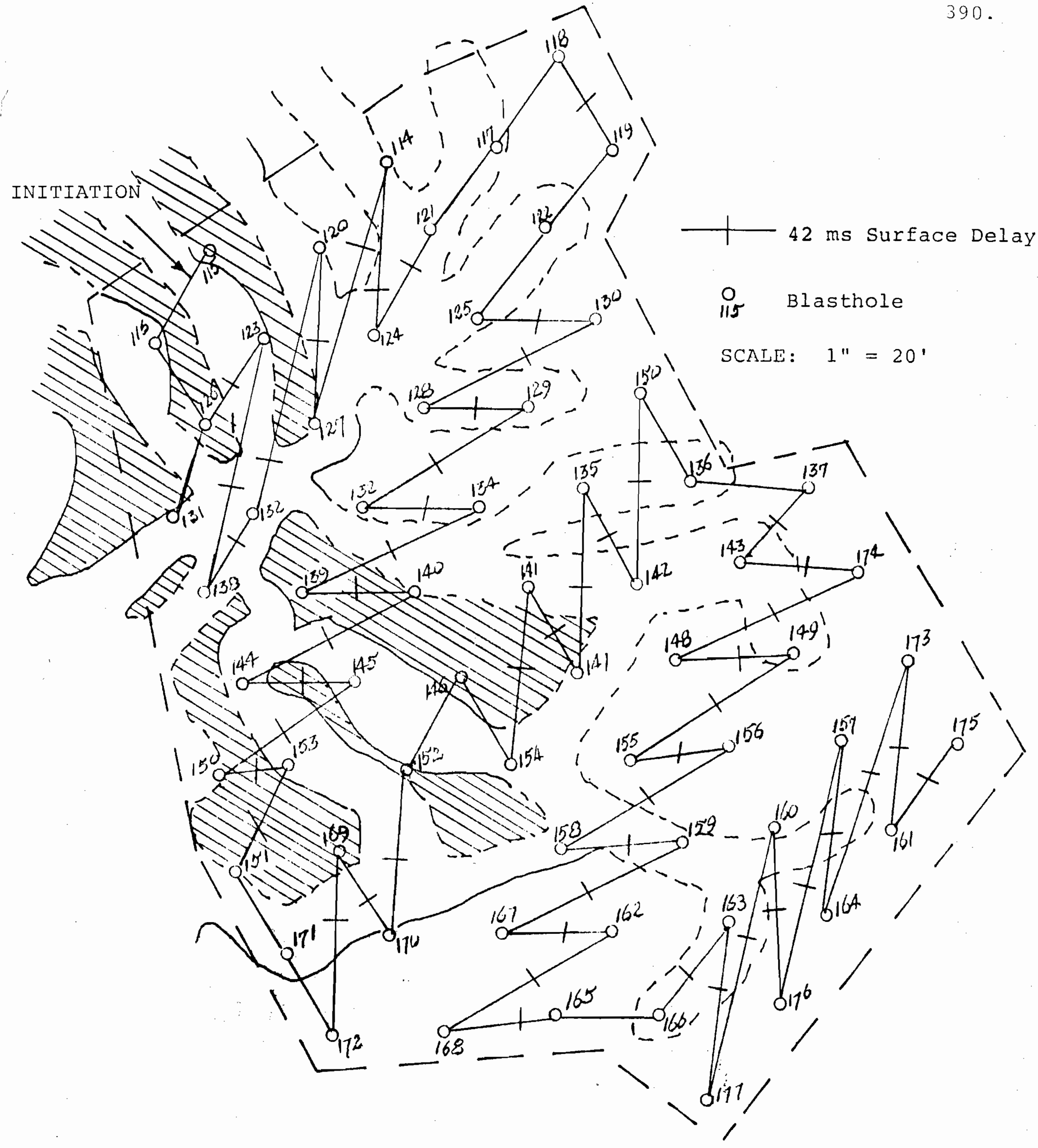

FIGURE B-8: BLASTHOLE LAYOUT AND TIE-IN DIAGRAM FOR 15x15 FOOT PATTERN BLAST B-8

AML PROJECT - BLAST SUMMARY

BLAST #: TEST SHOTS T1 & T2

LOCATION: WHITE

DATE 9/23/88

HOLE DIAM 6"

NO. HOLES 2

EXPLOSIVES CONSUMPTION

ANFO 2 BAGS = 100 LBS SLURRY 0 BAGS = 0 LBS

BOOSTERS 3 42 M/S DELAYS ____

DELAYS #4 ____ #5 ____ #6 ____ #7 2 #8 ____ #9 1

SEISMOGRAPH: No LOCATION: ____________ PPV: ____ IN/SEC

CAMERA USED: No TARGET DELAY #: ____ FPS: ____

#	TOC	VOID DRILLED	VOID MEAS	PLUG	DEPTH after fill added	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS
T1		36		34	31					
T2		21		17	16		24	3		