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BLASTING AS AN AML RECLAMATION METHOD
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BLASTING AS AN AML RECLAMATION METHOD

Volume I - Contract Report

Prepared for
Office of Surface Mining
Reclamation and Enforcement
Western Field Operations
Denver, Colorado

Contract HQ-51-CT-6-01570

Technical Project Officer
Eugene Hay

By
Bauer, Calder & Workman, Inc.
Washburn, North Dakota

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In 1986 a research contract was awarded by the Office of Surface Mining to Bauer, Calder and Workman, Inc., to investigate the technical and cost feasibility of blasting as an AML reclamation method. Blasting work was performed at a 10-acre site near Beulah, North Dakota.

The ojective of blasting for AML work is the collapse of underground development from surface. A second approach is the use of blasting to fill existing sinkholes. Successful blasting minimizes the chance of additional subsidence after reclamation, which enhances the long term safety and utility of the site.

In this type of blast application the only free face for material movement is vertically downwards into the abandoned development. Blast design was based on crater theory and the use of spherical charges. This report briefly describes the application of crater theory to AML blast design.

Field methodology and the results of the test blast program are discussed. Conclusions are made regarding the technical and cost feasibility of the method. The sensitivity of the cost of AML blasting to site and operating parameters is examined.

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Blasting work at 10-acre site near Beulah ND. Objec. of blast. for AML work is collapse of undergd develop. from surf. 2nd approach is use of blast. to fill exist. sinkholes. Success blast. minimizes chance of addit. subsid. after recla. which enhances lg term safety/util. of site. In this type of blast applic. only free face for material movement is vert. downwards into abandon. develop. Blast dsgn based on crater theory & use of spherical charges. Rpt briefly describes applic. of crater theory of AML blast dsgn. Fld method./results of test blast prog. discussed & conclus.

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BLASTING AS AN AML RECLAMATION METHOD

CONTRACT HQ-51-CT-6-01570 (J5160052)

VOLUME I

CONTRACT REPORT

SEPTEMBER 1987

PREPARED BY

BAUER, CALDER & WORKMAN, INC.

WASHBURN, NORTH DAKOTA

FOREWORD

The report has been prepared by Bauer, Calder & Workman, Inc. of 206 8th Street, Washburn, North Dakota in fulfillment of Office of Surface Mining Contract HQ-51-CT-6-01570 (formerly J5160052). The contract was administered by the Denver Regional Office of the Office of Surface Mining with Mr. Romer Gronbeck acting as Technical Project Officer. Mr. William Garrity was the Contract Specialist for the OSM. The input of these individuals has been appreciated by the authors.

The results of all studies performed under this contract are contained herein. The report includes work performed during the period of August 1986 to September 1987 and was submitted during September 1987.

Thanks are extended to the North Dakota Game and Fish Department for permission to use the Beulah Site. The North Dakota Public Service Commission AML staff are thanked for their assistance in finding information, and their interest in the project. Thanks are also in order to the Knife River Coal Mining Company for providing mine maps without which this research would have been much more difficult.

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1. INTRODUCTION

1.1. GENERAL DISCUSSION

The Coal Mine Reclamation Act (1977) provided monies to fund the reclamation of abandoned coal mine sites including both underground and surface mines. Prior to the mid-nineteen fifties the majority of United States coal mining was accomplished using underground methods. Therefore, hazards associated with old abandoned underground workings are widespread and often serious. Problems include public safety and environmental damage.

For several years, therefore, reclamation efforts have been directed at old underground mines (and surface mines) using a variety of different methods. For complete reclamation of a total area techniques have included:

- Remote backfill
- Daylighting
- Dynamic consolidation
- Blasting

For reclaiming individual hazards the following techniques have been used:

- Rock or earth fill
- Cement blockage
- Backsloping
- Fencing off
- Blasting

Blasting appears to have been the least used, perhaps due to less knowledge of how to design successful blasting rounds to collapse the workings and to fears of vibration damage to structures or public response to blast vibrations and airblast. The lack of technical data also leads to cost uncertainty which may discourage use of the blasting technique.

It was thought that blasting might well have a role in AML reclamation.[1] Good caving results could perhaps be obtained and costs might be competitive with other methods. The blasting technique believed to be most applicable to caving the underground voids was based on spherical crater blast design.

Certainly, blasting would not create problems when performed in areas not in immediate proximity to residential and other structures. On the other hand, blasting near to buildings is not uncommon in construction projects. Further, many quarries are situated near built-up areas and blasting with residential subdivisions less than five hundred feet from the quarry rim does occur. Therefore, blasting might well be used to mitigate problems nearer to structures and people than often thought.

One believed immediately however that blasting would not be applicable if sink holes started to open up within a residential area. Distances between structures and blasting area would be too small to allow effective results while avoiding damage. The structures themselves could be undermined and nearby blasting might cause caving under the buildings. Safety issues related to sealing the area before the blast and fly rock would be more difficult to deal with.

Another limitation to the use of blasting could be depth to the void. Greater depth would increase the drilling cost. Also room for material movement would be greatly restricted and it would be more difficult to determine the effectiveness of the result. More independently delayed explosive decks would be needed which could lead to greater probability of out-of -sequence firing and therefore poorer results.

Some AML related situations are emergencies. Such urgency results when public safety is greatly threatened. Blasting can be a good technique in these situations. Explosive suppliers and contract drillers are most often available nearby so mobilization is rapid. Provided information is available by which blast designs can be developed quickly, mitigating emergency situations by blasting can be quite effective. One purpose of the research reported in this text is to provide the necessary information.

In many cases effective and complete reclamation of the mined areas requires complete caving or filling of the voids. Once this is accomplished the area can be declared safe for the public and new subsidence will not occur in the future. Blasting is a method that can be used to completely cave the mined area. Furthermore, there is usually good visual evidence of the caving due to slumping of the surface into the void. This is in contrast to techniques such as remote backfill where it is more difficult to assess the results of the reclamation. The techniques and results of such "area" caving are reported in this research.

In other cases it is desired only to fill isolated open holes for emergencies, to reduce costs or in areas where only isolated subsidence has developed. The most common method of removing these post-mining features is to find a source of fill and backfill the holes.

An alternative is to drill blast holes around the caved area and blast material into the void. This may be especially useful when backfill material is hard to obtain. Also it was thought that the cost of reclaiming the individual sink hole by this method would be attractive. Costs might be quite good if a site required a combination of area caving and filling of isolated holes. The results of work on individual subsidence features is also discussed in this report.

Preliminary thought was that blasting could be beneficially applied in many instances. It was felt that except where one is in very close proximity to structures and to people blasting might provide benefits not always available using other methods. These benefits include:

1. Complete caving of the workings; certainty of reclamation success.

2. Rapid reaction time to emergencies.

3. Flexibility; the method can be used to mitigate both area and individual problems.

4. In some cases blasting would provide for total reclamation of the area without further requirement.

5. Topsoil might not need to be removed prior to reclaiming; an important point in unstable areas.

Thus under suitable conditions blasting might well be the method of choice.

1.2. PREVIOUS WORK

A literature search was completed to determine how blasting has been used in the past during AML reclamation projects. There is very little such literature.

A paper, given at the National Symposium and Workshop on Abandoned Mine Land Reclamation described one blasting project in North Dakota.[2] The author concluded that reclamation of the area had been generally achieved and that the result was cost competitive with other methods. It was the opinion of the author that a fifty-foot depth of cover was the maximum that could be blasted and that blasting would not be applicable if more than fifty per cent of the area had already collapsed.

A second paper, by J. L. Workman,[1] discussed principles of blast design likely to be pertinent in the AML blasting. This paper was not based on actual field blasting at abandoned mines but discussed, based on the author's experience in blasting, how such blasts might be designed. The view of this author was that a form of spherical crater blasting design would provide the best result and that blasting could be technically and cost competitive with other methods.

A third paper, by Bruce K. Stover,[3] discussed the use of blasting to seal individual mine openings such as adits and open shafts. Area reclamation was not discussed. Stover concluded that blasting was effective for the closure of openings. In remote areas he was of the opinion that this technique might be

the only viable alternative.

Blasting as an AML reclamation method has also been discussed by K. W. Royse[4] with reference to the Urlacher site in North Dakota. The paper describes a limited test on five to seven acres. Royse concludes that blasting is a viable method of reclamation when the circumstances are suitable. He claims that the cost is acceptable and suggests that costs in the range of $6,000 - $8,000 per acre are possible.

These papers represented the bulk of the reported literature about blasting as an AML reclamation technique. All concurred that it was a viable method. Only Workman discussed the use of crater blast design to effect caving of the overlying strata. Others had used primarily column charges to effect the caving of the overburden.

All authors concentrated on blasting in the underlying voids rather than blasting the pillars to precipitate general caving of the mined area. There appears to be a consensus among the authors, and others with whom these procedures have been discussed, that blasting the pillars would be difficult. Reasons are that the centers of the pillars would be hard to locate and prior caving of the roof might mean the pillar would have little room to displace and fragment upon detonation of the blast. The pillars could well be wet resulting in explosive loading problems. Therefore, work has concentrated on blasting to the voids left by mining.

The conclusion from surveying the literature was that the current project, would in many ways, break new ground. During the research detailed data would be collected so that techniques for abandoned underground works would be much better characterized. Data collection and analysis did not appear to be a major concern of previous work which was performed more on a construction basis.

1.3. PURPOSE AND METHODOLOGY

The primary purpose of this research project is to examine the possibility of using blasting as an AML reclamation method. The greatest interest is in the use of the method for area reclamation, but a few isolated subsidence features were also attempted to better characterize the possibilities of reclaiming these individual voids. This was considered important because of the many open adits, shafts, raises and sink holes that exist in the United States today.[5] The research consisted of a major field test program followed by extensive technical and cost analysis of the accumulated data. The field blasting procedures were carefully controlled. Each blast was designed and then laid out in the field with each hole marked by a wooden stake. Extensive data about the blast was recorded including blast vibration records and high-speed films.

The field work was done on a sufficient scale to insure that the conclusions drawn would be accurate. The program aimed at classifying several issues. These included:

1. Could crater blasting techniques be used successfully in AML reclamation?

2. Could overall reclamation be effected by blasting alone?

3. How much exploratory drilling would be involved?

4. Could drilling equipment work safely where subsidence had previously occurred?

5. How should such blasts be designed?

6. What blast patterns should be used?

7. What explosives would be best for use?

8. What millisecond delay timing would be appropriate?

9. What field procedures should be followed?

10. What caving success rate could be achieved using blasting to collapse the old works?

11. How much swell would result in the blasted material?

12. Would blast vibration be a problem?

13. Would blasting cause premature caving in adjacent undermined ground? What implication would this have for blasting in proximity to structures and facilities that were undermined?

14. What are the conditions of technical feasibility?

15. Would blasting be a cost effective approach?

All of these questions are addressed in this report.

Once the field work was complete the data was subjected to extensive analysis. The intent was to document what happened during the field testing and to project the results for future work.

The testing was completed at one site in order to perform a sufficiently extensive test under uniform conditions. Thus it was possible to insure that all likely phenomenon were experienced and accounted for. That is, one wanted to be certain that the test was sufficiently extensive to allow any problems to be

evidenced. One blast that successfully caved one room on a twenty-acre site would not, in our view, indicate that the entire site could be successfully blasted.

The primary disadvantage of a single site is that all work was completed in overburden exhibiting the same strata sequence. Therefore, the utility of the approach in other materials would have to be projected. However, it was believed that with the data obtained and a knowledge of cratering criteria in other materials such projections could be made.

One intent of the study has been to provide guidelines for others who may want to use blasting to reclaim abandoned underground mines. Further, it was desired to present cost data so that others could estimate the cost of reclamation by blasting and compare this with the cost of alternative methods. Both technical and cost information is presented in the report. Sensitivity analyses illustrating the effect of changed site and unit cost parameters on the overall cost of reclamation by blasting are included. A cost model has been developed that allows one to estimate the cost of blasting a site.

1.4 OTHER AML RECLAMATION TECHNIQUES IN NORTH DAKOTA

Several other methods of reclaiming abandoned mine lands were mentioned earlier in this chapter. A few of these have been used in North Dakota.

The primary method used for complete reclamation has been remote backfill. This method was used at several pits near Beulah, North Dakota for example where the stabilizing of areas in proximity to housing and roads was the objective.

Assuming that backfill fills all the voids then reclamation success will be complete. However, it is understandably difficult to assess whether this has occurred. The method is one of the few available that allows reclamation in very close proximity to dwellings. This approach does have substantial cost which can be in excess of $50,000 per acre.

For the reclamation of individual hazards the common method in North Dakota has been the use of earth fill to close the holes. This method has attractive cost being about $1,000 per acre on average. Of course the per acre cost will be highly dependent on the number of hazards per acre. One site was reclaimed for about $1.00 per cubic yard of fill.

The utility of this method will depend on the availability of fill and access to the property. The primary drawback is that filling existing holes does not guarantee that other holes will not open up later. Thus complete reclamation of the site may require several years of monitoring and fill work. Furthermore, closing existing holes may lead to a false sense of safety

about the site as a whole. Thus this method should generally be used when there is reason to believe that all caving has occurred. This could be estimated from mine maps and site surveys where this information is available, or might be assumed when extensive caving is in evidence.

These are the primary methods that have been used in the state. Daylighting has been tested but not used extensively. One small project in North Dakota suggested a cost of $18,500 per acre for this method including topsoil removal, seeding and monitoring. One purpose of this report then has been to determine how blasting compares to other methods in cost and in regard to its realm of application.

2. BLASTING THEORY APPLIED TO AML RECLAMATION

2.1. GENERAL POINTS

The principles by which the blasts in the test series were designed are discussed in this chapter. These principles may also be used to design blasts for reclaiming abandoned underground mines in the future.

When performing area reclamation, by which all the undermined land is reclaimed, a prime consideration is that the freedom for the blasted material to move is quite restricted. Close to the ground surface the overburden can be displaced by uplifting to the free surface. Elsewhere, when the hole detonates, material movement must be almost exclusively toward the mined void below. To maximize the caving, therefore, requires that the overburden be directed toward the void in a timed sequence that provides the lower material the opportunity to displace before overburden from higher up the hole moves downward. Otherwise the blast will jam up leading to potential bridging of the overburden and failure to fully cave the workings.

For this reason placing a full column explosives load in each blast hole and calculating a blast pattern based upon a suitable powder factor is unlikely to provide optimum results. Column loaded holes work well in bench blasting in open pits and quarries where there is an unrestricted free face in front of the first row of holes. The front row is free to displace to this face and subsequent millisecond delayed rows follow the first row out into the pit. When caving old workings however, there is no lateral free face alongside the holes and the motion must, in fact, be directed vertically downward. The only exception to this is at the upper surface where the material may be heaved upward and then allowed to fall back.

Using column loaded charges then is likely to be wasteful of explosives since much of the energy will be oriented in the wrong direction. Most of the explosive cannot effectively contribute to the caving of the overburden. It is unlikely that blast patterns can be as large as predicted. Powder factors will be greater than expected.

An alternate, and more suitable, approach would be one which directs the ground motion downward, minimizing the restriction to movement and minimizing the powder consumption for cost and blast vibration reasons. The method that seems most likely to meet these criteria is one that involves spherical cratering charges. The approach has similarity to the vertical crater retreat (VCR) method used in some underground metal mines.[6,7] However, it differs from VCR blasting. In VCR blasting individual cratering charges are placed in the bottom of the hole and detonated to blast off the next layer of

ore. After the ore is mined out the hole is reused for another cratering charge. AML work requires a series of independently placed cratering charges in the hole, timed by millisecond delays to detonate the charges from bottom to top with sufficient delay between each deck to allow good displacement of the material toward the void below. The principles of spherical cratering used in this project are described in the next section.

Also of interest is the design of blasts to close individual sinkholes. Since such holes, especially those that are newly formed, may be open to the void below they can be a significant hazard to public safety.

Closure blasts would not be designed in the same way as those described above. In this case a lateral free face exists. A column charge would be used designed to move overburden horizontally into the hole. Design considerations include the distance of the holes back from the rim of the sinkhole, volume to be blasted to fill the hole and blasted area (swell accounted for) and explosive loading used. The principles forming the basis of these designs are also discussed below.

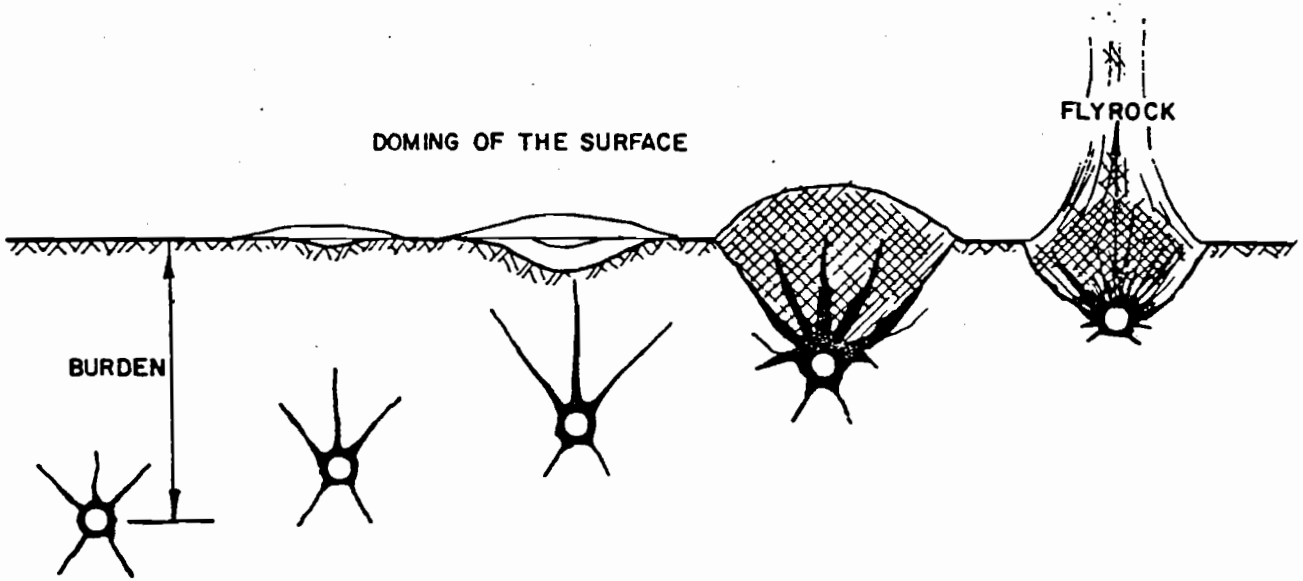
2.2. SPHERICAL CRATER BLAST DESIGN

The blast designs reported in this study are largely based on spherical cratering theory. The basis of this phenomenon has been well explained by Livingston[8,9]. Other authors have presented applications of the theory to blasting [10,11,12].

When a charge, that approximates a sphere, is placed in the ground and detonated it will act on the surrounding material in a manner consistent with the depth of burial of the charge and the charge weight. There will be a depth, in a given material, at which there is no detachment of the material surrounding the charge and no doming or slabbing of the surface. Only localized crushing and cracking will occur. This is called the critical depth, usually denoted as N. For depths of burial less than N increasingly greater fragmentation is achieved and the crater volume increases until an optimum is reached. Further decrease in the depth of burial then leads to increased fragmentation and flyrock, a reduced crater size and changes in the crater shape. Figure 2-1 illustrates this progression. Figure 2-2 defines the terminology associated with the production of a single crater.

For brittle rocks the optimum depth ratio, D_o, is often in the range of 0.45 to 0.55. The optimum depth ratio is the optimum depth of burial divided by the critical depth. For soft rocks, D_o will often be 0.8 to 1.0 [10].

This means that a greater depth of burial is possible for charges in soft, plastic materials. However the fall off from



COMPLETLY CONTAINED, ONLY FAILURE IS PULVERISATION NEAR THE CHARGE AND RADIAL · TENSILE FAILURE RUNNING OUT FROM IT.

START OF SURFACE FAILURE. BURDEN NOT BROKEN. SOME DOMING OF THE SURFACE.

SURFACE AND SUBSURFACE FAILURE ALMOST MEET THERE WILL BE A SHELF OF UNBROKEN ROCK BETWEEN THE TWO. DOMING OR SURFACE BULGING.

FULL CRATER, BURDEN COMPLETLY BROKEN OUT. SURFACE AND SUBSURFACE FAILURES RUN THROUGH TO THE SURFACE.

FULL CRATER, LOWER VOLUME THAN OPTIMUM FINE FRAGMENTATION. NOISE, FLYROCK, BOWL SHAPED CRATER.

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

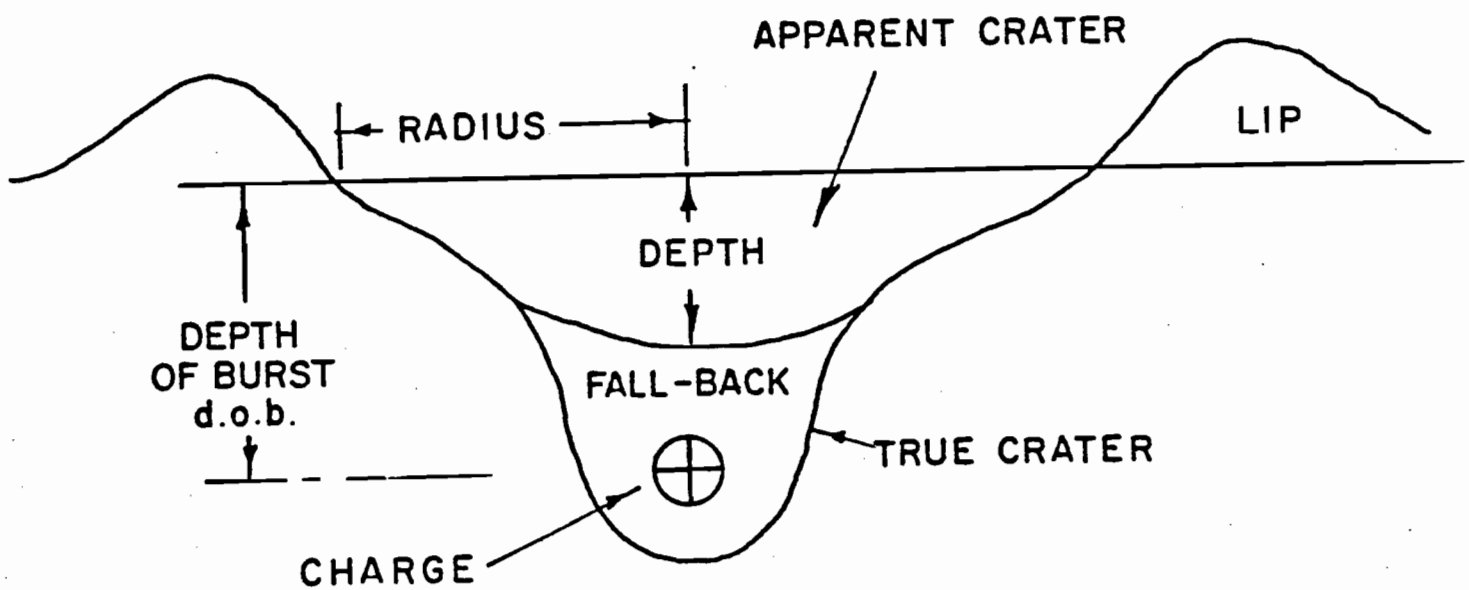


FIGURE 2-2: CRATER TERMINOLOGY.

optimum to critical depth is considerably more sharp than for brittle rocks like granite and magnetite. Care must be taken in locating the charges to avoid a fall off in cratering performance.

Livingston related the depth of burial to the charge weight by the equation:

$$N = EW_e^{1/3}$$

where

N = the critical depth in feet
W_e = the critical weight in pounds
E = the strain energy factor

Rocks breaking in brittle fracture would have strain energy factors greater than 3.5. Values of E less than 3.5 would be indicative of soft, plastic rocks with increasing shear failure.

Given the foregoing relationships one can plot scaled crater dimensions against scaled depth of burial (SDOB). The scaled depth of burial is the depth to the center of the charge divided by the cube root of the charge weight. The other dimensions may also be scaled to the cube root of the weight and plotted against the scaled depth of burial. Figure 2-3 is a plot of scaled crater dimensions versus scaled depth of burial and illustrates the relationships. The exception to this is the volume which scales to the first power of charge weight. These scaling factors arise because the weight of charge that can be placed in a spherical geometry is proportional to the cube of the radius. Therefore, linear dimensions are related to the cube root of the weight while the volume, also related to the cube of the radius, is scaled to the weight directly.

As figure 2-3 shows there is an optimum depth of burial for the charge, in the given conditions, which produces the maximum scaled crater depth. The same is true for other crater dimensions. Therefore by employing the correct depth of burial the cratering action can be maximized.

In practice one uses the optimum scaled depth of burial to determine the correct depth of burial. Knowing the explosive type and the hole diameter the weight of explosive is known. Then the depth can be determined:

$$\frac{d}{W^{1/3}} = S$$

$$d = SW^{1/3}$$

where d = depth of burial, feet
W = explosive weight, pounds
S = optimum scaled depth of burial

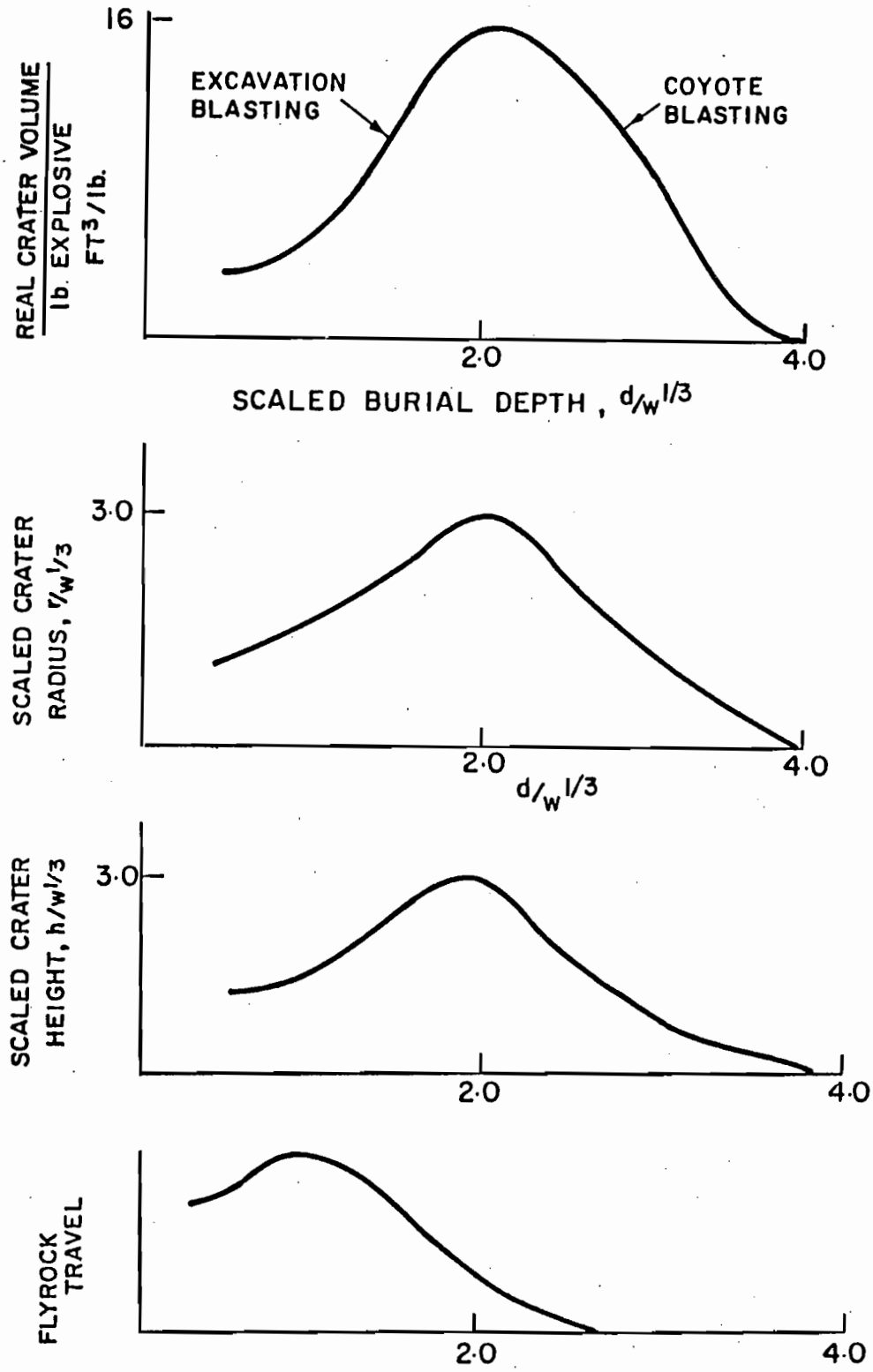


FIGURE 2-3: CRATERING RESULTS IN BRITTLE ROCK WITH SPHERICAL CHARGES.

The holes, in which the explosive is placed, are typically drilled by rotary drilling equipment. Therefore, the actual geometry is cylindrical, not spherical. However, if the charge is short it approximates a spherical geometry and cube root cratering relationships apply. The longest cylindrical charge that will approximate a spherical cratering charge is one that has a length to diameter ratio not greater than 8 to 1. The weight of explosive contained in a charge having a length of eight charge diameters can be calculated. Therefore, knowing the charge weight and the scaled distance the depth of burial can be computed.

It was considered that a series of such cratering charges would be required in each hole in order to blast the overburden into the old works successfully. The depth of burial of the individual charges would be from the top of the previous charge to the center of the charge being placed. The bottom charge would be scaled from the void to the center of the explosive column.

The concept was to provide the overburden the opportunity to move down into the void created by mining. Therefore, the first charge to detonate is the one at the bottom of the hole. It is important that cratering charges further up the hole not be detonated simultaneously with the bottom deck but should be delayed to maximize the freedom the material has to displace toward the open void below. A matter of design then, discussed later in this chapter, is the selection of suitable millisecond delay times.

2.3. DESIGN PRINCIPLES FOR CLOSING INDIVIDUAL SINKHOLES

Closure of an individual subsidence feature requires a different blast design than that described above. This procedure requires the casting of swelled surrounding overburden into the void created by subsidence. Therefore, displacement of material in the lateral direction is of greatest concern. A typical individual sinkhole is shown in figure 2-4

To generate such horizontal movement of the overburden requires that the charge be placed sufficiently close to the void so that the material can be completely detached from the surrounding mass and have sufficient velocity to displace into the sinkhole.

Cratering concepts can be used, in a general way, to describe the progress. However, because the cratering effect will be off the side of long cylindrical charges the cratering effect is scaled to the square root of the charge weight per foot of charge length.

There will be an optimum depth of burial which will create the largest crater depth. Reduced depth of burial will result

FIG. 2-4: EXAMPLE OF A TYPICAL INDIVIDUAL SINKHOLE IN ABANDONED MINE LAND

in greater mass velocity off the face and more displacement. Therefore, it will be important to locate the blastholes at the optimum depth of burial or a little less. In weak, plastic materials the fall off from optimum to containment will again be rapid so one wants to avoid the area of the curve to the left of optimum. Figure 2-5 is a typical curve for square root scaling of a cylindrical charge.

Cratering data generally shows considerable scatter. Therefore, as shown in the figure, the data plots as a band rather than a discrete curve. For this reason one would not design these blasts solely on cratering data, but should also consider the powder factors involved.

Individual sinkholes have different profiles. Newly developed features often have steep sides and may be open to the mined-out area below. Older sinkholes tend to have more gently sloping sides and are usually closed at the bottom. Of the two the newer holes are easier to blast in. These have steeper sides and the blastholes can more easily be located in an optimum position behind the rim of the hole.

Older sinkholes, because of the gentler slopes typical of these features, are harder to blast. The distances from the toe of the blasthole to the free surface become long for vertical drill holes. Therefore, it is more difficult to get the material moving. Figure 2-6 illustrates the problem. Where it is desired to blast in such features it may be well to consider inclined holes.

It is equally important not to place the holes too close to the subsidence feature. If this is done the result will be bursting of the detonation gases through the face with reduced effectiveness, potential for uncontrolled flyrock and increased airblast and noise potential. Not only does reduced blasting performance result but the blast may be disturbing to persons residing nearby. To avoid human response and potential damage one should avoid having too little burden on the charges.

2.4. EXPLOSIVE SELECTION FACTORS

There are several factors which contribute to the successful use of an explosive for a given application. The factors influencing selection are listed below:

1. Energy output; bulk and weight strengths

2. Critical diameter and the factors that influence it

3. Loading density

4. Ease of loading

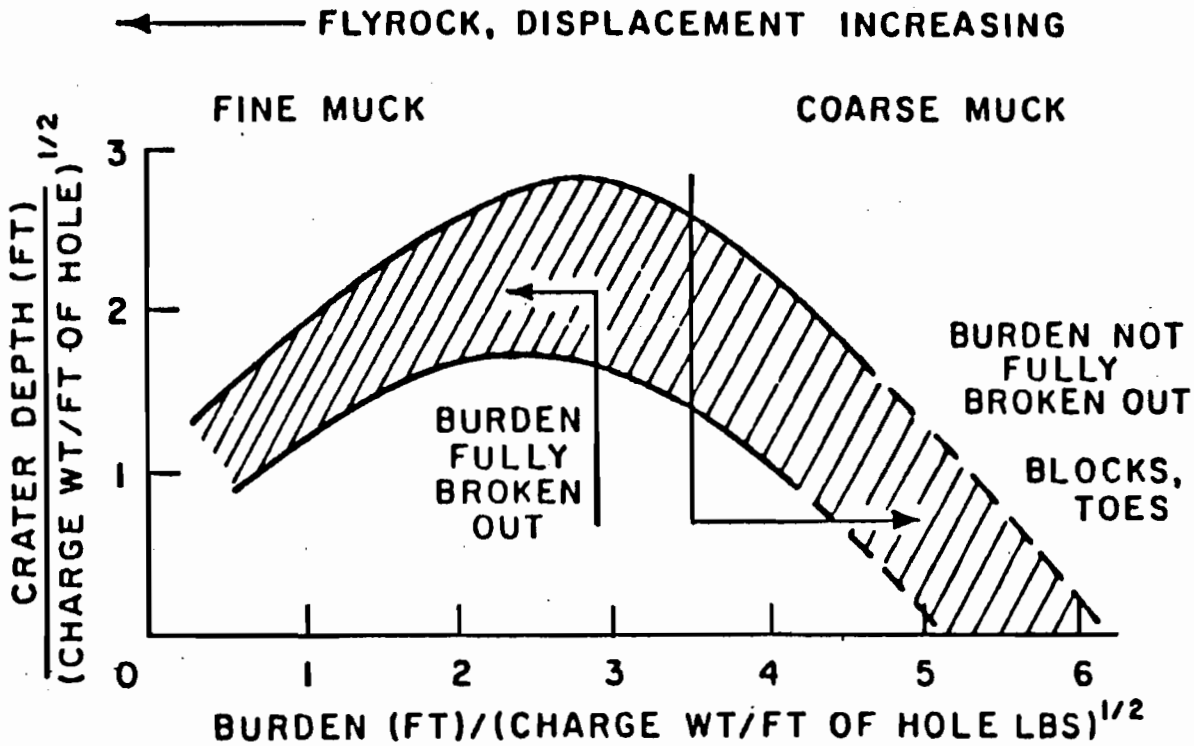


FIGURE 2-5: THE CHANGE IN CRATER DEPTH PRODUCED BY DIFFERENT BURDENS ON CYLINDRICAL CHANGES FIRED IN QUARTZITE

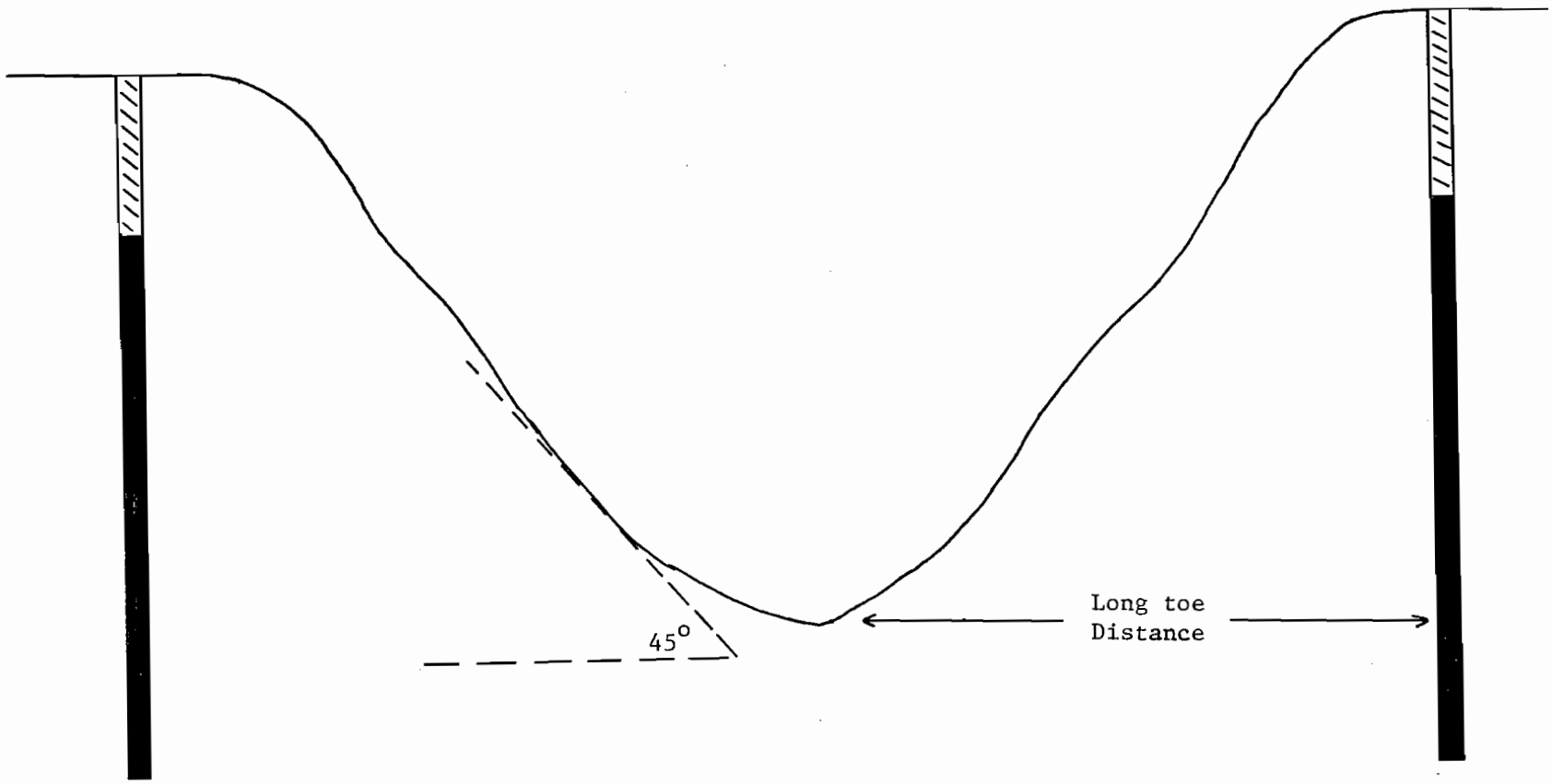


FIG. 2-6: DIAGRAMMATIC ILLUSTRATION OF THE LONG TOE DISTANCE TYPICALLY EXPERIENCED WHEN BLASTING OLDER SINKHOLES

5. Water resistance properties

6. Shelf life (sleep time)

7. Coupling properties

8. Gap sensitivity and cross-propagation

9. Reliability and quality control

The energy output of the explosive must be sufficient to break the rock experienced on the given pattern. The use of more energetic, more costly explosives does not generally pay off in soft formations where the drill cost is modest. The use of ammonium nitrate - fuel oil (ANFO) will therefore be expected to be effective in blasting down the material types usually found above old coal mine workings. Furthermore this product enjoys a cost per pound lower than that of almost all other typical blasting agents.

The critical diameter of the explosive is very important. If the charge diameter falls below the critical diameter then the explosive will fail to shoot consistently. Reclamation blasting will often be conducted using drill hole diameters that are smaller than those typically seen in open pit operations. Therefore, the critical diameter of explosives available for use should be carefully checked.

For ANFO in confined charges a critical diameter of less than three inches is usually observed. Therefore, under proper conditions of use, this product should shoot well in the hole diameters most likely to be used in AML work. However, if the product is attacked by water the critical diameter is likely to increase leading to the probability of unstable detonation and failure. Therefore, ANFO should be used in dry holes or with polyethylene dry liners in wet holes.

Other products experience changes in critical diameter related to changes in operating environment. These include variations in temperature, with the critical diameter increasing with decreasing temperature. Air bubble sensitized slurries and emulsions are most sensitive to temperature. Small diameter products are the most affected.

Air bubble sensitized products are also sensitive to the hydrostatic load applied to the explosive by the column of explosive and the stemming above. Increased overpressure leads to increased critical diameter. For some explosives the change can be quite pronounced. This is illustrated in figure 2-7. Reduction of the initial density of the explosive becomes necessary to sensitize the product. This is accomplished by using gassing agents to introduce gas bubbles into the mix. Sensitivity to overpressure can be of particular concern for holes in the five- to eight-inch range and it is important to

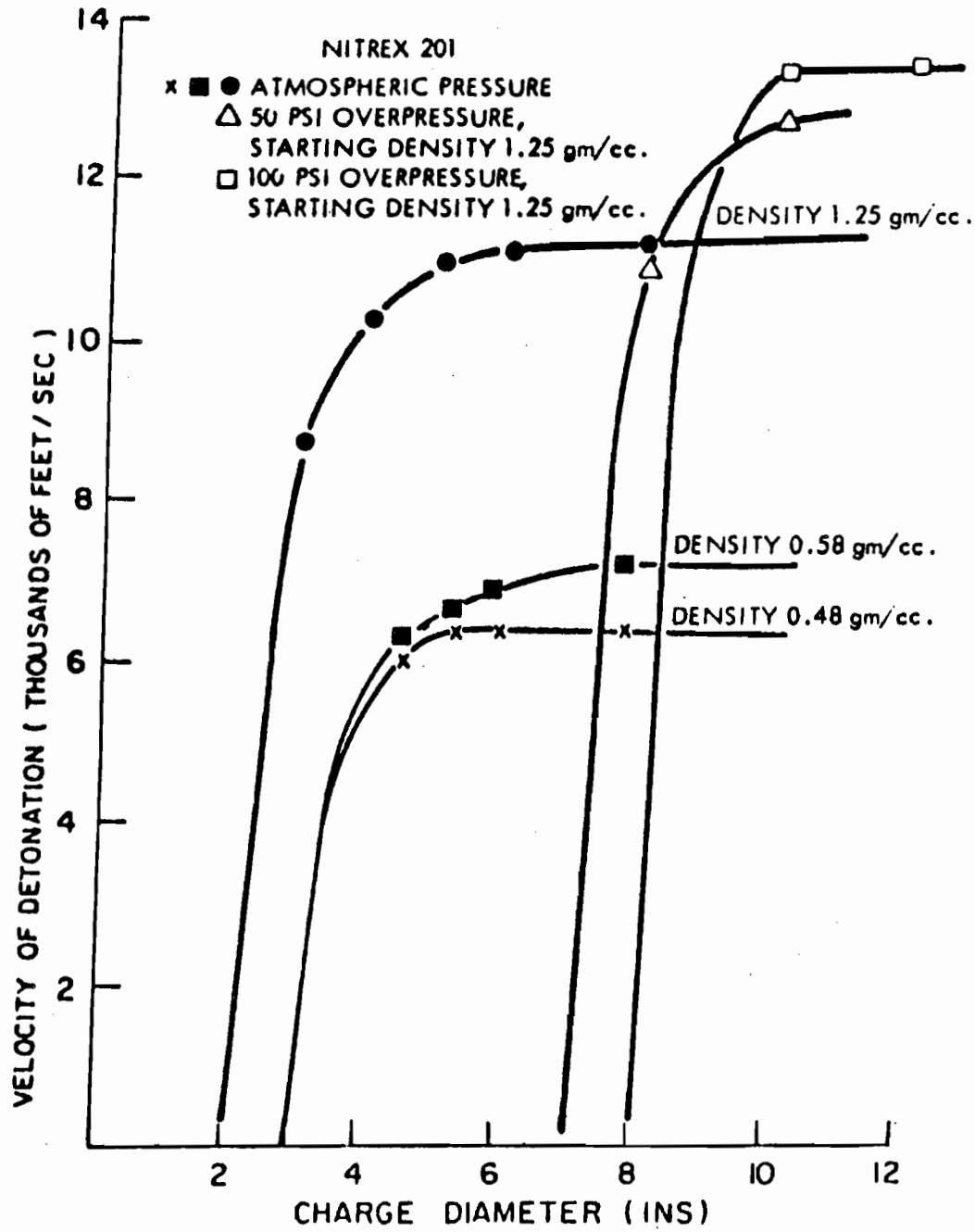


FIGURE 2-7: EFFECT OF PRESSURE AND DENSITY ON THE VELOCITY DIAMETER CURVE OF NITREX 201, @ 1% AL NCN SLURRY.

confirm that the product will shoot in the diameter of hole being used.

The density of the explosive should be considered. The density relates to the bulk strength (i.e. the energy output on a volume basis) that the explosive produces. In the case of cratering shots the length of the charge is limited by the need to approximate a spherical geometry. Therefore, a denser explosive will allow a greater charge weight to be placed within the required geometry. This can lead to greater depths of burial and crater dimensions. However, since the dimensions scale to the cube root of the weight benefits will need to be weighed against cost.

The density of the product should be consistent, whether bulk loaded or bagged. Variations in density will change the loading density and make proper control of the loading process difficult.

If bagged explosives are used the loading density may be reduced due to the undersized bags being dropped down the hole. This is undesirable. Cutting the bags open and dropping the explosive down the hole may prevent this problem. However, this research has shown that some products develop significantly reduced density when released from the bag.

For efficient operation the explosive loading procedures should be rapid and straight forward. Blasting agents, loaded in bulk are perhaps the best example of easily loaded products. Bulk loading may not be suitable in AML work however unless the project is quite large in scope. Bagged blasting agents may also be used. Bagged ANFO is easy to work with as the bags can be opened and the explosive simply poured down the hole.

Slurry and emulsion products can be loaded by dropping the bags down the blasthole. This procedure is permissable due to the insensitive nature of the explosive types. However, the bag must be of a lesser diameter than the hole. Therefore, decoupling occurs which is detrimental as it reduces borehole pressure. The bags may be cut open and the explosive chopped up and dropped down the hole. This eliminates the decoupling problem but is more time consuming to load.

Some explosives are more water resistant than others. For example, ANFO has virtually no water resistance. Figure 2-8 shows what happens as increasing amounts of water are added to ANFO. There is a rapid fall off in performance and when the water percentage reaches twelve percent the product fails to detonate. Exposing ANFO to water for as little as four hours leads to considerably reduced performance.

Slurries and emulsions on the other hand have good water resistance. These products can be loaded in wet holes with good results. Heavy ANFO is also water resistant when the emulsion

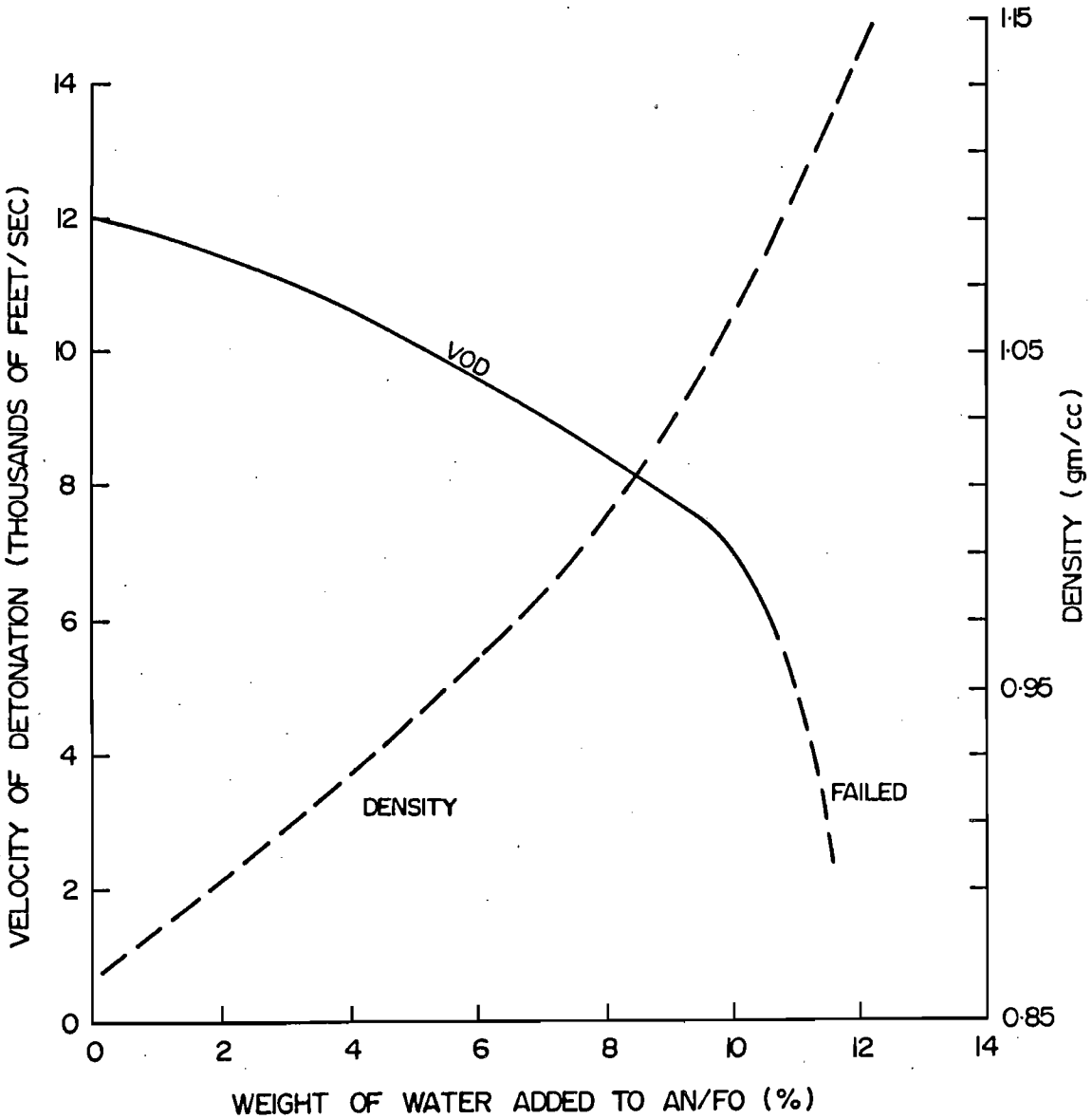


FIGURE 2-8: THE EFFECT OF WATER ADDITION TO AN/FO FIRED WITHIN 5 HOURS OF MIXING IN 4" DIAMETER PLASTIC PIPES USING 1# PROCORE PRIMERS.

percentage reaches about fifty percent. However, heavy ANFO should not be loaded through water as the prill is stripped away from the emulsion and water inclusion in the column can result. Inefficient detonation or failures may occur.

Therefore, in wet holes, waterproof products may be used to avoid performance problems. An alternative is to use polyethylene dry liners and ANFO. These plastic liners are placed in the hole and the ANFO is loaded inside the liner. Thus it is protected from the water and the explosive performs as expected. For larger projects it may well be that the cost of dry liners is less than the cost of more expensive water proof explosives. One should be aware though that it is difficult to place dry liners in holes of less than six-inch diameter.

Shelf life of the product should be considered. Some basic air bubble sensitized fuel oil slurries have shelf lives as low as forty-eight hours. Bulk loaded these must be shot in this time frame to avoid deterioration. Packaged products may have somewhat longer times but also need to be watched.

ANFO has a shelf life that is indefinite provided it is not affected by water. ANFO will also have a long storage life unless caking occurs due to prill breakdown in handling or temperature cycling. Caking is most likely to be a problem in hot, humid climates.

Slurries that are not solely dependent on air bubbles for sensitivity have longer sleep times. These would include products that include aluminum or TNT for sensitization. Emulsions that are sensitized using micro-balloons also have good shelf life. However, microballoons are more costly.

In AML work loading the holes and shooting the blast will usually take place the same day. Therefore, shelf life will not be critical. However, field procedures should be anticipated and, if explosives may be left in the hole for longer periods, shelf lives should be determined and a suitable selection made.

Coupling is the term which defines the degree to which the explosive fully fills the cross sectional area of the hole. For a one hundred percent coupled product the explosive column diameter is equal to the hole diameter. Coupling factors of less than one hundred percent describe the degree to which the explosive column diameter is less than the hole diameter.

Bulk loaded blasting agents have a one hundred percent couplng factor. So do bagged products if the explosive is removed from the bag during loading. However, ANFO in waterproof bags or slurry and emulsion bags that are simply dropped down the hole will lead to reduced coupling factors. This means reduced borehole pressures and reduced work on the overburden surrounding the charge.

In abandoned mine land work where cratering charges are being placed decoupling is especially undesirable. One has to keep the charge length less than or equal to eight times the charge diameter. At the same time as much explosive as possible should be loaded. Decoupling reduces the weight of explosives in the charge and drill patterns and depth of burial will have to account for this. Therefore, high coupling factors should be maintained. ANFO is ideal for this purpose as it is free poured and completely fills the hole cross section.

When blasting down the overburden above old mine workings a series of decked, independently delayed cratering charges are expected to be most effective. Therefore, one must try to avoid gap sensitivity and cross propagation problems.

The gap sensitivity of an explosive defines its tendency to propagate across a gap in the explosive column. In field practice gaps may consist of dirt such as deck stemming or that which is inadvertently knocked into the hole during loading. In some cases water gaps may occur in wet holes, especially if explosive is loaded by dropping it through the water. Air gaps can occur if the explosive is loaded in bags and a bag hangs up in the hole.

In general blasting agents do not propagate across the large gaps seen for some dynamites. The gap sensitivity of these products is less. However, it has been found that even with blasting agents propagation between decks is possible. The distance across which sympathetic detonation can occur increases with increasing hole diameter.

For example, in 6 1/2-inch diameter holes the sand gap distances for a five percent probability of cross propagation in slurry and emulsion are 12 feet and 6.2 feet respectively.[13]

In 12 1/4 inch holes the ninety-five percent probability of detonation occurs for a sand gap distance of 3.7 feet for slurry and 0.9 feet for emulsions. It should be noted that these results are estimates based on the results of studies on small diameter explosives. However, the extrapolated data fits observed data from high-speed camera studies quite well.

Emulsions appear to be less likely to detonate across a sand gap then slurries and water gels. Therefore these products might be more attractive for use in decked holes unless the deck stemming distances are long. ANFO may also be less likely to cause sympathetic detonations.

The reliability of the product and the associated quality control procedures are important. ANFO because of the simplicity of the system is one of the most reliable of explosives. Slurries, emulsion and heavy ANFO are more complicated to manufacture and are more prone to variations in density and composition. However, with proper quality control

by the manufacturer these products can also be reliably produced.

For ANFO the primary concerns are to insure the ammonium nitrate to fuel oil ratio is 94/6 and that the prills are sufficiently porous to retain the fuel oil. If aluminum is added to the ANFO then percent aluminum should also be checked.

The quality control tests for other blasting agents are more complex. It should be insured that the manufacturer is carrying out adequate quality control tests and the results of these should be reported.

In general, ANFO is most likely to suit all the needs of AML work. However, in wet ground other products may be more suitable. If the overburden is such that it is necessary to maximize the weight of explosive in each cratering charge for adequate breakage on a reasonable pattern then higher density explosives could be considered.

In many cases in AML work bagged powder will be used because the project is not large enough to warrant bulk loading. Also, if the explosive decks are small it may be hard to bulk load them accurately. ANFO is the easiest explosive to use in bags because it can be freely poured from the bag. Slurries and emulsions are more difficult to remove from the bag as they are not very pourable in most cases. Dropping them in the hole, bag and all, is not recommended in this type of work due to the decoupling that results.

For these reasons ANFO is expected to be the explosive of choice for most AML work. However, each project should be evaluated and the best products chosen for the specific site.

2.5. BLASTHOLE DIAMETER

An important consideration is the diameter of hole to be drilled. There are three competing design considerations involved. One is that larger diameter holes will reduce the number of decks required in the hole and increase the pattern size. A second is that if the hole is too large the pattern will not fit well with the mining geometry. Often in old mines the entries and rooms were small in dimension. Large diameter holes will result in the breakage of unnecessary material in such a case. Third, if the holes are large then the explosive weight per deck will be greater and the levels of ground vibration generated by the blast will increase. It will have to be insured that these vibrations will not exceed lawful levels and, in many cases, that they will not generate undue citizen complaints.

Therefore, the following criteria will most likely control the hole size used:

1. The dimensions of the workings being blasted.

2. The depth of the overburden. In deeper overburden larger hole diameters will be better because this helps to control the number of deck charges needed. A maximum of five such charges is recommended.

3. Availability of drilling equipment. Often truck mounted drills will be used and this limits hole size to about ten inches. Large track mounted rotary drills are expensive and not often available for this type of work.

4. Material type. Rock will require drilling with standard rotary bits whereas unconsolidated overburden may be drilled with large auger bits. In tar sands for example augers of up to 36-inch diameter have been used to drill holes for cratering charges.

5. The largest hole diameter should be used that is consistent with the dimensions of the workings.

6. The critical diameter of the chosen explosive. The hole size should exceed the critical diameter of the explosive by two inches or more.

7. Proximity to structures and people. When reclaiming old works near to people and residences or other buildings the vibration levels must be kept below allowable limits.

Previous development work has indicated that holes in the 4- to 6-inch diameter range might be most suitable. For efficient cratering, however, holes of less than 6 inches are likely to be less effective. For depths of overburden approaching sixty feet 8-inch diameter holes will be more suitable. Greater depths would require even larger hole diameters to limit the number of explosive decks per hole.

In general, then, one wants to use the largest hole diameter that is consistent with the dimensions of the workings, depth of overburden and acceptable blast vibration levels. Doing so will provide the most effective cratering possible for the given conditions.

2.6. MILLISECOND DELAY TIMING

Introducing millisecond delays into the blasts serves two very important purposes. One, it provides time for the material around the explosive charge to move before the next charge detonates, thereby providing room for the succeeding material to displace. Second, the use of millisecond delays allows the blast vibration levels to be minimized.

In this case accommodating the first need requires both surface and down-the-hole delays. Using a suitable combination of the two each deck can be made to detonate separately. Thus the freedom of the overburden to displace toward the mined area can be maximized.

For the method used in the test work the bottom deck was required to detonate first. The overburden just above the coal would therefore be propelled into the void. Then the subsequent decks would detonate in ascending order. In this way the material was allowed the greatest opportunity to displace.

A primary question is the length of the millisecond delays. The duration of the delay must be sufficient to allow for displacement of the material but not so long as to cause disruption of other undetonated decks or cutoffs. Soft plastic materials can generally accept longer delay times between detonations because of the the energy absorbing nature of these materials. Brittle rocks require shorter delay times to avoid cutoffs.

It is accepted that, in bench blasting, the minimum delay time for displacement of the burden is one millisecond per foot of effective burden. For fully adequate displacement it is often found that two to two and one-half milliseconds per foot of burden is optimum.

Although the concept employed in this research is not bench blasting as such it is reasonable to assume that similar relationships will hold true. In fact, given the restricted room for movement and the nature of the overburden at the test site (weak, plastic material) even longer times could be appropriate. Therefore, it was considered that millisecond delays greater than two times the depth of burial should be used. Later in the report the delays used are described and the milliseconds of delay per foot of burial are determined.

Surface delay elements can create considerable noise when detonated. Some units however are less noisy than others. For example, the use of NONEL noiseless trunkline delays eliminates the noise associated with the explosion of detonating cord on the surface. The noise from the delay element itself can be minimized by burying the delay with drill hole cuttings. Other systems such as the Hercudet tube initiation system and the Dupont Detaline system can help to reduce noise from the surface tie-in. Electric blasting is another method to reduce noise.

It should be understood also that currently manufactured millisecond delays do not always detonate at the time specified on the tag. Rather there is a scatter around the nominal firing time that is statistical in nature. In some cases overlap can occur between successive delays causing out-of-rotation firing. This result is particularly undesirable when blasting in old workings. An upper deck firing in advance of a lower deck will

have nowhere to go. Bridging of the overburden may occur which will prevent subsequent caving. Therefore, for the down-the -hole delays, there may be an advantage to using every other delay period provided that will not lead to cutoffs or cross propagation.

2.7. BLAST VIBRATION

Whenever blasting is performed ground vibration is a concern. Excessive levels of ground vibration can damage structures. Much lower levels result in citizen concern and complaint even though no damage results.

Also of concern is the noise and airblast that results from blasting operations. High levels of airblast can cause window breakage and if airblast levels are very high extensive window, door and structural damage can result.

There is another reason for desiring to minimize ground vibrations when performing AML blasting operations. Heavy vibration at nearby rooms and entries may cause caving and sinkholes to appear which will make subsequent blasting of these areas more difficult. It is also possible that such vibration, while not resulting in failure of material above adjacent openings, may cause the overburden to be in a weakened state making the operation of equipment in these areas more hazardous.

Therefore, even if damage and human response are not much of a problem reducing the ground vibrations to the lowest level will be advantageous to operation within the blasting area. For all of the above reasons the principles of blast design and timing that properly reduce blast vibration and air blast should be followed.

In the case of blasting above old workings some help may be expected from the nature of the site. The extensive voids beneath the blast area will help to attenuate the vibrations that might otherwise be expected.

The usual way of measuring ground vibration from blasting is by recording the particle velocity versus time in each of three mutually orthogonal modes; the longitudinal, transverse and vertical directions. This is accomplished using a blasting seismograph of which there are several models on the market.

For some years the criteria for the onset of damage (plaster cracking) was taken to occur at a peak particle velocity of 2.0 inches per second. This was based on studies by the U.S. Bureau of Mines that included their data, the data of Edwards and Swedish experience. This criteria prevailed regardless of the frequency. The peak particle velocity is the maximum velocity measured in any one of the three modes. It was found that there was no advantage to using the peak resultant

particle velocity

More recently investigators have concluded that frequency also has a role in the effects experienced from ground vibrations. Thus the structural response to a given peak particle velocity is greater when the frequency of the vibration is closer to the natural frequency of the structure. Since blasts in surface mines tend to have lower frequencies, in the range of natural response frequencies, the role of frequency has become of greater concern for mining operations.

There is, therefore, less certainty now as to the velocity at which the onset of damage will occur. However, by designing blasts to minimize the peak particle velocity the likelihood of damage is reduced. Certainly, for peak particle velocities less than 0.5 inches per second the possibility of damage is virtually nil.

The prediction of peak particle velocity is usually accomplished from a plot of particle velocity versus scaled distance. A typical plot is shown in figure 2-9. For this work it is usual to scale the particle velocity to the square root of the weight, reflecting the cylindrical geometry in which the charge is placed. However, in cases such as that in this research, where spherical cratering charges are used it will likely be more pertinent to scale the distance from the shot to the measuring point according to cube root scaling.

It has been found that the particle velocity is affected by the millisecond delay tie-in. The less the explosive weight per delay period the less the vibration. Therefore, graphs like that in figure 2-9 scale the distance not to the total charge weight but to the weight per delay period.

There is not a discrete particle velocity measured for each scaled distance. Rather, there is much scatter in the data. This scatter is due to factors such as changes in geological conditions, difference in wave types, varying charge geometry, scatter in the millisecond delay times, different explosive types and errors in reporting and measurement. Therefore, it is not possible to predict exactly the peak particle velocity that will be associated with a given scaled distance.

The usual approach is to determine the trend of the data by regression techniques and then draw a parallel line above all the measured data. This becomes the upper limit line and is shown in the figure. Designing the blast to have a scaled distance that yields a given peak particle velocity on the upper limit line insures that, for that scaled distance, there will not be particle velocities measured that are greater than the predicted level. Therefore, the blast vibration may be controlled to acceptable and lawful levels.

The control of airblast is also important. Airblast may

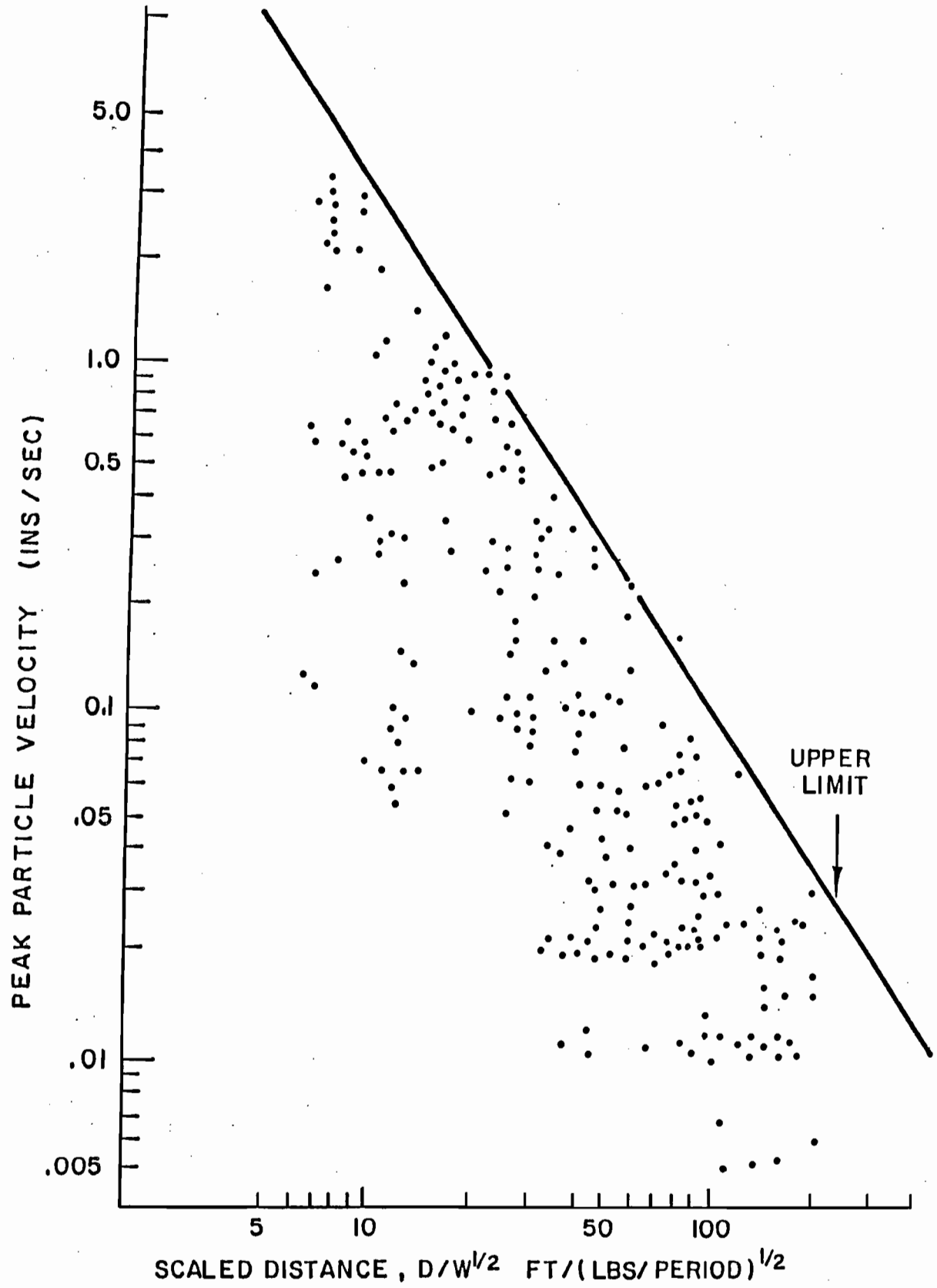


FIGURE 2-9: TYPICAL VIBRATION DATA FROM MULTIPERIOD DELAY BLASTS IN OPEN PITS AND ROCK STRIP MINES.

cause damage to windows. The level of overpressure however has to be quite high for this to result. A pressure of at least 0.1 psi (7 mbar) or 150dB is required. Figure 2-10 lists the typical values of blast overpressure that result in various kinds of damage[13].

The role of airblast therefore is not so much in actual damage in most cases. The importance is that the motions of the structure accompanying the arrival of airblast rattles unsecured objects within the building. This sudden, unexpected event surprises those in the structure and leads to distress and subsequent complaint. Therefore, in controlling airblast one is primarily attempting to reduce the degree to which the blasting is a nuisance to people in the nearby area.

For the type of blasting performed in this research and expected to be most useful for AML work the sources of airblast and noise are listed below:

1. Detonating cord trunklines.

2. Lack of proper stemming material.

3. Inadequate stemming height.

4. Surface delays and blasting caps.

5. Atmospheric conditions such as temperature inversions or wind in the direction of concern

6. Poorly confined shots next to sinkholes and individual subsidence features.

The first of these can be eliminated by the use of low grain count cord such as detaline or tube initiation systems of which NONEL is the most common. NONEL shock tube detonating at about 6,000 feet per second generates very little noise. If detonating cord must be used burying it with dirt will reduce noise. Another alternative is electric blasting. However, this is more complicated and more susceptible to stray currents and electric storms.

Poor stemming material does not contain the explosion gases in the hole properly Rather, rifling of the stemming occurs followed by venting of the very high pressure gases with attendant airblast. The best stemming material is -3/4 to +1/4 inch crushed rock. However, such material is not usually available and the common stemming material is drill cuttings. Normally this material works quite well but wet, saturated drill cuttings are very prone to rifling.

Inadequate stemming material means that the powder column has been allowed to rise too close to the surface. Bursting of the explosion gases through the upper surface occurs and

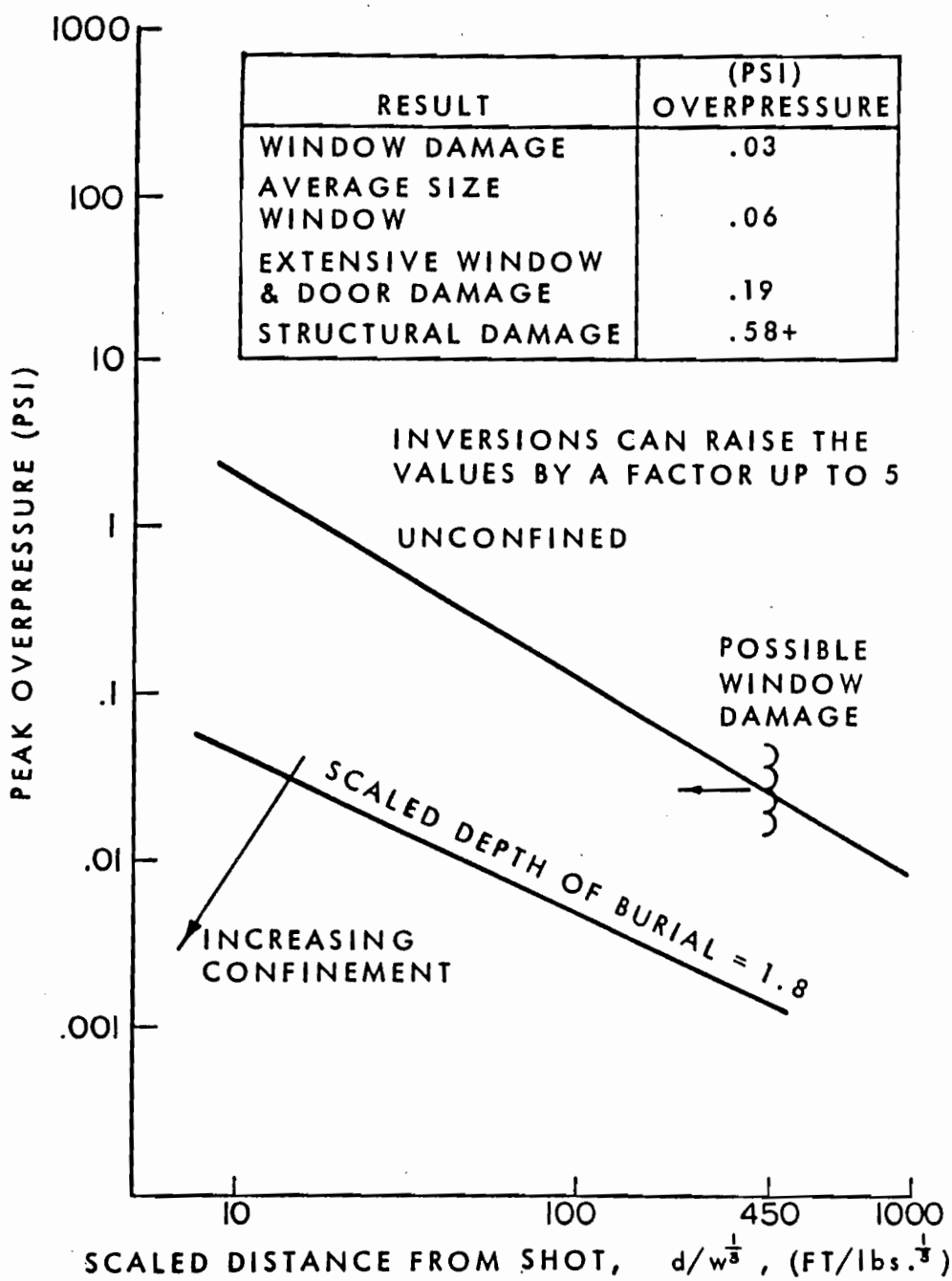


FIGURE 2-10: PEAK AIR BLAST PRESSURE GENERATED BY SPHERICAL CHARGES.

considerable airblast is generated. Therefore, calculations must be performed to determine the correct amount of stemming to contain the explosion while allowing the rock near the surface to be properly fragmented.

Blasting caps and millisecond delay elements detonating on surface create noise. Burying each delay and cap decreases the noise generated to a great extent. It is recommended that this procedure be followed wherever one is blasting in proximity to people and structures. Similarly, any detonating cord pigtails resulting from downlines in the blast holes should be buried.

Atmospheric conditions can play a significant role in airblast generation. Figure 2-11 is an illustration showing various atmospheric configurations and the resultant potential for airblast. This chart shows that temperature inversions can create unexpectedly high airblast at significant distances from the shot. Focussing can occur leading to regions of very intense airblast away from the blasting area. Therefore it is wise to avoid shooting when temperature inversions exist. Information concerning atmospheric conditions may be obtained from the local weather service. Small test shots (2 pounds) can be detonated on surface and the airblast measured with the blasting seismograph. Steam and smoke rising from industrial stacks and house chimneys can be observed. Rising gases followed by a leveling out at a given altitude indicates an inversion.

Wind will also affect airblast. Stiff winds in the direction of the structure can cause focussing of the airblast. On the other hand the upwind region will be quiet.

When shooting in an individual sinkhole or where a caving blast ends at a sinkhole it is important to insure that holes are not placed too close to the edge of the subsidence feature. If the burden is too small bursting will occur through the face. This leads to excessive airblast. In some cases it may be necessary to reduce the length of the powder column or eliminate upper decks in order to control airblast from this source.

There is often concern about the use of blasting because of anticipated problems with vibration and airblast. However, if the proper principles are applied blasting can be conducted quite close to people and structures without problems resulting. Later in the report the results of the vibration monitoring done during the test work is recorded and the implications are discussed.

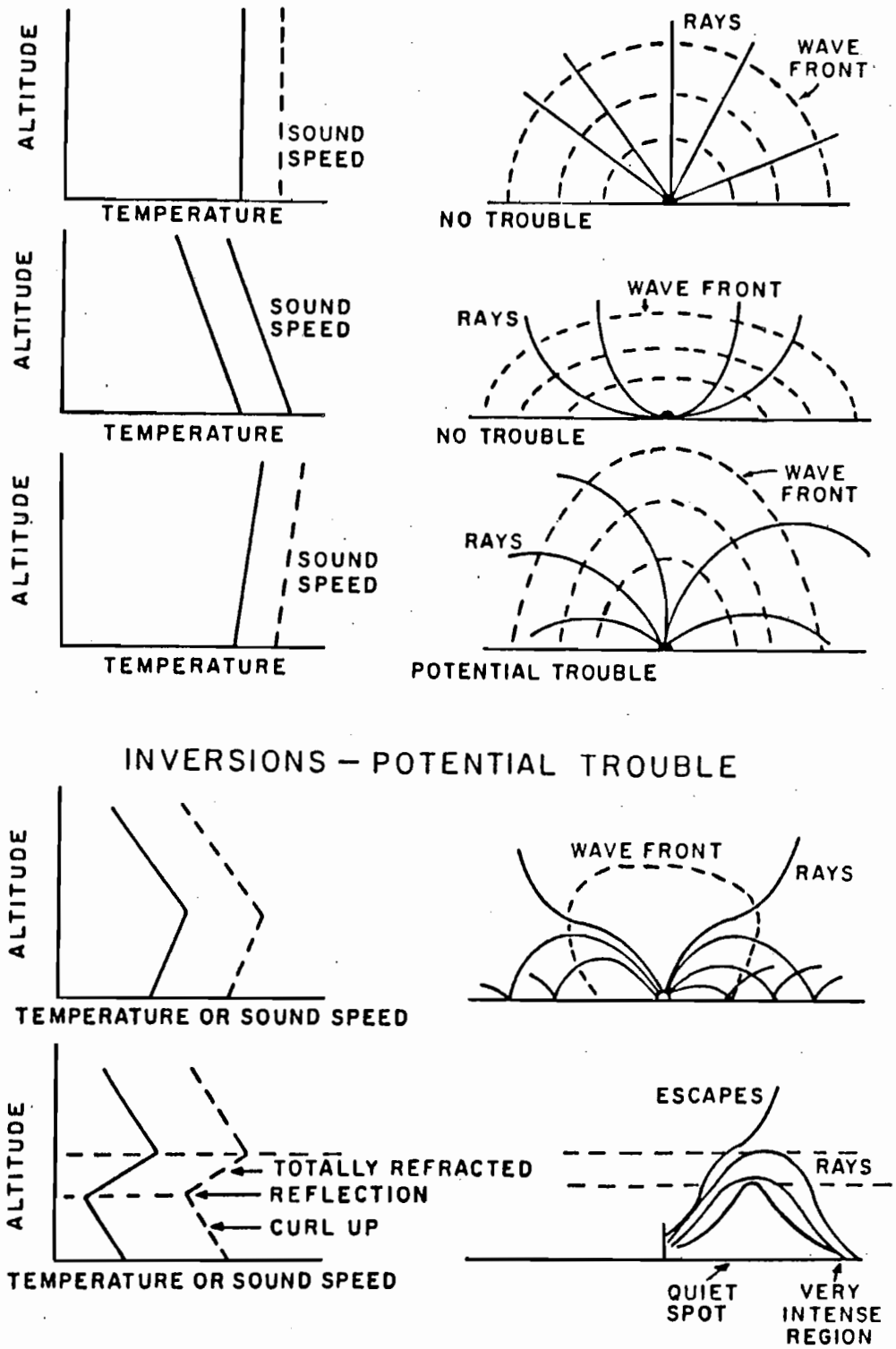


FIGURE 2-11: EFFECT OF ALTITUDE TEMPERATURE PROFILES ON AIR BLAST PROPAGATION.

3. SITE SELECTION, EVALUATION AND EXPLORATION

3.1. INTRODUCTION

This chapter describes how and why the site was selected and the work that had to be completed before actual blasting operations could commence. It is very important to have the best possible evaluation and knowledge of the site in order that the actual blasting work can be carried out efficiently and effectively. Overall project cost will be reduced when good pre-blasting design work is carried out.

The site location was finalized during August of 1987. Several possible areas were examined including AML land near Noonan, Wilton, New Leipzig and Beulah. All areas were located in North Dakota.

The location chosen was on land belonging to the North Dakota Game and Fish Department just east of Beulah, North Dakota. Coal had been mined at this property by underground techniques until 1954. There was much evidence of subsidence and sinkholes. These included features that had occurred in the past and others which were quite recent.

Any site where reclamation is to be effected will have to be evaluated prior to the onset of reclamation operations. Evaluation work is described in detail below. Included in this task is field reconnaissance to become familiar with the area, the locating and study of mine maps and the combining together of mine and topographical maps.

The purpose of this work is to establish the location of the workings and existing subsidence features so as to minimize on-site exploratory drilling which is costly. The success of pre-exploration work will vary however. Occasionally good records will be available but often the information is quite sketchy. The latter case will increase exploration drilling cost. It is a situation to be expected and therefore budgeted for in many cases.

Exploration drilling is also described in the chapter. Unless major advances occur in other types of void detection equipment this step will always be required. The drilling should be laid out in an orderly and planned function to optimize results. Drillers should keep careful hole logs recording depth, occurrence of void, location of rock layers and any information particular to the given hole.

Cost of site evaluation and exploration procedures is discussed below. It is very important to make realistic estimates of these costs. Otherwise the planned budget can quickly be spent simply trying to establish the characteristics of the property and the location of the workings.

3.2. CRITERIA FOR SITE SELECTION

In general terms AML sites are selected based on their priority. Abandoned mines that pose an immediate danger to health and safety have first priority. As these are reclaimed other sites that are less hazardous may be reclaimed and so forth.

For the current research other site selection factors were considered as well. These factors were important because of the research nature of the project and would not generally apply in choosing locations for AML reclamation.

All of the factors involved in selecting the site for the field testing are listed below:

- Amount and accuracy of available information including mine maps and topography maps. This was a project to test blasting techniques. Therefore, the more consistent the site the more time and money could be spent on blasting research. Also site irregularities would be less likely to skew the results.

- Site should have variable overburden. An objective of the contract was to determine the viability of blasting in overburden depths from shallow to deep.

- Site should be of adequate size. It was desired to shoot some four hundred blast holes. It was believed that a test of this magnitude would insure that valid conclusions could be drawn.

- If possible site should have workings of different dimensions. It was desired to determine the effect of development size on blasting design. If there were sinkholes on the property closure of individual holes could be studied.

- If possible the site should be one having some priority for reclamation within the state.

- Site should be one for which permission to operate could be obtained in a reasonable time. The contract became official in August. Therefore, it was necessary to start the field study quickly in order to finish before the onset of winter.

- Site should be within a reasonable distance of our offices. This would reduce travel and subsistence cost and make communication between the office and field easier.

- The property should have reasonable access so that explosives and heavy equipment could be brought in without the need for road building.

- The site should not be in close proximity to housing and other structures. This was a research project and therefore not all results are known in advance. To insure there would not be vibration or airblast problems it was felt to be prudent that there not be housing and so forth in close proximity.

- If possible the site should be one for which variations in reclamation might enhance the area. We wanted to determine the extent to which total reclamation could be effected solely by blasting and this was more likely where alternate reclamation could enhance the area.

3.3. SELECTION OF BEULAH TEST SITE

Four properties were considered for this test work. Initially a site at Noonan, North Dakota was considered and was suggested in the original proposal. However, the Noonan site was quite far removed from our offices making logistics more difficult. The property was next to a county road which meant that closing the area during blasting would be more difficult. Also a home was in close proximity to part of the site. The Noonan site was originally proposed because it contained an old highwall that could be blasted. However, we were requested to revise the proposal to exclude highwall blasting research. Therefore, the Noonan site lost much of its attractiveness.

The sites at Wilton and New Leipzig had two principle drawbacks. One was landowner reluctance to the proposed research. The other was that there was very little meaningful documentation of the sites. Therefore, it was considered that initial exploratory requirements would be excessive.

The Beulah property met all the criteria. This mine had been operated by a large mining concern. Complete and accurate mine maps had been kept that we were able to obtain.

Also the property had been flown and topographic maps were

produced that included the sinkholes that existed up until 1980. The mine maps could be superimposed on the topographic map to produce a composite map that was very helpful in site work.

Overburden above the workings varied from 35 to 65 feet. Thus a good study of the effect of overburden depth on blasting effectiveness could be made.

This location covered many acres. It was possible to isolate a ten-acre site which was quite adequate for our needs and could be worked on without hampering reclamation activities or plans in other areas of the overall site.

The Beulah property included rooms which were 22 feet wide by more than 200 feet long with 20 feet pillars between. It also had panel entries 12 feet wide. The panel entries had thicker roof coal left for stability than did the rooms. There were also numerous individual sinkholes on the property.

This site had priority for reclamation. Other sections of the site had been or were currently being reclaimed. This reclamation was primarily by direct fill of subsidence features.

The property is owned by the Game and Fish Department of the State of North Dakota. The Department was interested in looking at alternative reclamation techniques that could reclaim an entire area, such as by total collapse of the workings, and that might provide enhanced habitat for wildlife. Therefore, it was possible to obtain permission to utilize this property on a timely basis.

The Beulah site was located about 45 miles from our offices. This greatly reduced travel and subsistence costs. Also the site was only 12 miles from the offices and magazines of the explosive supplier which greatly facilitated logistics. It improved communication between this office and field as personnel were going out on a daily basis and meetings could be held prior to their departure.

There was an acceptable road into the property for access by semi-trailers (explosives), drills and attendant equipment. Loss of access only occurred during one major blizzard in early November.

The nearest housing was 2,500 feet from the site. This was considered not to be a problem. There was active surface mining in the area so persons living in this region were accustomed to blasts being detonated. Further the property was at least a mile away from main roads which made sealing the area for blasts easier.

This is a Game and Fish Department site. Therefore there is interest in how such an AML site could be reclaimed to enhance wildlife habitat as well as usefulness to the public.

It was thought that the varying topography likely to result from blasting might, therefore, be more useful than reclamation techniques that leave the area more like a cultivated field. Further, complete collapse of the workings would help to minimize hazards in the future whereas further subsidence could occur in an area where direct fill of existing sinkholes was employed as the reclamation method.

For these reasons the Beulah site was deemed the most suitable for this research. Work therefore proceeded on this property. The site was found to be very adequate for our needs. Figure 3-1 is a map of North Dakota showing the general location of the site.

Having selected the Beulah site as the preference for the field blasting program, several meetings were held with the North Dakota Game & Fish Department, managers of the site area. Most of the liaison work was carried out by James Thompson, of RPM Inc., who coordinated the site selection process with the PSC, and obtained written permission from the Game & Fish Department to use their land.

3.4. SITE EVALUATION

The following is a brief description of the site evaluation work carried out following selection of the test site for the 1986 field blasting work.

The list below summarizes the main operations carried out during site evaluation and exploration.

- site selection
- obtain aerial photos and old mine maps if available
- superimpose map and photo data onto working field map
- site reconnaissance and survey
- layout exploration drilling pattern
- carry out exploration drilling to locate development and to determine depth to which caving has occurred
- evaluate results; define areas suitable for blasting
- layout drilling to determine width and lateral extent of underground openings
- detailed exploration drilling

In general this may be used as a guideline for future work of this type. We were, however, somewhat fortunate to obtain such good maps of the area for this project.

Some old underground coal workings follow a very haphazard distribution. In part this may reflect ground conditions, and in other cases it may simply reflect the lack of a rational mine

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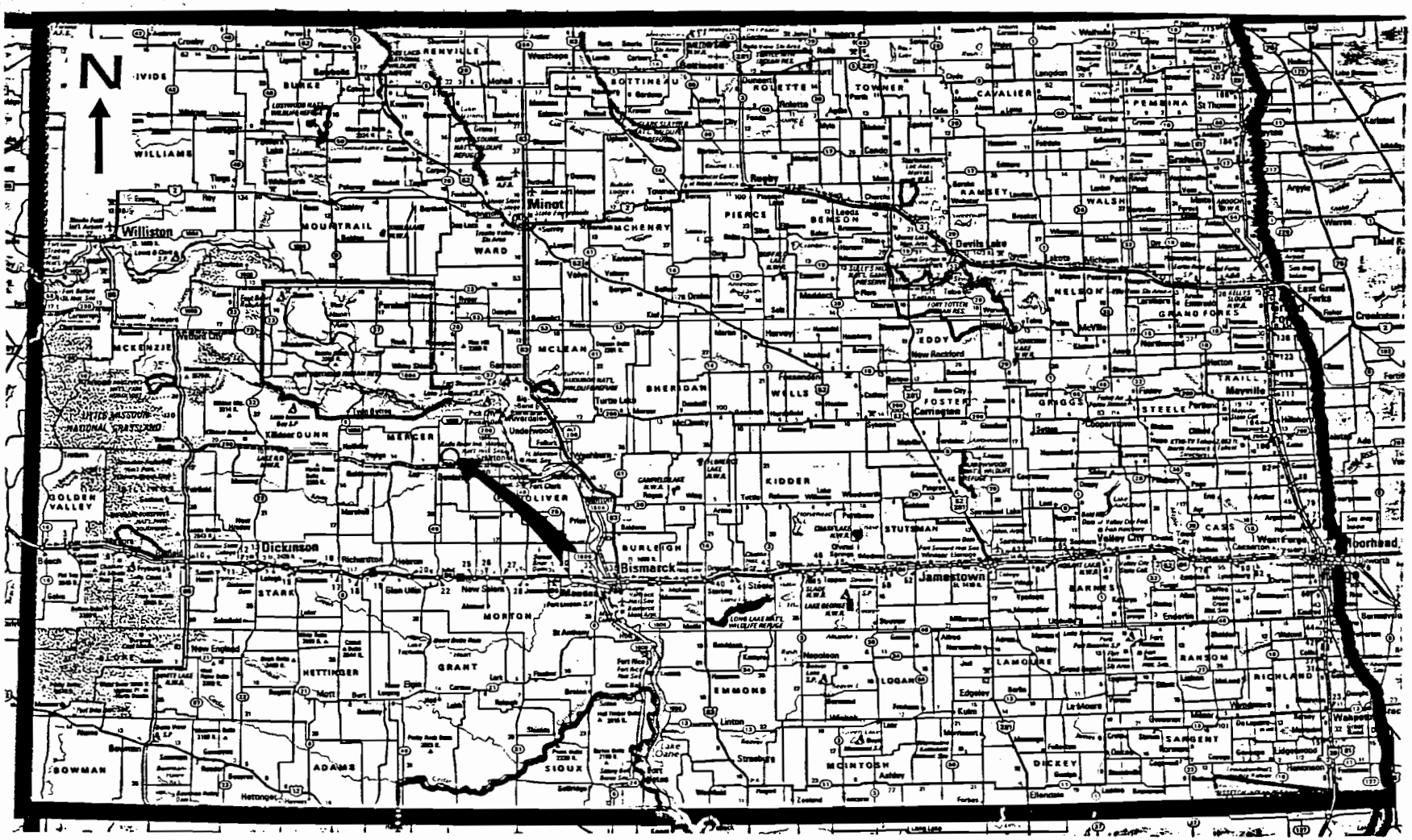


FIGURE 3-1: GENERAL LOCATION OF THE BEULAH TEST SITE

○ Site Location

plan. Fortunately, the site employed for the 1986 testwork contained a regular system of underground development. In addition, there was close agreement between mine plans which were available and the actual location of development as determined by exploration drilling.

However, it should be appreciated that most areas which now constitute AML sites were mined over 30 years ago. Mine plans for such operations, even where these were largely adhered to as in the case of the 1986 test site, were guidelines, and not rigid. In any area where only 50 feet or so of cover, loosely consolidated, is encountered there are going to be local variations in ground conditions which may require deviation, or actual abandonment, of production faces.

Good agreement between surface evidence and underground mine maps was a major factor in selection of this site. It certainly allowed savings to be made in the cost of site evaluation, surveying, and the exploration drilling effort.

3.4.1. Pre-exploration Work

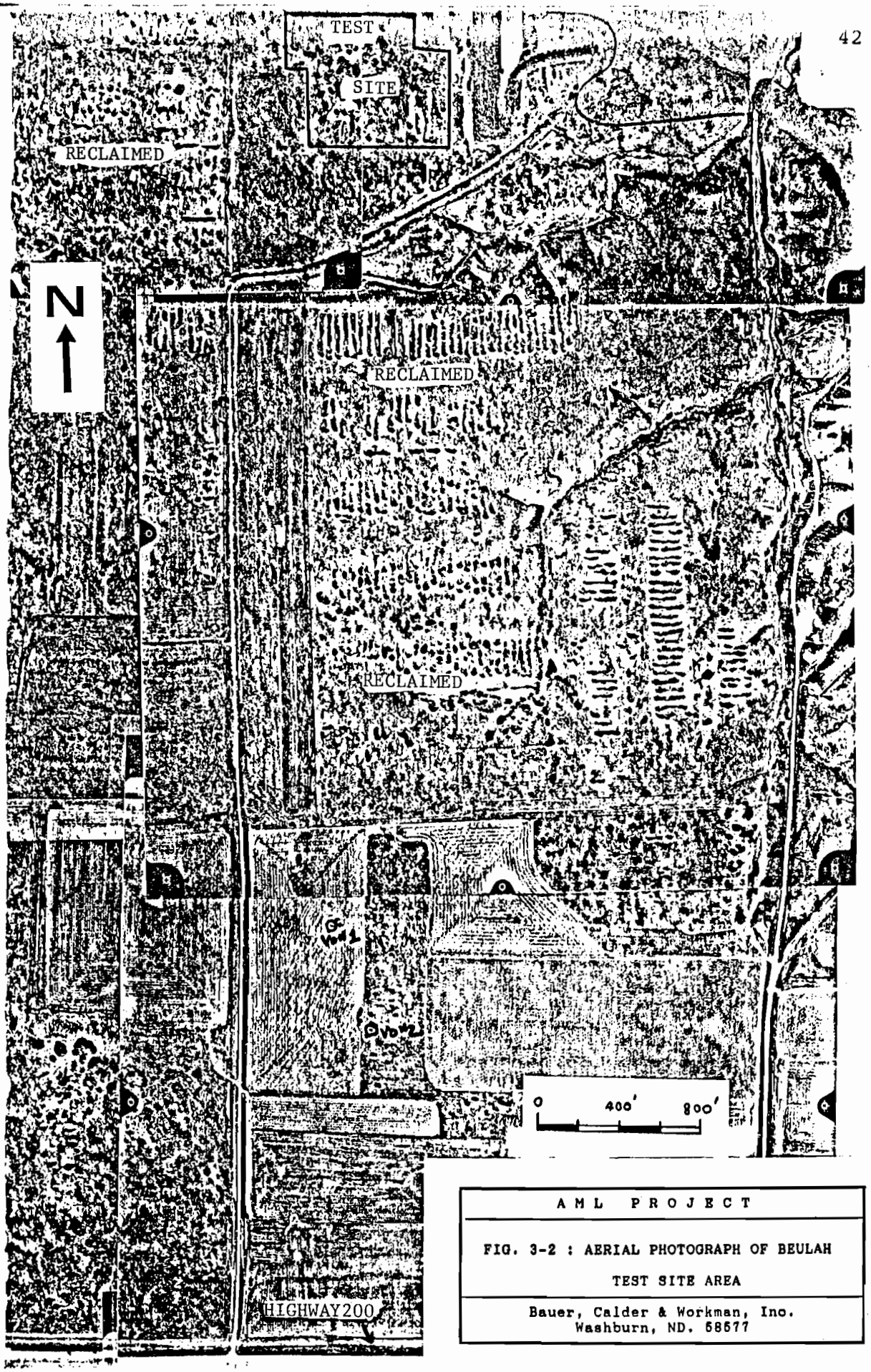
Evaluation of the Beulah AML Test Site commenced on September 5th, 1987. Fig. 3-2 is a reduced photocopy of a composite aerial photo of the area. The site is located about one and a quarter miles north of ND Highway 200, at the northern limit of the Knife River Coal Company's North Beulah Mine.

Sinkholes visible on Fig. 3-2 to the south and west of the test site have been filled by the North Dakota Public Service Commission since the aerial photos were taken. The area immediately to the west of the test site was filled in the summer of 1986, between the original site reconnaissance program and the site evaluation stages.

As part of a major AML site evaluation project in 1979-80, RPM Inc of Bismarck, North Dakota, prepared 1:100 scale maps of priority sites. These were available for the North Beulah Mine area, and consisted of topographic maps on which had been plotted the positions of sinkholes as indicated from aerial photos.

3.4.1.1. Field Reconnaissance

The map of the site area was used as a base topographic map for field reconnaissance work. A reduced photocopy of this map is shown in Fig. 3-3. The first stage of site evaluation work consisted of the establishment of west-east trending survey lines which traversed the area of interest. These are shown as dotted line in Fig. 3-3. The numbers associated with sinkhole locations here refer to the depth of the sinkholes in feet.



AML PROJECT

FIG. 3-2 : AERIAL PHOTOGRAPH OF BEULAH TEST SITE AREA

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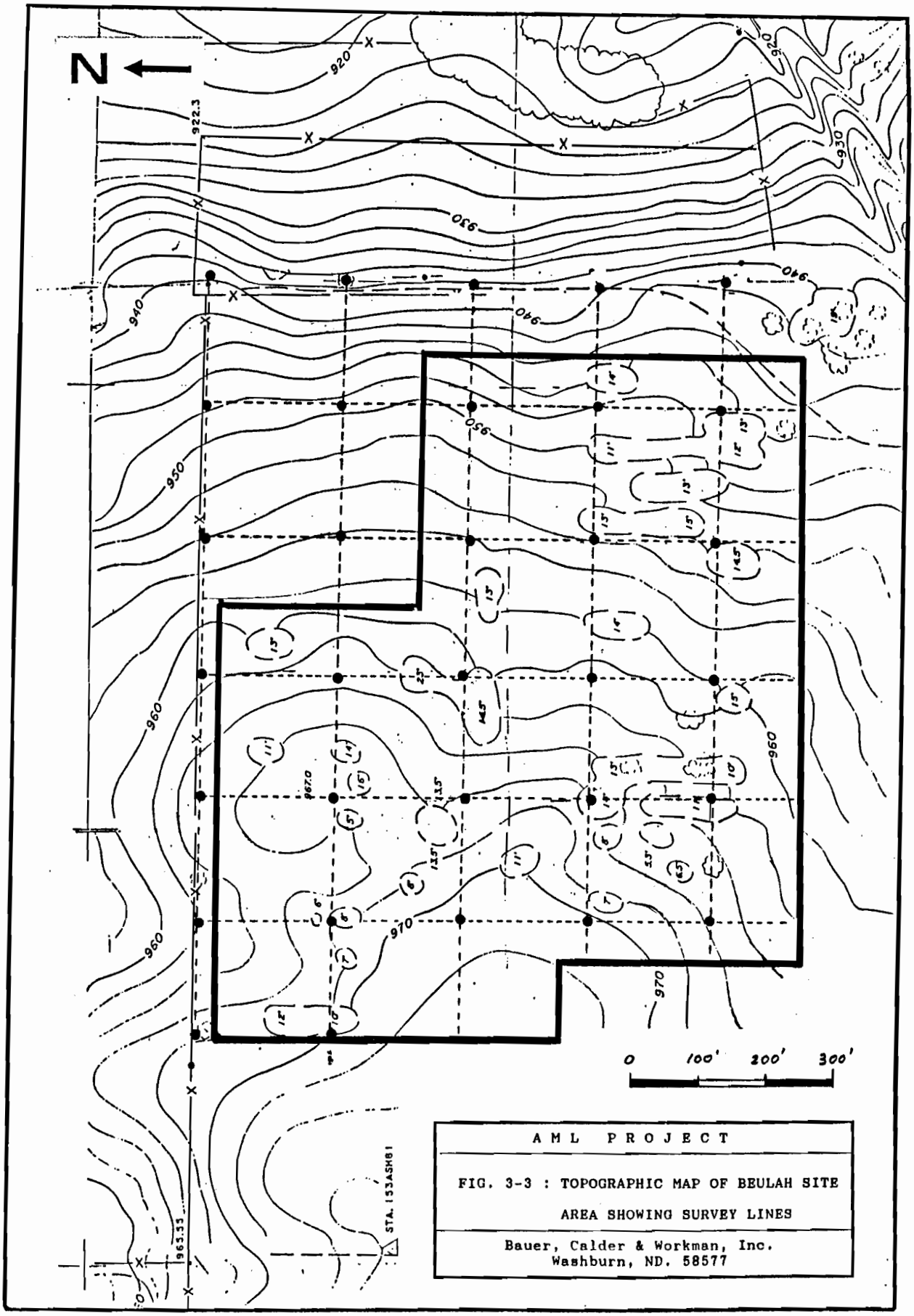

AML PROJECT

FIG. 3-3 : TOPOGRAPHIC MAP OF BEULAH SITE AREA SHOWING SURVEY LINES

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A reference grid was located using a measuring tape and a Brunton compass. The grid was established at 150 foot centers. At the outset of the field program the vegetation was in excess of three feet tall in most of the area, and despite the gentle topography, visibility was often limited to only small sections of the site.

Five feet high wooden stakes were driven into the ground at the positions indicated by filled black circles in Fig. 3-2. In view of the quality of the topographic maps available, and the presence of landmarks for locating positions in the field, it was decided to forego a formal surveying exercise.

3.4.1.2. Mine Maps

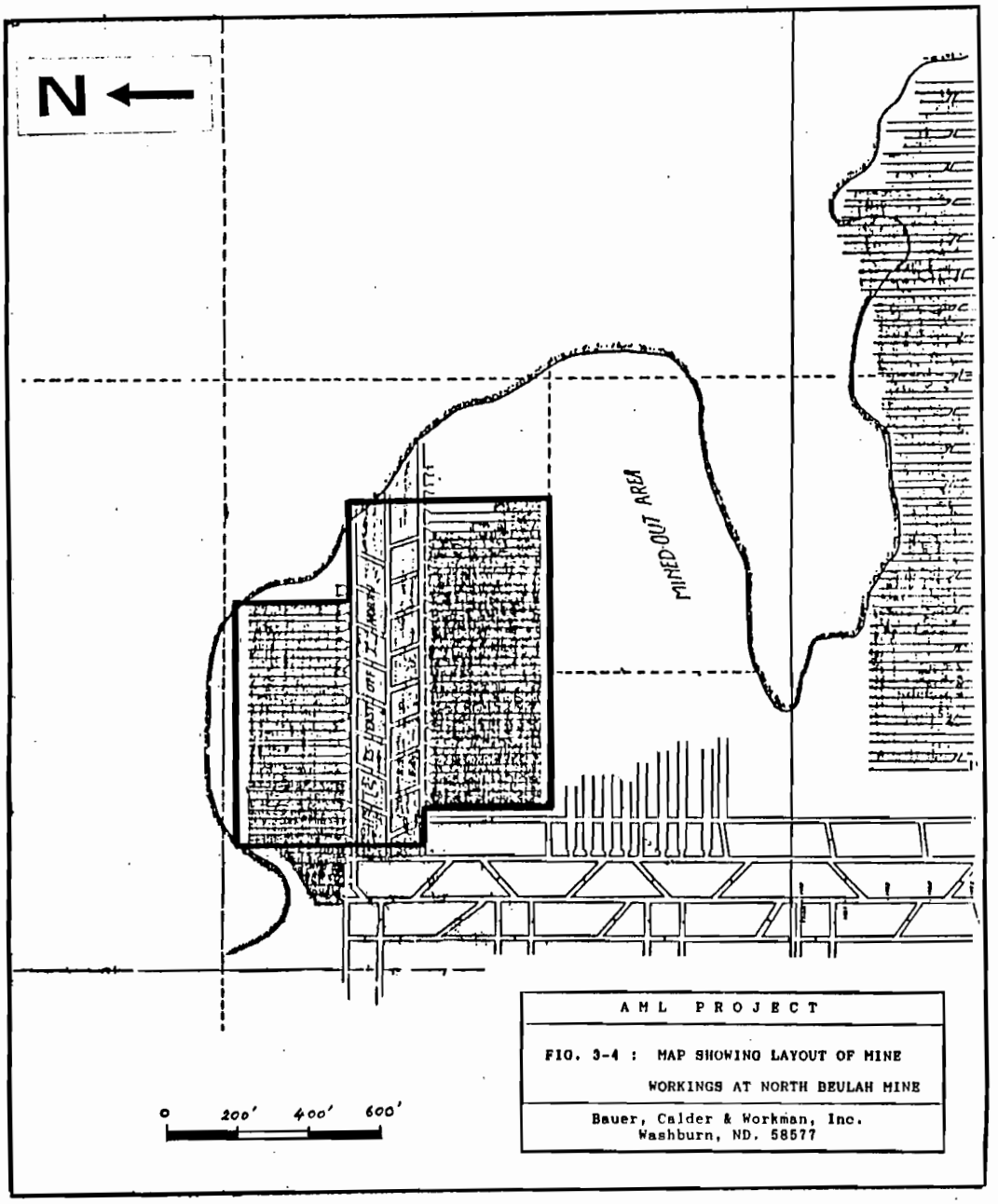
Maps of the North Beulah mine were made available by the Knife River Coal Mining Company, which operated the mine. These were in effect mine plans - there was no post-mining survey. A reduced photocopy of the map for the site area is shown in Fig. 3-4. It shows two areas of room and pillar mining, separated by a central area of panel entries and connecting cross-cuts.

Rooms on this plan are 22 feet wide, separated by 18 foot wide pillars. The panel entries are 80 feet apart, with the connecting cross cuts also about 80 feet apart. The areas associated with the different parts of the test site are shown in table 3-1.

Table 3-1 : DIMENSIONS OF TEST SITE AREA.

Location	Dimensions (feet)	Area (acres)
North Room-and-pillar	550 x 250	3.16
South Room-and-pillar	625 x 300	4.30
Central Panel Entries	700 x 150	2.41
TOTAL	-	9.87

The area to the south of the test site was known as the Boettcher Mine, and was not mined by Knife River Coal Company. However, a map for this mine was available, very well surveyed; it demonstrated a very different mining philosophy from the regular room and pillar operation.



AML PROJECT

FIG. 3-4 : MAP SHOWING LAYOUT OF MINE WORKINGS AT NORTH BEULAH MINE

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3.4.1.3. Composite Maps

One of the most important pre-exploration engineering efforts was to produce a composite map showing surface topography, sinkholes and underground workings. This is presented in Fig. 3-5, and shows quite clearly the north-south alignment of sinkholes in the north and south room-and-pillar mining areas. In addition it shows the close agreement in the central area of panel entries between sinkholes and the intersections of cross-cuts and panel entries.

As one would expect, there was less than total agreement between the positions of sinkholes and the individual rooms as shown on the mine plan. In areas of bad ground it is probable that the room could not be started in exactly the desired position from the panel entry. Local ground conditions would have forced rooms to be mined at less than the 22 foot planned width, and in extreme cases they would have been terminated short of the planned mining limit. One of the major functions of exploration drilling was therefore to determine the degree of correlation between the mine plan and actual mining.

Fig. 3-6 shows an enlarged view of the composite map of the site area, with the positioning of actual rooms and pillars. It can be seen that there was generally very good correlation between the mine plan and actual mining, as shown by sinkhole positions. It should be appreciated that the irregularities in the 22 foot room and 18 foot pillar pattern shown in the northern mining area are in fact those deduced from exploration drilling.

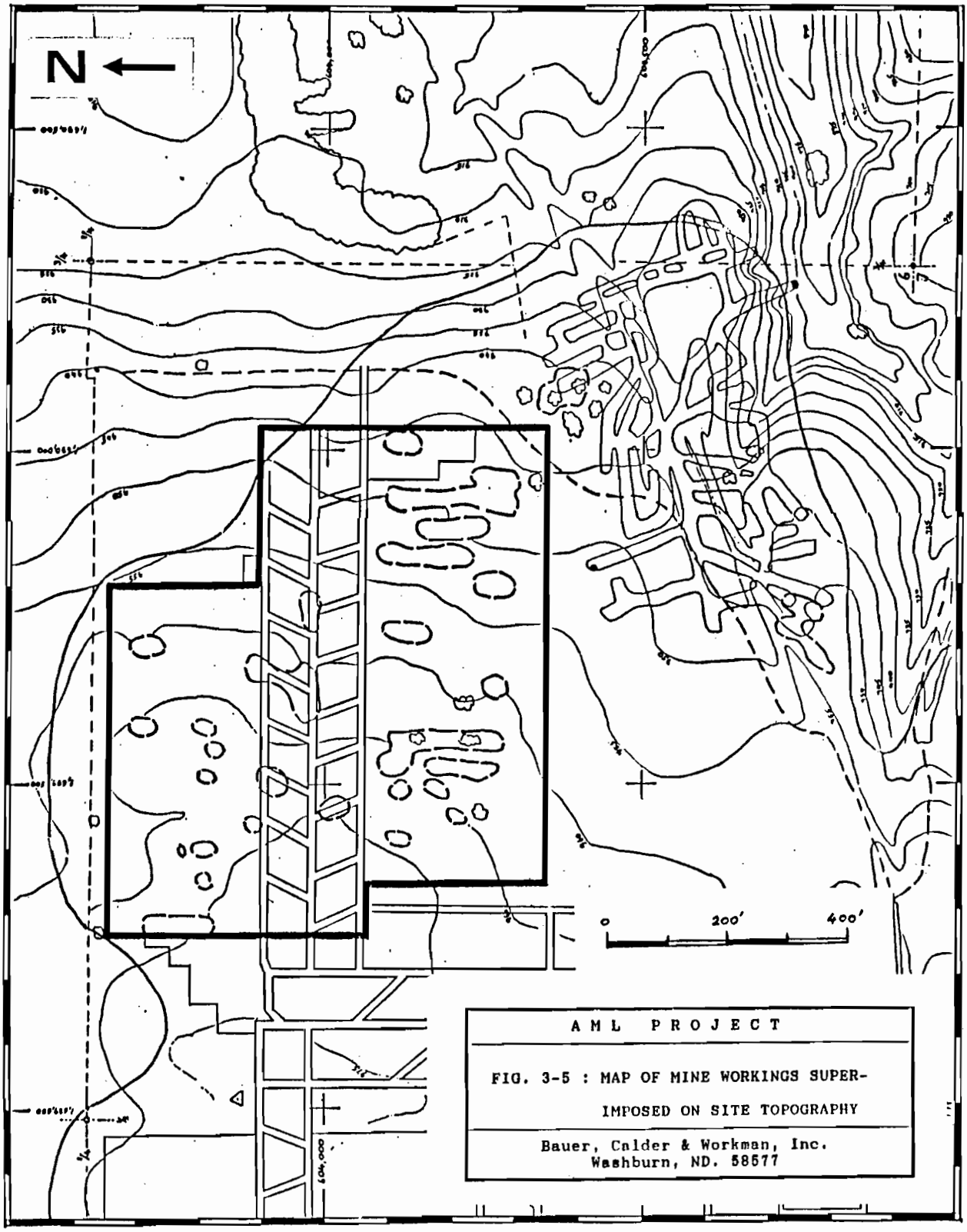
This was used as a "base-map" for much of the fieldwork, and is employed elsewhere in this report to present other information such as exploration drillhole and blast location. There were 14 rooms in the northern, and 15 in the southern room-and-pillar mining areas. These were numbered in each case in ascending order from west to east as shown on Fig. 3-6, using the prefixes N- and S- for rooms north and south, respectively, of the panel entries.

The panel entries themselves were given the codes NH, CH and SH for North, Central and South panel entries respectively Connecting cross-cuts were numbered in ascending order from west to east, and given the codes NC- and SC- for north and south locations respectively.

3.4.2. Exploration Drilling

3.4.2.1. Aims of Drilling Program

The purposes of exploration drilling at any AML site to which blasting is to be applied as a reclamation method are



AML PROJECT

FIG. 3-5 : MAP OF MINE WORKINGS SUPER-IMPOSED ON SITE TOPOGRAPHY

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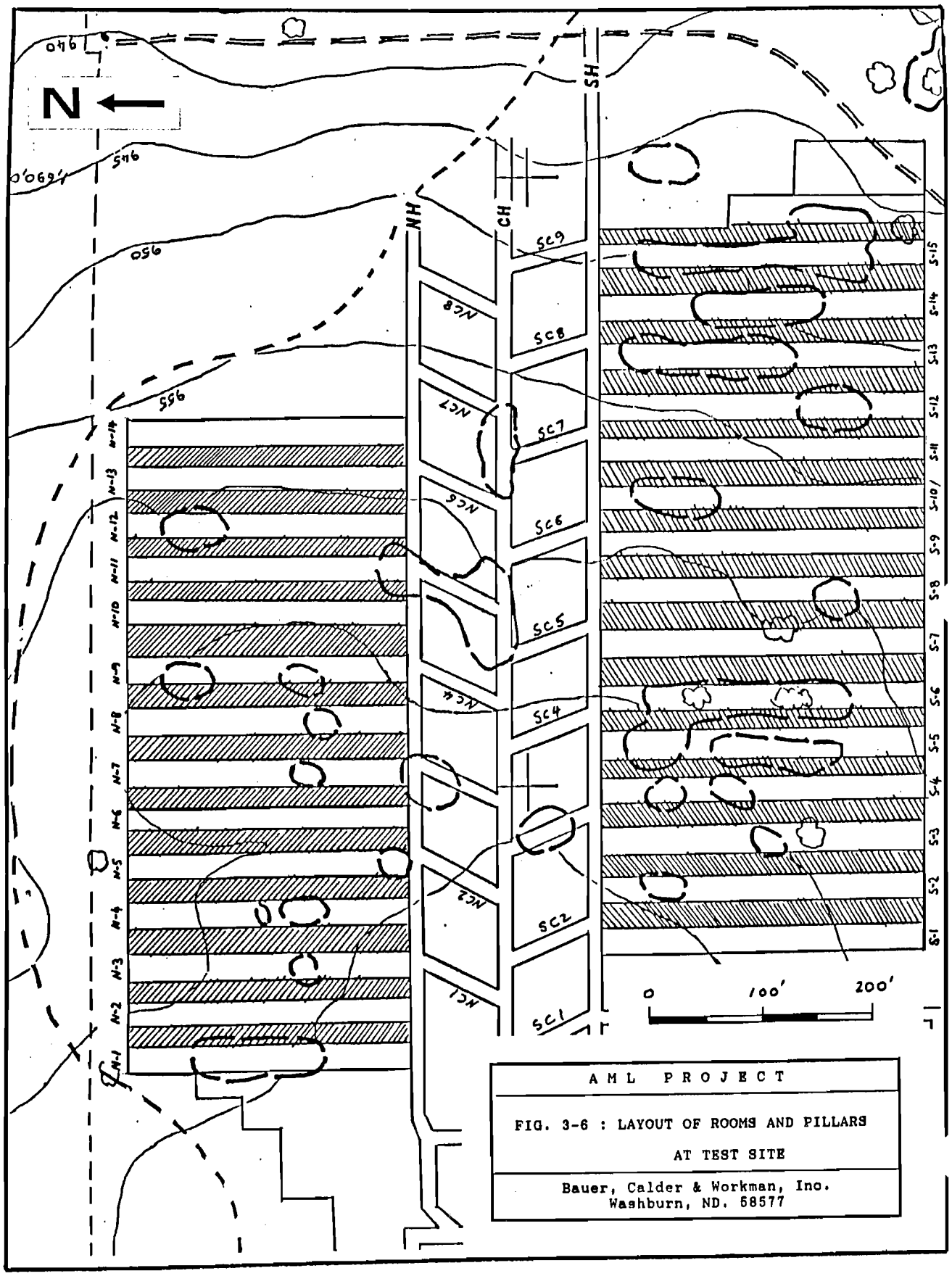

N
SH
NH
CH
SC9
SC8
SC7
SC6
SC5
SC4
SC2
SC1
NC8
NC7
NC6
NC4
NC2
NC1
N-1
N-2
N-3
N-4
N-5
N-6
N-7
N-8
N-9
N-10
N-11
N-12
N-13
N-14
S-1
S-2
S-3
S-4
S-5
S-6
S-7
S-8
S-9
S-10
S-11
S-12
S-13
S-14
S-15
0
100'
200'
AML PROJECT
FIG. 3-6 : LAYOUT OF ROOMS AND PILLARS AT TEST SITE
Bauer, Calder & Workman, Inc.
Washburn, ND. 58577

summarized in the following:

- to locate underground workings which have not collapsed

- to determine the degree of general correlation between mine plans (if available) and actual mining

- to establish whether rooms were mined to their planned limits, lateral and longitudinal

- to determine whether rooms are still open, or whether partial collapse has taken place even though there is as yet no surface sinkhole

- to determine overburden types and depths for the site area

With this information available it is possible to plan the blasting operation as a whole. In view of the research oriented nature of the 1986 testwork, it was particularly important to obtain a good idea of the distribution and state of the underground workings prior to blasting.

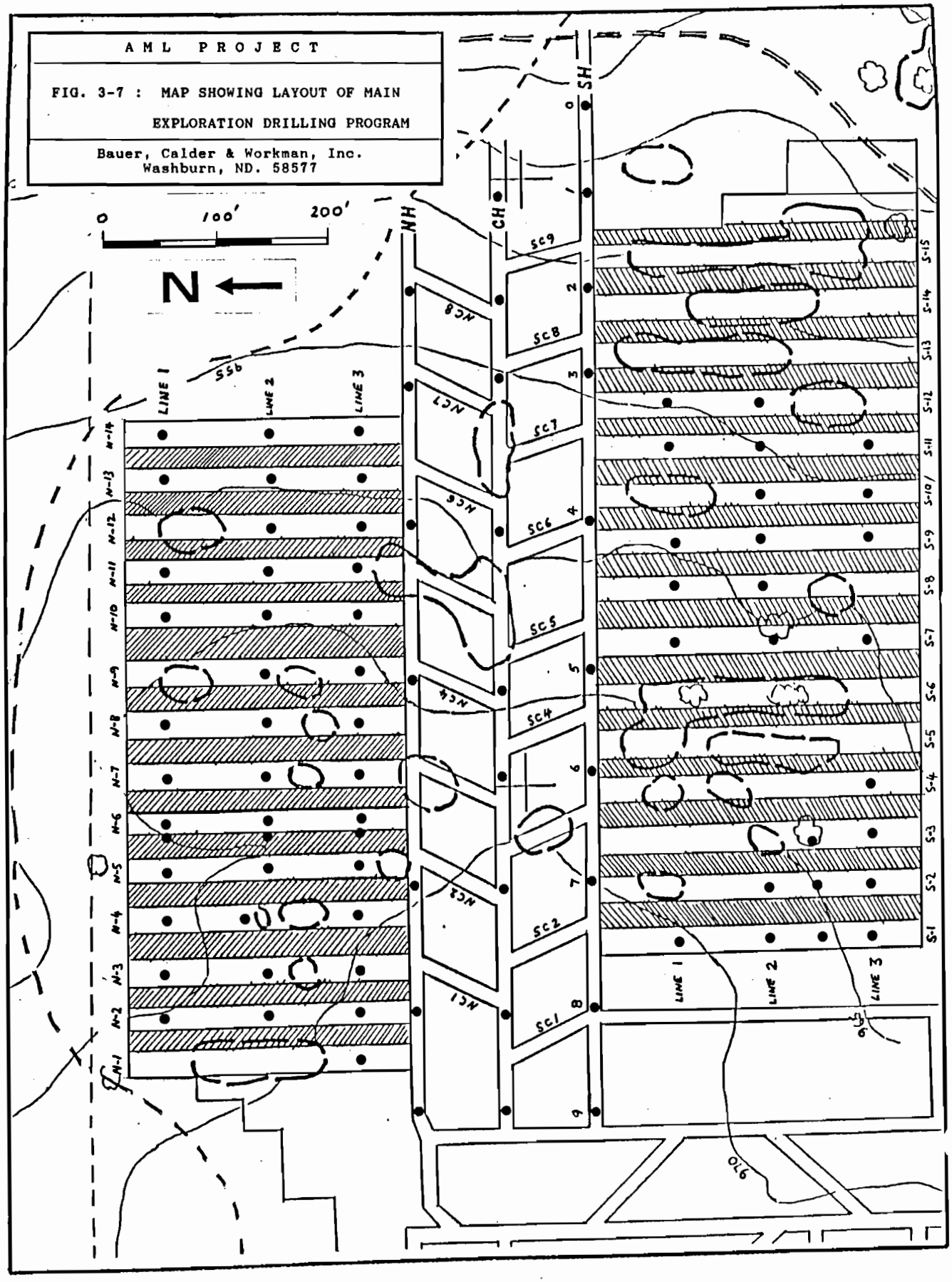
3.4.2.2. Exploration Drilling Layout

Since the combined room and pillar width was known to be 40 feet, exploration was planned using holes 40 feet apart in the west-east direction. In practice a number of exploration hole positions were located by a simple alignment on the centers of existing sinkholes. Wooden stakes were driven into the ground at these positions, and then a measuring tape was used to locate the estimated lateral position of other rooms. In some cases the position of wooden stakes located during pre-exploration survey work provided a useful basis for drillhole location.

Three lines of exploration holes were laid out; these are shown, together with borehole locations, in Fig. 3-7. Line 1 in the northern area was laid out in general some 30 feet south of the planned mining limit. Lines 2 and 3 were laid out at approximately 80 and 160 feet south, respectively, of line 1.

Exploration holes in the south mining area were laid out in the same way, using where possible an 80 x 40 foot grid. Again the lines were numbered in ascending order going south, as shown in Fig. 3-7. Line 1 was established about 60 feet south of the southernmost panel entry.

Natural caving in the south area was rather more advanced than in the north, and for this reason there were fewer exploration holes drilled here. The numbering system used for


AML PROJECT
FIG. 3-7 : MAP SHOWING LAYOUT OF MAIN EXPLORATION DRILLING PROGRAM
Bauer, Calder & Workman, Inc.
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0
100'
200'
N
LINE 1
LINE 2
LINE 3
N-1
N-2
N-3
N-4
N-5
N-6
N-7
N-8
N-9
N-10
N-11
N-12
N-13
N-14
S-1
S-2
S-3
S-4
S-5
S-6
S-7
S-8
S-9
S-10
S-11
S-12
S-13
S-14
S-15
NH
CH
SH
NC1
NC2
NC3
NC4
NC5
NC6
NC7
NC8
NC9
SC1
SC2
SC3
SC4
SC5
SC6
SC7
SC8
SC9
1
2
3
4
5
6
7
8
9
955
960
970

exploration boreholes was set up to coincide with the numbering system for rooms and drilling lines. Hole number N-7-2 therefore refers to the hole drilled in the north mining area on room number 7, line 2.

Exploration holes in the area of panel entries were planned to intersect the center lines of these entries. They were drilled on each line, where possible, at 80 feet separation. The numbering system went from east to west - in other words, hole number CH-4 was the fourth hole going west along the line of the central panel entry.

3.4.2.3. Field Procedure

Drilling was contracted out to Moe Drilling, Inc. of Mott, North Dakota, using a Gardner Denver truck-mounted rig. All exploration drilling was carried out using compressed air at 6 inch diameter. Twenty foot long drill rods were employed in the drill stem; the drill rig is illustrated in Fig. 3-8.

Some initial exploration drill holes were laid out at the Beulah test site on September 15 and 16. All of the exploration drilling program was directly supervised by the Field Engineer, and took place between September 18 and September 25 1987.

Minor changes in drillhole location were carried out in the field as drilling progressed to reflect results from previously drilled holes. For this reason, it is felt that the presence of the Field Engineer throughout exploration drilling is justified in such work.

There was in general a very high success-rate for the exploration drillholes, due to the quality of the map information available, and the regular nature of the mining method employed. An irregularity in the room and pillar widths in the area of rooms N-5 and N-6 resulted in the requirement to re-drill holes on all three lines. In the southwest of the test site area rooms S-1 to S-3 were found to be shorter than planned and an intermediate line of holes had to be drilled in this area (see Fig. 3-7).

The following information was recorded at each exploration drilling site:

- borehole location
- depth of major overburden type changes (such as sand, clay, rock layers etc
- depth of top of coal seam
- depth to void
- depth to bottom of room (in some cases)

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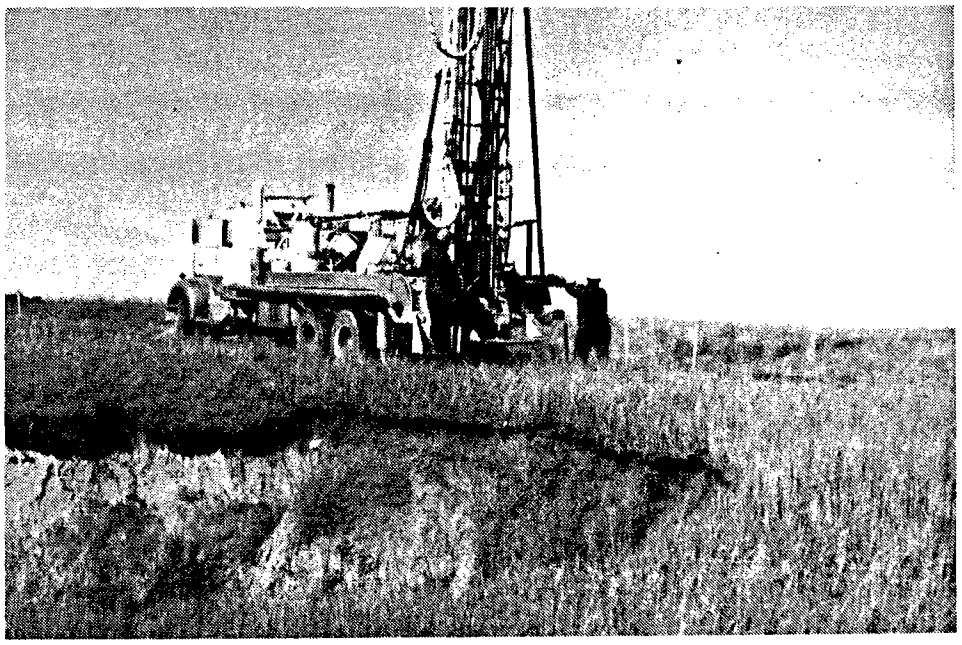

FIG. 3-8: DRILLING OPERATIONS DURING FIELD TESTWORK PROGRAM

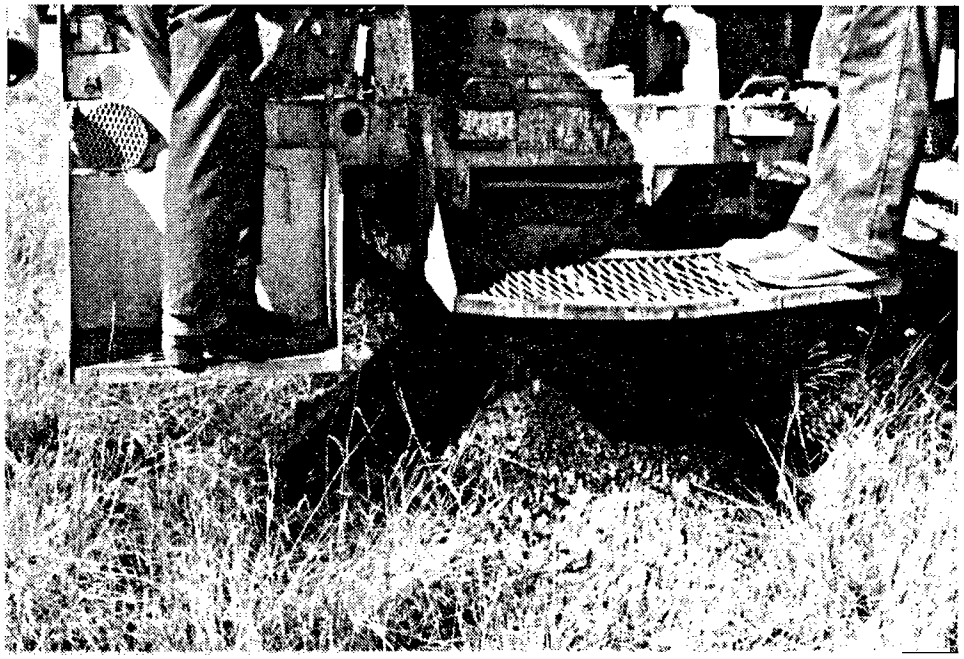

FIG. 3-9: CLOSE-UP OF DRILL-CUTTINGS HEAP SHOWING CHANGE IN COLOR AS COAL SEAM IS INTERSECTED

The presence of coal dust in the stemmings as the seam was hit was a very marked indication of this contact. This is illustrated in Fig. 3-9. The location of the void was generally very obvious, as there was an immediate downward movement of the rig as bearing pressure on the bit ceased.

In cases where the overburden had caved above the coal, and the void was filled with loosely consolidated material, this was generally indicated by a loss of air circulation in the drill stem.

3.4.2.4. Drilling Statistics

Statistics related to the 1986 exploration drilling program at the Beulah test site are presented in Table 3-2. A total of 96 holes were drilled, with a total drilled length of 5485 feet. Each hole in table 3-2 is identified by a sequence number and a hole number according to the numbering technique described in the previous section.

The table was created using a spreadsheet, which was employed throughout the project to keep track of drilling statistics and costs. The column titled "SURF. EL." is the approximate surface elevation of the borehole collar, based on the available topographic map for the area.

Distances down the borehole to top of coal ("TOC"), bottom of coal ("BOC") if a void was encountered, and bottom of seam ("BOS") if no void was encountered are recorded in the table. The column marked "VOID" refers to the depth to the underground opening encountered. In cases where the opening had caved above the coal seam, the depth at which a void was found is entered in the column headed "CAVED".

Total overburden depth and roof coal thickness is calculated by the spreadsheet in the columns titled "TOT.OB" and "TOT. COAL" respectively. The top of coal elevation is also calculated with respect to the surface elevation at the site. These figures indicate that the coal seam was essentially horizontal, but with a very slight dip towards the south of about half a degree.

TABLE 3-2 : AML PROJECT - EXPLORATION DRILLING SUMMARY

NO	HOLE NO.	DATE	SURF.EL	TOC	BOC	BOS	VOID	CAVED	TOT.OB	TOT COAL	TOC ELEV	HOLE DEP	DRIL. DEP
1	N-2-1	18 SEP	965.0	52.0	56.0		56.0		52.0	4.0	913.0	56.0	56
2	N-3-1	18 SEP	964.0	51.0	57.0		57.0		51.0	6.0	913.0	57.0	60
3	N-4-1	18 SEP	964.0	51.0	56.0			56.0	51.0	5.0	913.0	60.0	60
4	N-5-1	18 SEP	964.0	52.0	61.0	61.0			52.0	9.0	912.0	75.0	75
5	N-6-1	18 SEP	965.0	52.0	72.0	72.0			52.0	20.0	913.0	72.0	75
6	N-7-1	18 SEP	966.0					25.0				25.0	30
7	N-8-1	18 SEP	966.0	54.0	57.0		57.0		54.0	3.0	912.0	57.0	60
8	N-10-1	18 SEP	964.0	53.0	72.0	72.0			53.0	19.0	911.0	72.0	75
9	N-11-1	18 SEP	963.0					18.0				18.0	20
10	N-2-2	18 SEP	968.0					40.0				40.0	40
11	N-3-2	18 SEP	967.0					35.0				35.0	60
12	N-4-2	18 SEP	967.0					36.0				36.0	40
13	N-5-2	18 SEP	967.0					47.0				50.0	50
14	N-6-2	18 SEP	967.0	54.0	74.0	74.0			54.0	20.0	913.0	74.0	75
15	N-7-2	18 SEP	967.0	54.0	56.0		56.0		54.0	2.0	913.0	56.0	60
16	N-8-2	18 SEP	967.0	57.0	58.0		58.0		57.0	1.0	910.0	58.0	60
17	N-9-2	18 SEP	966.0					45.0				45.0	50
18	N-10-2	18 SEP	965.0	53.0	58.0		58.0		53.0	5.0	912.0	58.0	60
19	N-2-3	18 SEP	970.0	59.5	65.0		65.0		59.5	5.5	910.5	65.0	70
20	N-3-3	18 SEP	970.0	58.0	67.0		67.0		58.0	9.0	912.0	67.0	70
21	N-1-3	18 SEP	971.0	60.0	67.0		67.0		60.0	7.0	911.0	67.0	70
22	N-4-3	18 SEP	969.0					59.0				59.0	60
23	N-5-3	18 SEP	968.0	56.0	65.0		65.0		56.0	9.0	912.0	65.0	65
24	N-6-3	18 SEP	967.0	55.0	75.0	75.0			55.0	20.0	912.0	75.0	80
25	N-7-3	18 SEP	967.0	54.0	74.0	74.0			54.0	20.0	913.0	74.0	76
26	N-7-3A	18 SEP	967.0	55.0	59.0		59.0		55.0	4.0	912.0	59.0	60
27	N-6-3A	18 SEP	967.0	55.0	57.0		57.0		55.0	2.0	912.0	57.0	60
28	N-8-3	18 SEP	966.0	54.0	55.0		55.0		54.0	1.0	912.0	55.0	56
29	N-9-3	18 SEP	966.0	54.0	59.0		59.0		54.0	5.0	912.0	59.0	60
30	N-10-3	19 SEP	964.0	51.0	55.0		55.0		51.0	4.0	913.0	55.0	60
31	N-11-3	19 SEP	964.0	49.0	53.0		53.0		49.0	4.0	915.0	53.0	60
32	N-12-3	19 SEP	962.0	46.0	49.0		49.0		46.0	3.0	916.0	49.0	60
33	N-13-3	19 SEP	959.0	45.0	53.0		53.0		45.0	8.0	914.0	53.0	60
34	N-14-3	19 SEP	958.0	42.0	47.0		47.0		42.0	5.0	916.0	47.0	50
35	CH-2	19 SEP	953.0	38.0	44.0		44.0		38.0	6.0	915.0	44.0	50
36	CH-6	19 SEP	968.0	55.0	61.0		61.0		55.0	6.0	913.0	61.0	61
37	CH-8	19 SEP	971.0	61.0	68.0		68.0		61.0	7.0	910.0	68.0	70
38	N-6-2A	19 SEP	965.0	55.0	57.0		57.0		55.0	2.0	910.0	57.0	60
39	N-6-1A	19 SEP	965.0	53.0	60.0		60.0		53.0	7.0	912.0	60.0	60
40	N-13-1	19 SEP	959.0					45.0				45.0	50
41	N-14-1	19 SEP	957.0					32.0				32.0	35
42	N-11-2	19 SEP	963.0	52.0	57.0		57.0		52.0	5.0	911.0	57.0	60
43	N-12-2	19 SEP	962.0	49.0	55.0		55.0		49.0	6.0	913.0	55.0	60
44	N-13-2	19 SEP	960.0	46.0	49.0		49.0		46.0	3.0	914.0	49.0	50
45	N-14-2	19 SEP	958.0	42.0	46.0		46.0		42.0	4.0	916.0	46.0	50
46	NH-2	19 SEP	953.0	42.0	48.0		48.0		42.0	6.0	911.0	48.0	50
47	SH-2	19 SEP	951.0	36.0	42.0		42.0		36.0	6.0	915.0	42.0	45
48	SH-4	23 SEP	959.0	43.0	47.0		47.0		43.0	4.0	916.0	47.0	50
49	NH-4	23 SEP	961.0	46.0	49.0		49.0		46.0	3.0	915.0	49.0	50
50	NH-8	23 SEP	971.0	60.0	79.0	79.0			60.0	19.0	911.0	79.0	80
51	NH-8A	23 SEP	971.0	60.0	79.0	79.0			60.0	19.0	911.0	79.0	80
52	NH-8B	23 SEP	971.0	60.0	67.0		67.0		60.0	7.0	911.0	67.0	70
53	SH-8	23 SEP	972.0	59.0	66.0		66.0		59.0	7.0	913.0	66.0	70
54	SH-9	23 SEP	972.0	61.0	68.0		68.0		61.0	7.0	911.0	68.0	70
55	CH-9	23 SEP	973.0	62.0	69.0		69.0		62.0	7.0	911.0	69.0	70
56	NH-9	23 SEP	972.0	62.0	68.0		68.0		62.0	6.0	910.0	68.0	70
57	NH-7	23 SEP	969.0	58.0	64.0		64.0		58.0	6.0	911.0	64.0	70
58	CH-7	23 SEP	970.0	59.0	66.0		66.0		59.0	7.0	911.0	66.0	66
59	SH-7	23 SEP	970.0	57.0	65.0		65.0		57.0	8.0	913.0	65.0	65
60	SH-6	23 SEP	967.0	55.0	62.0		62.0		55.0	7.0	912.0	62.0	65
61	SH-5	23 SEP	965.0	52.0	57.0		57.0		52.0	5.0	913.0	57.0	60
62	CH-5	23 SEP	965.0	52.0	57.0		57.0		52.0	5.0	913.0	57.0	60
63	NH-5	23 SEP	965.0	53.0	58.0		58.0		53.0	5.0	912.0	58.0	60
64	CH-4	24 SEP	959.0					10.0				10.0	20
65	SH-3	24 SEP	955.0	38.0	45.0		45.0		38.0	7.0	917.0	45.0	50
66	CH-3	24 SEP	955.0	38.0	45.0		45.0		38.0	7.0	917.0	45.0	50
67	NH-3	24 SEP	956.0	40.5	47.0		47.0		40.5	6.5	915.5	47.0	50
68	CH-1	24 SEP	947.0	35.0	42.0		42.0		35.0	7.0	912.0	42.0	45
69	SH-1	24 SEP	951.0	32.5	38.0		38.0		32.5	5.5	918.5	38.0	40
70	SH-0	24 SEP	946.0	31.0	50.0	50.0			31.0	19.0	915.0	50.0	50
71	S-1-1	24 SEP	971.0	56.0	59.0		59.0		56.0	3.0	915.0	59.0	60
72	S-1-2	24 SEP	969.0	54.0	56.0		56.0		54.0	2.0	914.0	56.0	60
73	S-2-2	24 SEP	967.0					47.0				47.0	50
74	S-1-3	24 SEP	965.0	48.0	60.0	60.0			48.0	12.0	917.0	60.0	60
75	S-2-3	24 SEP	964.0	48.5	60.0	60.0			48.5	11.5	915.5	60.0	60
76	S-2-3A	24 SEP	964.0	48.0	60.0	60.0			48.0	12.0	916.0	60.0	60
77	S-3-3	24 SEP	963.0	48.5	60.0	60.0			48.5	11.5	914.5	60.0	60
78	S-3-2.5	24 SEP	964.0	50.0	55.0		55.0		50.0	5.0	914.0	55.0	60
79	S-2-2.5	24 SEP	965.0	50.0	55.0		55.0		50.0	5.0	915.0	55.0	60
80	S-1-2.5	24 SEP	968.0	51.0	56.0		56.0		51.0	5.0	917.0	56.0	60
81	S-4-3	24 SEP	962.0	48.0	54.0		54.0		48.0	6.0	914.0	54.0	60
82	S-7-3	24 SEP	961.0	42.0	60.0	60.0			42.0	18.0	919.0	60.0	60
83	S-7-2	24 SEP	961.0	43.0	49.5		49.5		43.0	6.5	918.0	49.5	60
84	S-7-1	24 SEP	963.0	48.0	50.0		50.0		48.0	2.0	915.0	50.0	60
88	S-8-1	25 SEP	962.0	42.0	45.0		45.0		42.0	3.0	920.0	45.0	50
89	S-9-1	25 SEP	960.0	42.0	47.0		47.0		42.0	5.0	918.0	47.0	50
90	S-8-2	25 SEP	960.0	43.0	46.0		46.0		43.0	3.0	917.0	46.0	50
91	S-9-2	25 SEP	958.0	41.0	44.0		44.0		41.0	3.0	917.0	44.0	45
92	S-10-2	25 SEP	956.0	39.0	41.0		41.0		39.0	2.0	917.0	41.0	45
93	S-11-2	25 SEP	955.0	38.0	41.0		41.0		38.0	3.0	917.0	41.0	45
94	S-11-1	25 SEP	956.0	39.5	44.5		44.5		39.5	5.0	916.5	44.5	45
95	S-12-1	25 SEP	955.0	36.5	41.0		41.0		36.5	4.5	918.5	41.0	45
96	S-12-2	25 SEP	953.0	36.5	41.0		41.0		36.5	4.5	916.5	41.0	45
97	S-9-3	25 SEP	958.0	41.5	44.5		44.5		41.5	3.0	916.5	44.5	45
98	S-10-3	25 SEP	955.0	39.5	46.0		46.0		39.5	6.5	915.5	50.0	65
99	S-11-3	25 SEP	954.0	37.0	45.0		45.0		37.0	8.0	917.0	45.0	50
96	TOC = Top of Coal			BOC = Bottom of Coal		BOS = Bottom of Seam					TOTAL:	5176.5	5485

3.5. COST ESTIMATE FOR SITE SELECTION, EVALUATION AND EXPLORATION

The following is a cost estimate for the site selection, evaluation and exploration phases of this project.

3.5.1. Site Selection

2 x 4 hour meetings with Public Service Commission

- Senior Blasting Engineer
  8 hrs @ $27.00/hr = $216.00
- Senior Consultant (RPM Inc)
  8 hrs @ $22.00/hr = $176.00

3 x 1-day Field Trips to visit potential sites

- Senior Blasting Engineer
  24 hrs @ $27.00/hr = $648.00
- Senior Consultant (RPM Inc)
  24 hrs @ $22.00/hr = $528.00
- Field Engineer
  24 hrs @ $18.75/hr = $450.00

part-day meetings with ND Game & Fish Dept, including site visit

- Senior Consultant (RPM Inc)
  24 hrs @ $22.00/hr = $528.00
- Senior Blasting Engineer
  4 hrs @ $27.00/hr = $108.00

Site Selection :	BCW, Inc :	$1422.00
	+ overheads @ 50% :	$711.00
		$2133.00
	: RPM, Inc :	$1232.00
	+ overheads @ 50% :	$616.00
		$1848.00
TOTAL COST		$3981.00

3.5.2. Site Evaluation

field reconnaissance and survey work

- Senior Blasting Engineer
  18 hrs @ $27.00/hr = $486.00
- Senior Consultant (RPM Inc)
  18 hrs @ $22.00/hr = $396.00
- Field Engineer
  18 hrs @ $18.75/hr = $337.50

preparation of field maps

- Field Engineer
  12 hrs @ $18.75/hr = $225.00

exploration drilling planning work

- Senior Blasting Engineer
  8 hrs @ $27.00/hr = $216.00
- Field Engineer
  8 hrs @ $18.75/hr = $150.00

Site Evaluation :	BCW, Inc :	$1414.50
	+ overheads @ 50% :	$707.25
		$2121.75
	: RPM, Inc :	$396.00
	+ overheads @ 50% :	$198.00
		$594.00
TOTAL COST		$2715.75

3.5.3. Exploration Drilling

Layout of exploration drilling pattern

- Senior Consultant (RPM Inc)
  18 hrs @ $22.00/hr = $396.00
- Field Engineer
  18 hrs @ $18.75/hr = $337.50

Site visit with drilling contractor

- Field Engineer
  4 hrs @ $18.75/hr = $75.00

Drilling supervision (Sept 18,19,23,24,25, 1987)

- Field Engineer
  44 hrs @ $18.75/hr = $825.00

Compilation of drilling statistics for cost control and field work planning

- Field Engineer
  10 hrs @ $18.75/hr = $187.50

Exploration drilling (contractor : Moe Drilling, Inc.)

- Equipment Mobilization
  = $250.00

- Drilling cost
  5485 ft @ $0.75/ft = $4113.75

Exploration drilling :	BCW, Inc :	$1425.00
	+ overheads @ 50% :	$712.00
		$2137.50
	: RPM, Inc :	$396.00
	+ overheads @ 50% :	$198.00
		$594.00
	: Moe Drilling Inc:	$4363.75

TOTAL COST $7095.25

TOTAL COST - SITE SELECTION, EVALUATION AND EXPLORATION

..... $13,792.00

3.6. DISCUSSION OF SITE SELECTION, EVALUATION AND EXPLORATION COST

The total cost of this phase of the project was around $14,00. A breakdown of these costs by cost centers indicates the following:

Site selection	$3981.00	..... 29%
Site evaluation	$2715.75	..... 20%
Exploration	$7095.25	..... 51%

The use of these figures to estimate the cost of similar functions in future work of this kind is recommended only as a rough guide. Each AML site has different characteristics - some may be much less regular, with respect to mining method, and if good maps are not available will require much more exploration work.

As stated earlier, the cost of site evaluation and exploration work would have been much greater for this project were it not for the quality of available map information, from previous survey work carried out by RPM Inc, and old mine maps from the operating company. The regular room-and-pillar mine plan adopted also helped considerably in reducing the exploration drilling requirement. The drilling cost, of $0.75 per foot, was also lower than will be found in many cases, or for work in areas where costs are higher.

In future cases organizations such as the Public Service Commission, or a state AML agency, would probably decide on a priority site, and contract an engineering firm to carry out blasting work. This firm would need to study the site, and any available information about it to assess the overall feasibility of blasting as an effective AML reclamation method at that site. However, it may be assumed that permits regarding access and work on the site would be handled by the agency contracting out the work.

Thus in work of this type which is not research oriented, it is probable that the contract cost of site selection would not be as high as 30%. However, the concern shown by the general public about blasting operations in general may require the presence of the responsible blast engineer at some meetings, possibly public meetings, in the case of sites located in populated or sensitive areas.

As a very broad guideline, the breakdown of costs in a "contracted" AML blasting project may be something like the following:

Site selection	..... 10%
Site evaluation	..... 30%
Exploration	..... 60%

For a site with reasonable map information and a regular mining method, the total cost of this phase may be in the order of $10,000 to $15,000 for a ten acre site. In cases where a larger amount of exploration drilling is required, or drilling costs are significantly higher, this may increase to $20,000 to $25,000 for a site of this size. As larger sites are considered, the selection and evaluation cost will not increase in direct proportion to site size. However, as more drilling footage will be involved, the cost of exploration will be higher for larger sites.

4. TEST BLAST DESIGN AND SUMMARY

4.1. INTRODUCTION

In this chapter the design and implementation of the test blast program is discussed. Test blasts involved six acres of a ten-acre site. The program was intended to be large enough so that the probability of success with caving old works could reasonably be ascertained and so any problems with blasting in the overburden above such works would come to light.

The first step was to design each blast. While the same principles applied and the layout was generally the same adjustments had to be made to conform with the specific nature of each blast. This was particularly true regarding the explosive loading. Changes were made to account for the amount of roof coal left behind, the hole depth, and any rock layers reported by the drillers on their hole logs.

Standard loading data was developed for each hole depth, for a given diameter, and was used wherever possible. This data shows the column locations where explosives and deck stemming should be placed. The information was placed on loading boards for ease of use in the field. Substitute loading boards were used on any specific blast where conditions indicated a change. The development of standard loading data is discussed in detail below.

The blasts were located so as to test the method under several conditions. Blasts were conducted in cover ranging from 35 to 65 feet. Long blasts (up to 200 feet) were detonated as were shorter blasts (less than 100 feet). Twenty-two foot wide rooms were blasted and twelve-foot wide panel entries were also shot. All of this was done to determine the effect of various conditions on the technical and cost feasibility of the reclamation method.

There were twenty-one blasts in all. The majority were designed to cave rooms or entries. Two were designed to close individual sink holes. One of these was performed in conjunction with a room blast; the other was shot separately. The blasts were detonated using both ANFO and slurried explosives. Some blasts were loaded with all ANFO but most had a slurry in the bottom one or two decks.

Since all the test work was performed at one site the results were obtained for one type of strata sequence. The overburden above the coal seams in the Beulah area is a combination of clays and glacial tills. The clays trend from brown near the surface to grey in depth. These clays are overconsolidated and in many cases have the thinly bedded appearance of shales. However they are not sufficiently

consolidated to be classified as shales as such.

Periodically there occurred interbedded layers of weak sandstone, which usually did not exceed four feet in thickness. This sandstone was in the form of lenses which were not continuous over the project test site. The presence of such layers was noted during drilling and was recorded in logs so that design changes, related to deck location, could be made if required.

Strength testing of the strata was not carried out. However, general experience in western North Dakota would indicate that no strata had unconfined compressive strengths of more than 5000 psi. Figure 4-1 illustrates a lens of cross-bedded sandstone underlying clay in the side of one of the sinkholes at the test site.

4.2. BLAST DESIGN

4.2.1. Explosive Charge Placement

The method used to design the blasts is discussed in this section. The design procedure followed the general cratering theory techniques discussed in Chapter 2. Variations to the plan were made, as the program progressed, based on observations of the blasted areas. These variations are largely discussed in the results chapter.

It was necessary to estimate an optimum scaled depth of burial for the individual cratering charges in the clays and clay shales observed at the test site. From previous work it was known that the optimum scaled depth of burial in unfrozen tar sand was 3.1. Tar sand was considered to be a material similar in nature to that seen at the test area. Table 4-1 lists parameters for crater blasting in tar sand.

TABLE 4-1 PARAMETERS USED IN CRATER BLAST DESIGN IN UNFROZEN TAR SAND

Hole Diameter	30-inch (augered)
Hole Depth	55 feet
Explosive Type	ANFO
Explosive Weight	3,500 lbs.
Scaled Optimum Depth of Burial	3.1 ($d/W^{1/3}$)
Stemming Height	40 feet
Spacing	70 feet by 70 feet
Delay Arrangement	100 ms between holes

FIG. 4-1: CROSS-BEDDED SANDSTONE LAYER EXPOSED IN THE WALL OF A SINKHOLE AT THE BEULAH AML TEST SITE

It was felt that the experience in tar sand was applicable to the material existing at the test site. However, the AML case where a series of crater charges are successively detonated in the hole is more confined than that of a single cratering charge in tar sand. Therefore, it was decided to reduce the scaled depth of burial from 3.1 $ft/lb^{1/3}$ to 2.5 $ft/lb^{1/3}$ at least for the initial blasts. Each deck was initially designed so that the distance from the base of the previous deck charge to the center of the charge in question was scaled at a $d/W^{1/3}$ of 2.5 $ft/lb^{1/3}$. In the case of the charge directly above the opening the depth of burial was from the roof of the void to the center of the charge.

The maximum length of the cratering charge equals eight times the diameter of the blast hole. For a six-inch hole the length of the charge is 48 inches. The typical density of free poured ANFO is 0.85 gm/cc although some variation in density may be seen between manufacturers. The weight per foot of ANFO in a 6-inch holes is 10.4 pounds. Therefore four feet of charge contains 41.6 pounds of ANFO.

For a scaled depth of burial of 2.5 $ft/lbs^{1/3}$ the depth of burial can be computed according to the following expression:

$$d = SW^{1/3} = 2.5\ W^{1/3}$$

$$d = 8.6 \text{ feet}$$

The depth of burial is taken to the center of the charge.

To determine the deck stemming height four charge diameters must be subtracted from d. Therefore the stemming height in this case would be 6.6 feet. Since the depth of burial, d, is to the center of the charge the total charge length is 4 feet (8 times the diameter).

Some adjustments had to be made for individual blasts to account for specific situations. Changes included variations to the explosives column length and the deck stemming. These were to account for hard bands in the overburden, thick roof coal left behind during mining and fitting the number of decks into the given hole length. These changes and results are discussed in a later chapter.

Using this design one can examine the effect of different hole diameters on the depth of burial. For example, an 8-inch diameter hole loads ANFO at 18.5 pounds per foot when the free poured density is 0.85 gm/cc. Eight diameters of an 8-inch hole is 64 inches. Therefore, the total weight in the cratering charge is 98.7 lbs. The depth of burial is 11.6 feet and the deck stemming height is 8.9 feet. These increases relate to the square of the increase in the radius. The important point is that larger hole diameters will reduce the number of decks

needed to fully cave the void. This can be important in deeper cover because it helps to maintain a reasonable number of decks and reduces the chances for out-of-rotation firing and cutoffs.

The placement of the charges in the hole then was a matter of getting the explosive plus stemming lengths to match the hole depth. The final design requirement, in order to achieve a total hole design, was to determine how much stemming should be used between the uppermost explosive deck and the ground surface. It is important to heave and break up the upper surface. However, excessive flyrock due to uncontrolled top movement is to be avoided. These competing requirements can generally be well controlled in soft plastic rocks. Where brittle, hard rocks form the upper surface it is considerably more difficult to control flyrock while achieving good top breakage.

For soft materials good top breakage can often be obtained with scaled depths of burial of 4.0 or more. However, in this case it was thought that some of the energy of the detonation of the upper deck would be directed downward so a more conservative scaled depth of burial should be selected. Initially a scaled depth of burial of 3.1 $ft/lbs^{1/3}$ was selected and the stemming height was 8.5 feet in 6-inch diameter holes. This was subsequently increased to 9.0 feet and then 9.5 feet of stemming and corresponds to an average scaled depth of burial of 3.25 $ft/lbs^{1/3}$.

For 8-inch diameter holes the stemming heights were designed at 13.0 feet. This resulted from a scaled depth of burial of 3.5 $ft/lb^{1/3}$.

Table 4-2 shows the loading arrangement for one blast. Figure 4-2 is a cross section of a 56 foot deep blasthole showing the explosive and stemming decks.

4.2.2. Hole Spacings

Another important matter of design was to determine the spacing between the holes on the blast line. Often a spacing equal to 1.5 times the depth of burial, d, is found to be optimum. For the tar sands blasts in table 4-1, the center of the charge is 7.5 feet above the bottom of the hole or 48.5 feet from surface. The 70 x 70 foot spacing is then 1.48 times the depth of burial.

For the caving of old works it was thought that a spacing greater than 1.5 times the depth of burial might be acceptable because the area being cratered was undercut by the mining. Therefore, the first blast was designed with 17-foot spacings which are 2.0 times the depth of burial of 8.6 feet for a 6-inch hole diameter. This design was used in an attempt to find the limit of spacing. The first blast, located on room N-14, was a single row blast. The line of holes was placed on the center

HOLE DEPTH : 56 FT.

DEPTHS	STEM HT (FT)	CHARGE LENGTH (FT)	CHARGE WEIGHT ANFO(LBS)	CHARGE WEIGHT SLUR(LBS)	SCALED DOB(ANFO) (FT/LB^3)	SCALED DOB(SLUR) (FT/LB^3)
0.0						
	9.5				3.217	3.048
9.5						
		5.0	52.095	61.288	2.413	2.286
14.5						
	6.5					
21.0						
		5.0	52.095	61.288	2.413	2.286
26.0						
	6.5					
32.5						
		5.0	52.095	61.288	2.413	2.286
37.5						
	6.5					
44.0						
		5.5	57.305	67.417	2.403	2.276
49.5						
	6.5					
56.0						
TOTAL		20.5	213.6	251.3		

TABLE 4-2: CHARGE AND STEMMING POSITIONS IN A 56-FOOT, 6-INCH DIAMETER BLASTHOLE

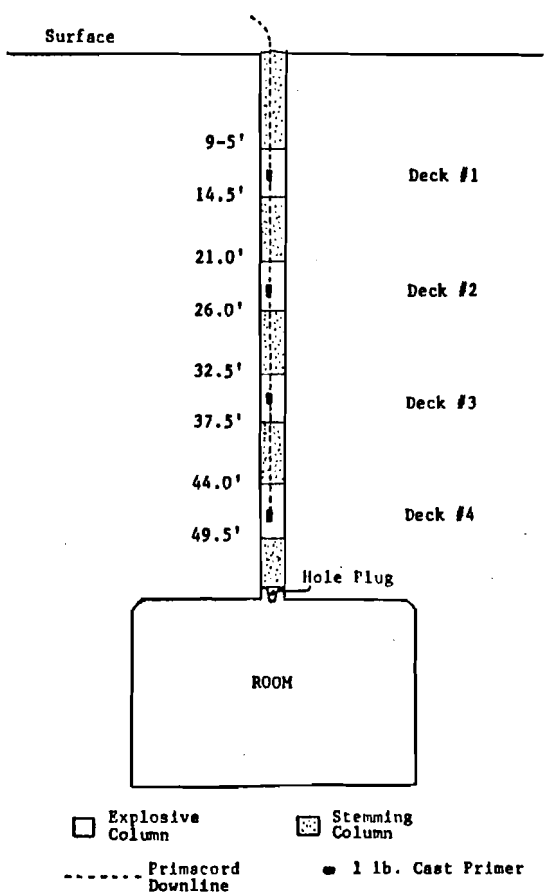

FIG. 4-2: CROSS-SECTION OF 56-FOOT SIX-INCH DIAMETER BLASTHOLE SHOWING CHARGE AND STEMMING PLACEMENT

axis of the room approximately 11 feet from each pillar.

The blast caved the room except for an area extending from the panel entry at the south end of the blast about 65 feet along the room. This area appeared to bridge. The remainder of the blast caved the room, as evidenced by a surface depression of considerable depth. The blast did not, however, appear to break across the full width of the room.

For these reasons it was decided to adjust the spacings and use two staggered rows of blast holes to obtain better caving. The next blast therefore was drilled with holes spaced 13 feet apart on two lines spaced 8 feet apart. The distance from a pillar to a line of holes was 7 feet. This is shown in figure 4-3. The spacing along the line was 1.5 times the depth of burial. The spacing between the rows and the spacing between the rows and the pillars were approximately 1.0 times the depth of burial. This resulted in large part from the dimensions of the room and the need to fit the two rows of holes into this geometry.

Later in the test program, based on field observation the spacings between holes on a row were increased to 15 feet or approximately 1.75 times the depth of burial. Near the end of the test blasts a single row was again tried. In view of the initial experience with a single row of holes it was decided to use spacings less than 1.5 times the depth of burial. The spacings were 12, feet or 1.35 times the depth of burial. Comments on the effectiveness of experimentation with blasthole spacing are made in Chapter 8 of this report.

4.2.3. Standard Loading Scales

In order that explosive loading operations could proceed rapidly and without error it was necessary to develop a system of loading data that would be as consistent as possible and easy to refer to in the field. For this reason loading schedules were produced, based on the hole diameter, and the explosive used so that the loading crew could tell where and how much explosive and deck stemming were required. The basic calculations for the location of the explosive decks and stemming are as discussed above.

The information for a given hole depth was written on a board for use in the field. This showed the distances down the hole at which charge and stemming interfaces occur, together with the position of the explosive. As it was possible to standardize loading for holes of a given depth, a series of "loading boards" were made up for use in the field. One of these is illustrated in Fig. 4-4. These boards were placed next to blastholes with the appropriate depths, as indicated by the measurement recorded on the wooden stake, for use during hole loading (see Fig. 4-5).

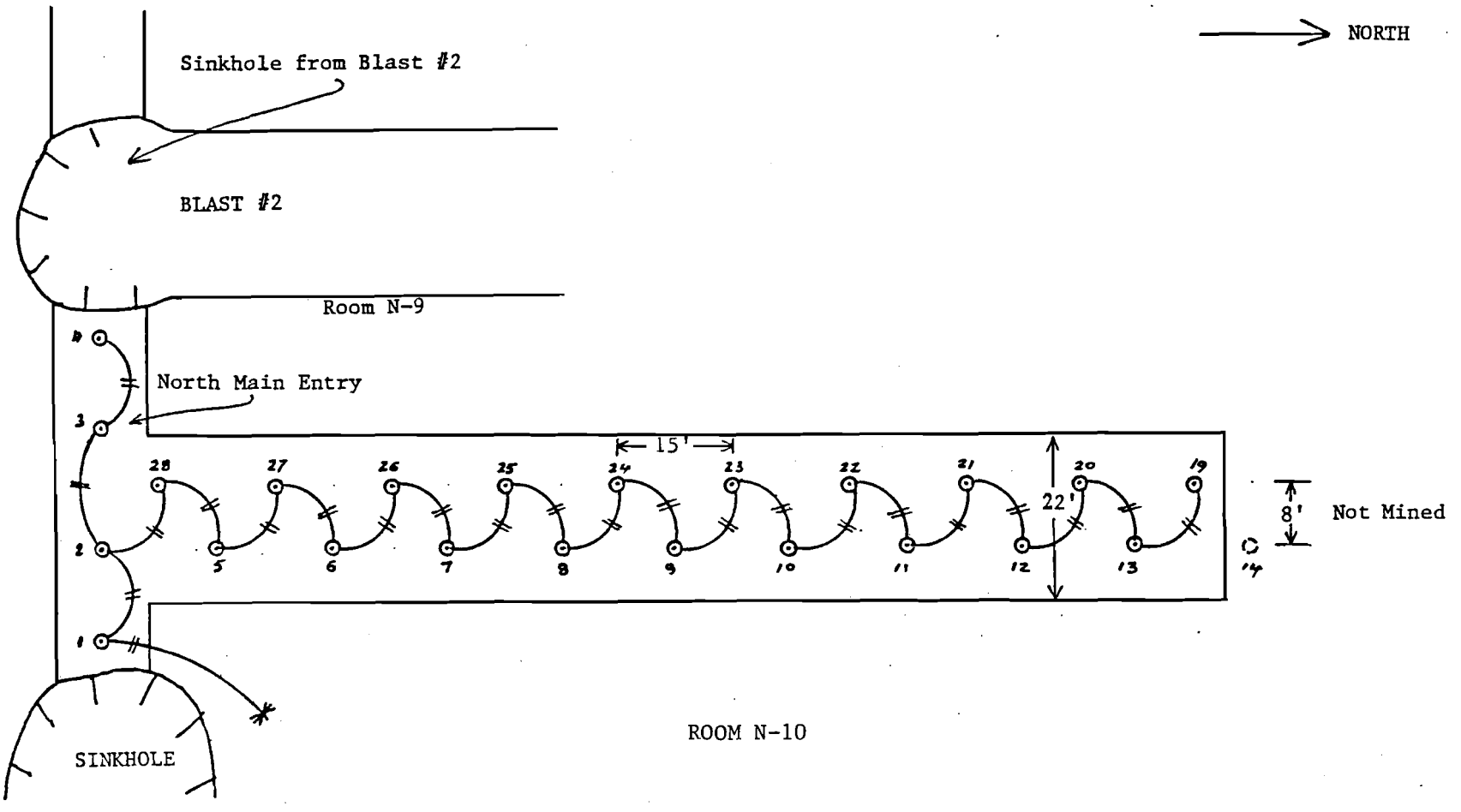

Fig. 4-3: TYPICAL BLAST LAYOUT USING DOUBLE ROW OF BLASTHOLES

LEGEND
⊙ 6" Blasthole
42ms Nonel Surface Delay
Point of Initiation

BAUER, CALDER & WORKMAN, INC.	
A M L P R O J E C T BLAST LAYOUT SKETCH	
Blast # 13	Date: 10/30/86
Scale: 1:20	

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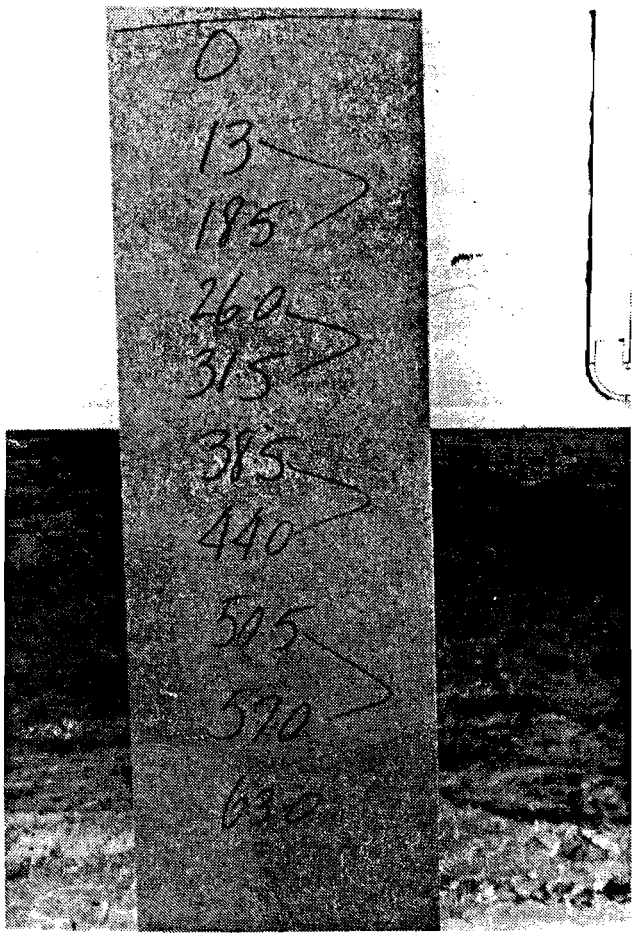


FIG. 4-4: ILLUSTRATION OF TYPICAL LOADING BOARD FOR USE IN FIELD BLASTING OPERATIONS

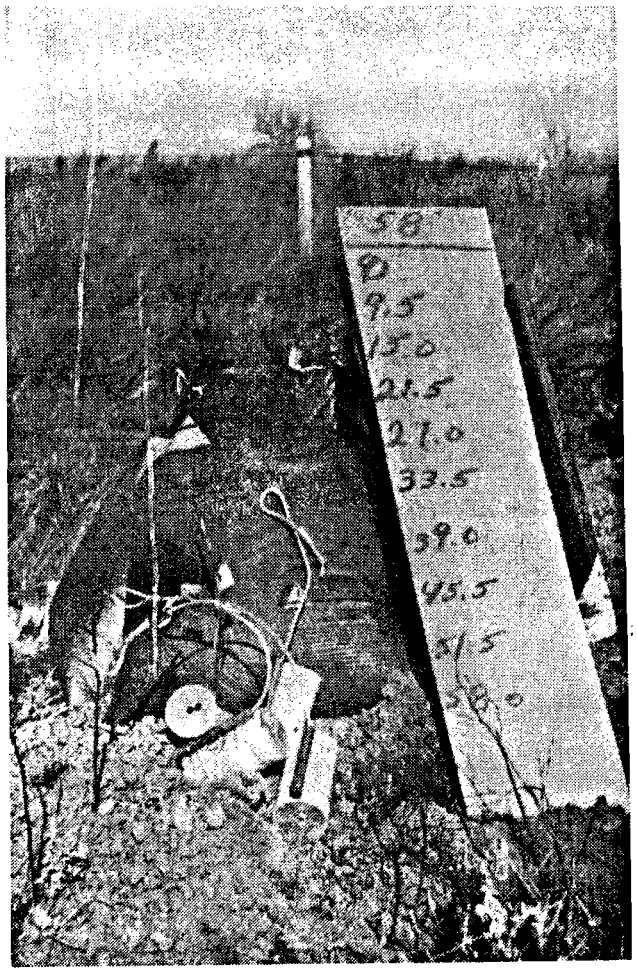


FIG. 4-5: LOADING BOARD PLACED ALONGSIDE BLASTHOLE IN THE FIELD PRIOR TO LOADING OPERATIONS

The standard loading boards were reviewed for each blast. Where necessary changes were made and a new board with the revised information was drawn-up.

The primary reason for change to a standard loading board was the occurrence of rock layers in the overburden. Although the overburden was primarily over-consolidated clays there were intermittent lenses of rock that were only a few feet thick. It was important that these layers were well fragmented so that large chunks would not bridge over and prevent the remaining overburden from falling toward the void.

To insure fragmentation of these layers required knowing the location of the rock in the overburden. For this reason the drill crew was required to carefully record the rock layers on their drillers logs. With this information available adjustments could be made.

In some cases one or more of the explosive decks was lengthened to bring explosive into closer proximity to the rock. Another alternative was to relocate the deck to be better positioned with respect to the hard layer. This could be done if the change needed was small. A large relocation of the deck would alter the overall cratering characteristics too much and good performance could not be expected. In some cases altering both the length of the explosive column and the location of the deck was required.

Another reason for changing the loading boards for a specific blast had to do with the amount of roof coal left behind. Generally the roof coal amounted to three or four feet. However, as much as eight feet were left in some cases. This occurred most frequently in the panel entries and in the portion of the rooms next to the entries. When the coal was thick the scaled distance was adjusted downward and the charge was placed closer to the coal layers. Adjusted loading boards were then marked up for the given blast.

The boards, then, were developed based on the cratering principles discussed above. However, in field operation it was not always possible to follow exactly the results from the standard calculations. In addition to the need to cater for thin rock layers in the overburden and thick roof coal one also had to make adjustments for the varying hole depths.

For a given explosive and hole diameter combination there is only one hole length that will exactly match the requirements of the optimum cratering action. Other hole depths will require at least slight modification to the powder location to insure a proper load in the hole. This may mean lengthening the explosive deck columns until a hole depth is reached whereby an additional deck may be added.

Once the next deck is added the explosive columns may be

less than eight diameters in length initially and will again be lengthened as the hole depth increases. One must be careful however not to vary the charge dimensions and location too much relative to the initial design or the cratering action will lose its effectiveness. Bridging and hang-up of the overburden may occur.

Table 4-3 shows a comparison of an initial design in 6-inch holes and the variations created by varying depths of cover. The table shows how the loads were varied to accommodate the changing hole depths.

In this table deck 1 refers to the highest deck in the hole with the remaining decks in descending order. The hole with 41-foot depth is shown in terms of initial design and actual loadings. In the field four decks were used rather than three to be consistent with other holes in the blast. Less explosive and reduced depth of burial were employed.

For the remaining holes the table shows the variations necessary to account for changing depths. Experimentation during the test program showed that four decks could be used for depths up to about 60 feet. Beyond that either an 8-inch hole diameter was needed to maintain four decks or a fifth deck must be added in the 6-inch diameter holes. In any event as loading boards were prepared, and checked for each blast, the effect of hole depth was taken into account.

Table 4-3 illustrates that the collar stemming is not used to adjust the powder locations for changing hole depths. Once a collar location has been found that achieves good breakage without excessive flyrock, this should be used in every hole. Failure to do so will lead to variable results. Too little stemming may lead to flyrock, venting, and airblast. Changes to collar stemming should only result from changes in hole diameter, changes in explosives type, or changes in geological conditions.

4.2.4. Explosives Selection

Based on the nature of the overburden and ease of use it was believed that ANFO would be the most suitable explosive for the bulk of AML work. For this reason the most common explosive used in the test program was ANFO. This product was obtained in standard fifty pound bags. During loading the bags were opened and the ANFO was poured into the hole in order to fully couple the hole and maximize the explosive load in the required crater charge geometry.

ANFO has an energy output of 890 calories per gram. This is quite adequate for fragmenting the types of strata found in the overburden at the test site. ANFO also produces large gas volumes upon detonation and therefore maintains borehole

TABLE 4-3 : COMPARISON OF BASIC BLAST DESIGN IN SIX-INCH DIAMETER BLASTHOLES, AND ACTUAL LOADING FOR VARIOUS HOLE DEPTHS

			DECK 1		DECK 2		DECK 3		DECK 4	
Hole Type	Hole Depth (ft)	Collar Stemming (ft)	Explosive Column (ft)	Stemming (ft)	Explosive Column (ft)	Stemming (ft)	Explosive Column (ft)	Stemming (ft)	Explosive Column (ft)	Stemming (ft)
Initial Design	41	9.5	—	—	4.0	6.5	4.0	6.5	4.0	6.5
Actual Load	41	9.5	3.0	4.0	3.0	5.5	3.0	5.0	3.0	5.0
Actual Load	42	9.5	3.0	4.0	3.0	5.5	3.0	5.0	4.0	5.0
Actual Load	43	9.5	3.0	4.0	3.0	5.5	4.0	5.0	4.0	5.0
Actual Load	44	9.5	3.0	4.0	3.0	5.5	4.0	6.0	4.0	5.0
Actual Load	45	9.5	3.0	4.0	3.0	5.5	4.0	6.0	4.0	6.0

pressures at a high level for substantial periods of time. This is important when blasting in soft, plastic materials where considerable energy is absorbed before movement and fragmentation start to take place.

Normally the energy outputs of blasting agents are stated as a relative weight strength and a relative bulk strength. The weight strength is the energy output per unit of weight; the bulk strength is the energy output per unit of volume. Usually these strengths are taken relative to ANFO. Therefore, ANFO has a relative weight and relative bulk strength of 1.00. Other products have relative strengths greater than or less than 1.00 depending on their energy output in calories per gram, and in the case of bulk strength, their densities.

The primary disadvantages to the use of ANFO are that it has virtually no water resistance and its free poured density is low (0.85 gm/cc). The water problem can be addressed by the use of polyethylene dry liners to sleeve the hole. Dry liners can be used in the holes of greater than 5-inch diameter. The density cannot easily be increased unless a blend of prill sizes is used. However, this approach is not generally feasible in using free poured product because of caking problems.

To improve the density and therefore increase the weight per foot that can be loaded one may select other products including heavy ANFO, emulsions and slurries. All of these products can be produced at densities greater than 1.00 gm/cc. These explosives are also waterproof which is an advantage if the holes are wet. However, such products are considerably more expensive than ANFO.

For the field testing a slurried product was used in the lowest deck in the hole. The reasons were that it was thought that if water were present it would be in the bottom area of the hole and a greater loading density was expected which would be helpful with regard to the depths of burial and the roof coal left in place. In fact the loading densities were not as high as expected; this is discussed in more detail later in this report.

The first slurry explosive used was IREGEL 140, a water -based gelled product with a nominal density of 1.20 gm/cc. This product was viscose but was more or less pourable. In order to insure full coupling to the hole the product was freed from the bags and allowed to fall down the hole. Another reason for freeing it from the bag was that the 5-inch diameter bags would not always fall freely in a 6-inch diameter hole. Finally it was thought that removing the explosive from the bag would maximize the loading density in the deck.

Subsequently a different product called Energel 500 with a nominal density of 1.15 gm/cc was used. This product included substantial amounts of prill in the mix. It was much stiffer than the first product and could be sliced into chunks and

dropped down the hole.

In a few blasts ANFO alone was used. The intent was to determine the relative effect of using a slurry in the lower deck or of having no slurry. Except for the change of explosives these blasts were designed in the same manner as others. Table 4-4 lists the properties for the explosives as supplied by the manufacturers.

All of the explosives were delivered in bags. The ANFO was in fifty pound bags. The Iregel 140 and Energel 500 were in 5-inch diameter, thirty pound bags.

TABLE 4-4 PROPERTIES OF EXPLOSIVES USED IN THE FIELD TESTING

Explosive	Density gm/cc	Weight Strength	Bulk Strength	Velocity of Detonation ft/sec	Water Resistance
ANFO	0.85	1.00	1.00	12,500	Poor
Iregel 140	1.20	.96	1.40	13,800	Good
Energel 500	1.15	.86	1.31	16,400	Good > 18 hrs

4.2.5. Millisecond Delays

The need for millisecond delays was discussed earlier in the report. Both surface and down-the-hole delays were used to cause each deck in each hole to detonate independently of all the others. In this manner the overburden surrounding the blastholes was displaced toward the void below as a series of cratering charges detonated from the bottom to the top of the hole.

The primary question was which delays to select. The important considerations in this regard were providing sufficient delay time to allow each deck to move before the next detonated and not allowing the delays to be so long that cutoffs or sympathetic detonations across the decks might occur. The down-the-hole delay system used was a slider primer and delay system supplied by Ensign-Bickford. In this case the delay consisted of the non-electric delay cap with a NONEL pigtail.

Initially successive periods were used in the hole starting with period 4 (100 ms) in the bottom deck. For a four deck hole

the remaining decks had period 5 (125 ms), 6 (150 ms), 7 (175 ms) delays. If a fifth deck was required, because the holes were deep, then period 8 (200 ms) was used. Therefore, there were 25 milliseconds between the detonation of decks within a hole.

Surface delays were used to obtain the appropriate delay between holes and insure that each deck in the blast detonated independently of all other decks. For this purpose 42 millisecond NONEL noiseless trunkline delays were used. With these surface delays in combination with the down-the-hole delays each deck fired separately at least 8 milliseconds after the previous deck. This helps to maximize the relief as the blast progressed. Also, the weight per delay was minimized which meant the ground vibration from blasting was kept to a minimum.

After observation of a series of blasts it was decided to try 50 milliseconds of delay time between decks within the hole. To do this, periods 4, 6, 8 and 9 (250 ms) were used as down-the hole delays. The 42 ms delays continued to be used on surface. It was thought that this array would lead to a greater relief within the hole and along the blast. Figures 4-6 and 4-7 show the typical delay sequence for the two cases. Note that the use of 50 millisecond delays led to a uniform diagonal detonation path along the holes and therefore more relief.

The potential drawback to the use of the longer delays between decks within the hole is the greater potential for cross propagation between the decks or cutoffs due to large movements along fracture planes within the overburden. However, in the soft, plastic rocks found at the test site these problems were not expected unless very long delay times were used between the decks.

4.3. SUMMARY OF BLASTS

The test blasting program for this project was carried out between September 25th and November 20th, 1986. After two blasts work was suspended for a week to allow time for completion of exploration and line-up drilling work. One week was lost in November due to a severe winter storm which closed access to the test site. The final 4 blasts were carried out under alternately very cold and snowfall conditions

A summary of general information pertaining to the blasts is included as Table 4-5. Twenty-one blasts were carried out on the ten acre test site. Nineteen of these were designed to cave rooms, or parts of rooms, in the north and south mining areas. Blasts were located by their room number as described in Section 3.4.2. of this report; a location map is included as Figure 4-8.

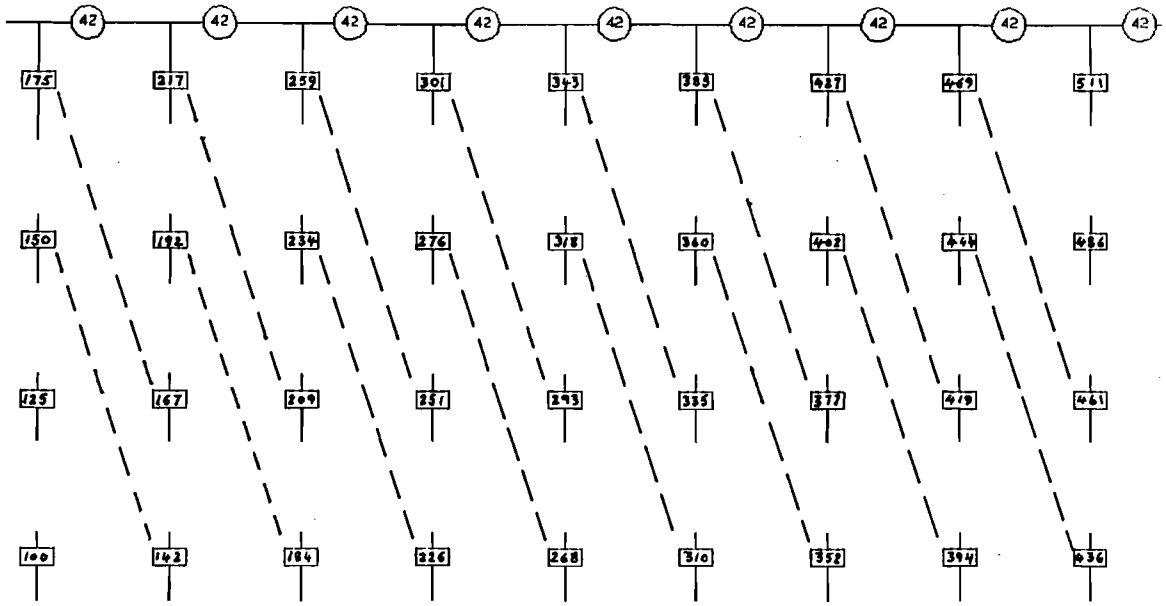


FIG: 4-6: TYPICAL DELAY SEQUENCE USING 42ms SURFACE DELAYS AND 25ms INTERVAL BETWEEN DECKS

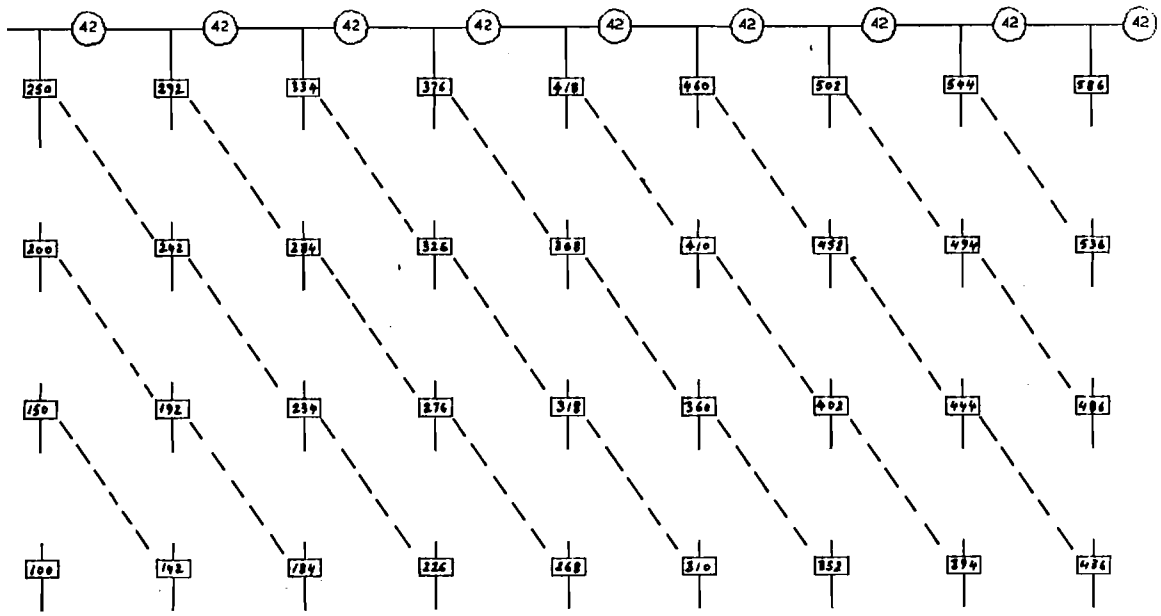


FIG. 4-7: TYPICAL DELAY SEQUENCE USING 42ms SURFACE DELAYS AND 50ms INTERVAL BETWEEN DECKS

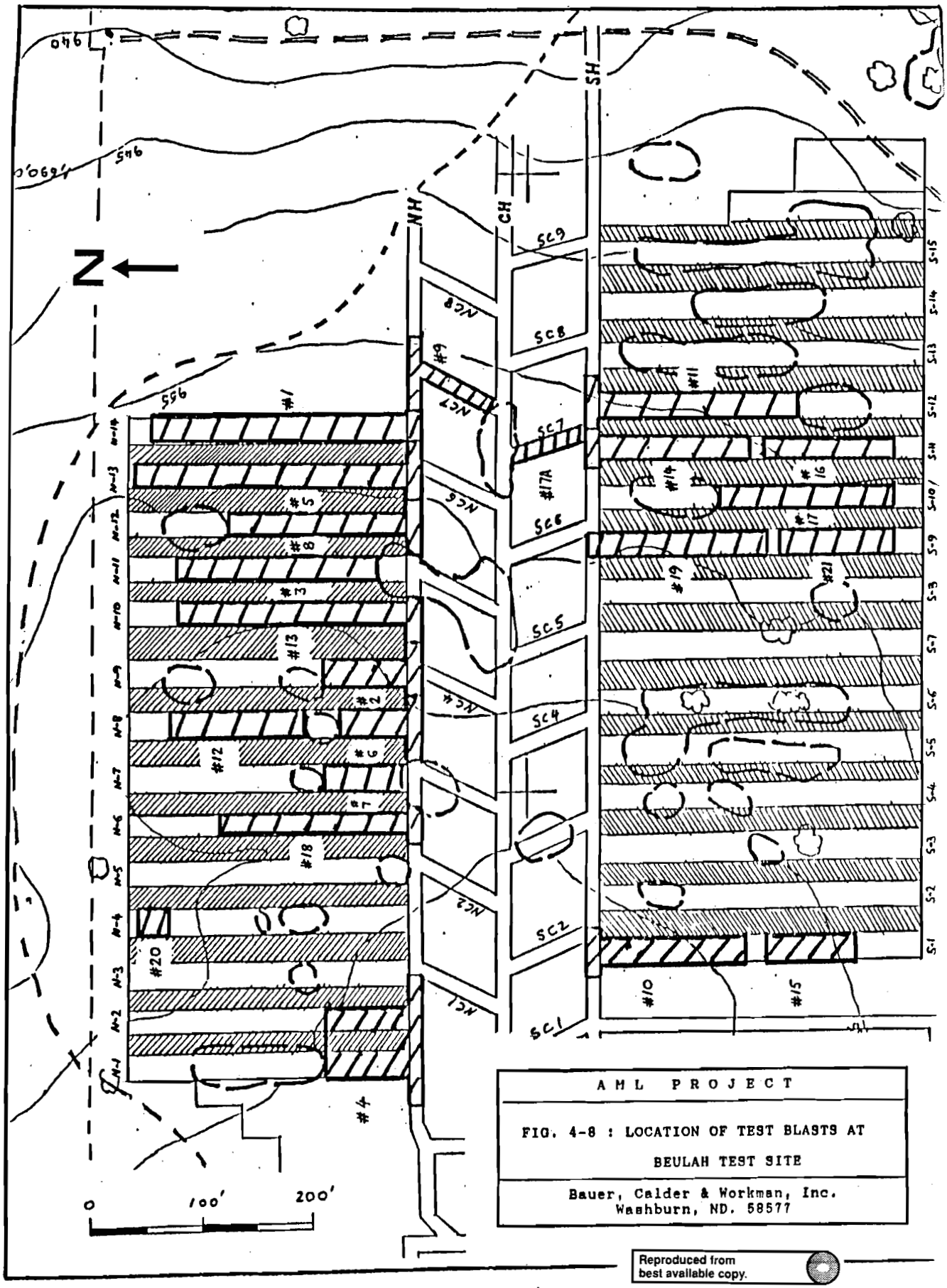
TABLE 4-5 : SUMMARY OF FIELD TEST BLASTS

Blast #	Location	Date	Hole Φ	# Holes	Length	# Rows	Hole Spacing	Aver. Depth	Expl. Types	Delay #'s
1	N-14	9/25	6 "	10	165'	1	17'	46	AN	4567
2	N-9	9/26	6 "	10	70'	2	13 x 8	58	AN	45678
3	N-11	10/7	6 "	20	135'	2	13 x 8	55	AN,IR	4567
4	N-1,N-2	10/9	8 "	22	60'	2	16 x10	64	AN,IR	4567
5	N-13	10/13	6 "	30	200'	2	13 x 8	49	AN,IR	4567
6	N-8(S)	10/14	6 "	10	55'	2	13 x 8	57	AN,IR	4567
7	N-7	10/16	6 "	19	70'	2	13 x 8	56	AN,IR	4567
8	N-12	10/17	6 "	21	145'	2	15 x 8	52	AN,IR	4567
9	NC-7	10/20	6 "	12	120'	1	12'	45	AN,IR	4567
10	S-1(N)	10/21	6 "	26	135'	2	13 x 8	62	AN,IR	45678
11	S-12	10/24	6 "	31	190'	2	15 x 8	41.5	AN	5678
12	N-8(N)	10/28	6 "	17	130'	2	15 x 8	57	AN	5678
13	N-10	10/30	6 "	23	155'	2	15 x 8	56	AN	4567
14	S-11(N)	10/31	6 "	23	160'	2	15 x 8	44	AN	4689
15	S-1(S)	11/3	6 "	16	105'	2	13 x 8	55	AN,EN	4689
16	S-11(S)	11/5	6 "	15	115'	2	15 x 8	41	AN,EN	4689
17	S10,SC7	11/6	6 "	26	150'	2,1	15 x 8	43	AN,EN	4689
18	N-6	11/17	8 "	12	145'	1	15'	57	AN,EN	4689
19	S-9(N)	11/18	6 "	20	155'	2	15 x 8	46.5	AN,EN	4689
20	N-4 *	11/19	6 "	10	n/a	*	10 x 5	20	AN,EN	45
21	S-9(S)	11/20	6 "	11	130'	1	12'	46	AN,EN	4689

* = sinkhole blast only

AN = ANFO
IR = Iregel (slurry)
EN = Energel (slurry)

Delay Number	Millisecond delay period
4	100
5	125
6	150
7	175
8	200
9	250



AML PROJECT

FIG. 4-8 : LOCATION OF TEST BLASTS AT BEULAH TEST SITE

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Washburn, ND. 58577

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One blast, #17, was primarily to collapse a room, but also incorporated an access cross-cut between two panel entries. Blast #7 included an attempt to fill adjacent sinkholes; blast #20 was carried out exclusively to fill a single sinkhole. Blast #9 was carried out exclusively in access development (a cross-cut and part of a panel entry).

Two blasts, #'s 4 and 18, employed eight inch diameter holes, the remainder of the work was carried out using six inch blastholes. Five blasts included use of a central, single row of blastholes; the remainder employed two rows. Overburden depths were, on average, in the range of 41 to 64 feet.

A total of 51,250 pounds of ANFO explosive and 18,030 pounds of slurry were used during the project. The slurry explosive was generally located, when used, in the bottom decked charge only. Actual explosives consumptions for each blast are included in Table 4-6. This table also shows consumption of surface and down-the-hole delays, and some information regarding use of the high-speed camera and engineering seismograph.

Two blasts, #'s 2 and 10, involved use of five explosive decks, due to high overburden cover. The remainder of the blasts employed 3 or 4 decks. Two decks only were employed in blasts which were used to fill sinkholes. For most of the blasts there was a 25 millisecond delay period between the initiation of explosive decks. From blast #14 onwards, however, the delay period between decks was increased to 50 milliseconds as part of the experimentation with blast design.

Approximately half of the two-row blasts used a spacing of 13 feet on rows 8 feet apart. The hole spacing was subsequently increased to 15 feet at the same row separation.

Access development was generally shot at 12 foot spacings. In the latter part of the program this spacing was reduced to 9 feet for portions of panel entries which were blasted immediately adjacent to rooms. Hole spacings for 8 inch holes were higher than for blasts of a similar type using 6 inch blastholes.

The blasts were variable in length, but in general were in the range of 120-160 feet long. Blast length was controlled mainly by the presence of sinkholes - where possible, the blast was laid out to collapse the room over its entire length. Three short blasts (less than 100 feet long) were taken in rooms. The 120 feet indicated for blast #9 includes 60 feet of cross-cut and 60 feet of adjoining panel entry. Blast #17 collapsed 50 feet of cross-cut in addition to the 150 feet of room S-10.

Three blasts in the south of the test area were deliberately split into two sections for operational or experimental reasons. One room in the north of the test area was blasted in two separate parts due to the presence of a sinkhole.

AML PROJECT - BLAST SUMMARY

#	DATE	LOCATIONS	HOLE DIAM	NO. OF HOLES	EXPLOSIVES USED: AN FO BAGS	AN FO LBS	SLURRY BAGS	SLURRY LBS	BOOSTERS	DELAYS USED: 42 MS	#4	#5	#6	#7	#8	#9	HIGH SPEED FILM: CAMERA	DELAY #	FPS	SEISMOGRAPH: SEISMOS	RPPV
1	SEPT 25	N-14	6"	10	26.5	1325			41	10	10	10	10	10			-	-	-	1450' SW	0.03
2	SEPT 26	N-9	6"	10	36.0	1800			47	10	10	10	10	10	7		-	-	-	1300' SW	0.03
3	OCT 7	N-11	6"	20	58.0	2900	31	930	81	20	21	20	20	20			-	-	-	1375' SW	0.06
4	OCT 9	N-1,N-2	8"	22	114.5	5725	88	2640	88	23	22	22	22	22			-	-	-	1150' SW	0.23
5	OCT 13	N-13	6"	30	60.0	3000	60	1800	120	30	31	30	29	30			SC-4	-	500	400' E	-
6	OCT 14	N-8(S)	6"	10	29.0	1450	23	690	38	10	10	10	9	9			W. OF N-1	7	500	675' SE	0.1
7	OCT 16	N-7	6"	19	31.0	1550	16	480	48	19	9	7	11	21			W. OF N-1		200	425' S	0.24
8	OCT 17	N-12	6"	21	48.0	2400	40	1200	85	21	21	22	21	21			E. OF N-14	4	500	450' E	-
9	OCT 20	NC-7	6"	12	23.0	1150	21	630	44	12	9	11	12	12			SC-4		500	400' N	0.31
10	OCT 21	S-1(N)	6"	26	85.0	4250	55	1650	109	26	22	22	22	26	17		W. BLAST	-	400	400' W	0.28
11	OCT 24	S-12	6"	31	69.0	3450			102	31		19	27	28	28		S-2	6	500	300' N	0.75
12	OCT 28	N-8(N)	6"	17	64.0	3200			66	17		16	16	17	17		W. OF N-1	8	500	575' E	0.14
13	OCT 30	N-10	6"	23	104.0	5200			92	23	23	23	23	23			E. OF N-14	7	300	640' W	0.1
14	OCT 31	S-11(N)	6"	23	67.0	3350			92	23	23		23		23	23	N OF S-1	9	500	550' N	0.11
15	NOV 3	S-1(S)	6"	16	29.0	1450	59	1770	65	16	21		14		15	15	S. OF S-1	9	200	700' E	0.2
16	NOV 5	S-11(S)	6"	15	28.0	1400	24	720	54	15	15		15		15	9	TRAILER	4	500	425' W	0.33
17	NOV 6	S-10,SC-7	6"	26	44.0	2200	41	1230	88	26	25	4	21		19	19	TRAILER	6	500	1400' W	0.03
18	NOV 17	N-6	8"	12	59.0	2950	44	1320	45	12	12		12		11	10	-	-	-	1100' SW	?
19	NOV 18	S-9(N)	6"	20	26.0	1300	54	1620	82	20	23		20		20	19	-	-	-	1200' SW	0.1
20	NOV 19	N-4	6"	10	9.0	450	14	420	19	10	9	9		1			-	-	-	-	-
21	NOV 20	S-9(S)	6"	11	15	750	31	930	41	11	8		10		11	11	TRAILER	8	500	300' E	0.08
TOT				384	1025	51250	601	18030	1447	385	324	235	347	250	183	106					

TABLE 4-6: SUMMARY OF TEST BLAST DATA AND EXPLOSIVES CONSUMPTION

4.3.1. Description of Blasts

General points of interest about each blast in the test program are described briefly in the following sections. Most of the relevant characteristics are summarized in Tables 4-5 and 4-6. A sketch of each blast, plotted at 1:20 scale, is included in Appendix A of this report. These sketches show the numbering and location of blastholes, the surface tie-in, point of initiation, relevant dimensions, and proximity to other blasts and existing sinkholes at the site.

Field measurement summary sheets, which are described in a later chapter, are also included in Appendix B. These are essentially for the record, with most of the relevant information contained therein summarized in tables elsewhere in this report.

A complete set of photographs showing the post-blast profile of each blast was taken in the Spring of 1987. Some of these are used in the following description to illustrate some typical results. They are also employed to show certain interesting or less successful results

Conclusions about the general success of the test blast program, and a discussion of the trends observed as design changes were made, are drawn in Chapter 8 of this report.

4.3.1.1. Blast #1 : N-14 - Sep. 25, 1986

This was carried out using a single line of blastholes. Gravel stemming was employed between explosive decks. The blast caved the room over the area covered by the 6 northernmost blastholes. The first 4 blastholes in the firing sequence exhibited surface craters, and failed to collapse the room in the region adjacent to the panel entry.

Fig. 4-9 is a view over the caved portion of this room towards the surface heave at its southern end.

4.3.1.2. Blast #2 : N-9 - Sep. 26, 1986

This was a short two-row blast situated between a sinkhole and the north panel entry. A deep sinkhole was created at its southern end, where the room intersected the panel entry. This is illustrated in the foreground of Fig. 4-10. Most of the blast heaved up, however, and there exists doubt as to whether it caved over the entire depth of overburden.

4.3.1.3. Blast #3 : N-11 - Oct. 7, 1986

This was the first large-scale blast taken using two rows of holes centered on a room. It was successful over the

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FIG. 4-9: POST-BLAST PROFILE OF BLAST #1(N-14)
LOOKING SOUTH

FIG. 4-10: POST BLAST PROFILE OF BLAST #2(N-9)
LOOKING NORTH

entire length blasted. There was not sufficient time on the blast day to load all of the holes at the northern end. Some material heaved up against the pre-existing sinkhole at the southern end of the blast.

4.3.1.4. Blast #4 : N-1,N-2 - Oct. 9, 1986

This blast was taken on two short adjacent lengths of room, and included the connecting panel entry. This was the deepest average overburden cover encountered during the test program, and 8-inch diameter blastholes were employed. Two "closure" holes were drilled and blasted to 60 feet depth at each end of the entry.

The closure holes were not successful, and sinkholes of 20 and 15 feet were created at the western and eastern ends, respectively, of the panel entry. The blast exhibited considerable surface heave over its entire length; in places this heave was as much as 5-6 feet in height. Part of this blast is illustrated in Fig. 4-11. Fig. 4-12 shows an upwards buckling of the rock strata at the western end of this blast, evidence that there was "bridging" of the upper explosive decks.

4.3.1.5. Blast #5 : N-13 - Oct. 13, 1986

This was the second attempt to collapse a long room using a double row of blastholes. It was very successful, over a distance of approximately 120 feet. The ground surface dropped between 6 and 8 feet over much of the blast. A deep hole appeared over the southern end, at the intersection with the panel entry. This subsequently filled with sloughing material, to a depth of about 16 feet. This blast is illustrated in Fig. 4-13.

4.3.1.6. Blast #6 : N-8(S) - Oct. 14, 1986

This short blast, using the same drilling pattern as Blast #5, was located over a portion of underground opening between a partly-caved panel entry and a sinkhole. Three holes, spaced at 12-foot intervals, were also blasted along the adjacent panel entry. Surface heave of between 1 and 3 feet in height occurred over most of the blast. This can be seen in Fig. 4-14; the feature in the foreground is the pre-existing sinkhole.

4.3.1.7. Blast #7 : N-7 - Oct. 16, 1986

This blast was in a short length of partially caved room between two sinkholes. It resulted in surface heave of about 2-3 feet.

FIGURE 4-11: POST-BLAST PROFILE OF BLAST #4(N-1, N-2) LOOKING NORTH

FIGURE 4-12: POST-BLAST PROFILE OF BLAST #4 (N-1,N-2) LOOKING EAST, SHOWING PROBABLE "BRIDGING" OF UPPER PART OF BLAST

FIG. 4-13: POST-BLAST PROFILE OF BLAST #5 (N-13), LOOKING SOUTH

FIG. 4-14: POST-BLAST PROFILE OF BLAST #6 (N-8(S)), LOOKING SOUTH

In addition, 10 short holes were blasted in an attempt to partly fill adjacent sinkholes. The holes were located about 4 feet back from the sinkhole rim. Four blastholes were loaded with a single 5 foot cylindrical charge column and succeeded in breaking material into the sinkholes to the north of the blast. The sides of these sinkholes were approximately vertical prior to blasting.

Six holes were used to blast the sinkhole to the south of room N-7. These were loaded with a four foot column of ANFO, separated by 2 feet of stemming from a second three foot deck of the same explosive. Though some minor surface slumping occurred, these holes were unsuccessful in breaking material into the sinkhole, whose sides sloped at about 45-50 degrees.

4.3.1.8. Blast #8 : N-12 - Oct. 17, 1986

This was a double-row blast bounded at the north by a pre-existing sinkhole. For the first time the hole separation was increased from 13 to 15 feet on rows which remained 8 feet apart. Three holes were blasted at 12 foot spacing in the panel entry at its southern limit. The room blast itself resulted in the formation of a depression with a V-shaped cross section, and also broke some material laterally into the sinkhole.

There was surface heave, however, over the panel entry; this is shown in Fig. 4-15. The surface heave is in the foreground, followed by a region in which material was disturbed but remained at essentially the pre-blast ground level. The remainder of the blast caved with a very even profile, with straight sides reflecting the positions of the pillar edges.

4.3.1.9. Blast #9 : NC-7 - Oct. 20, 1986

This blast, the first attempted which was exclusively in 12-foot wide access development, produced very variable results. A very large opening was created at the position of the intersection of cross-cut with panel entry -it is likely that appreciable pillar failure had occurred here prior to the blast, creating a very large void.

At the south end of the cross-cut, and at western and eastern ends of the panel entry, there was appreciable surface heave. A single line of six-inch holes spaced 12 feet apart was used. Two closure holes of 25 feet depth were drilled at the eastern end of the panel entry and loaded with two explosive decks.

FIG. 4-15: POST-BLAST PROFILE OF BLAST #8 (N-12) LOOKING NORTH

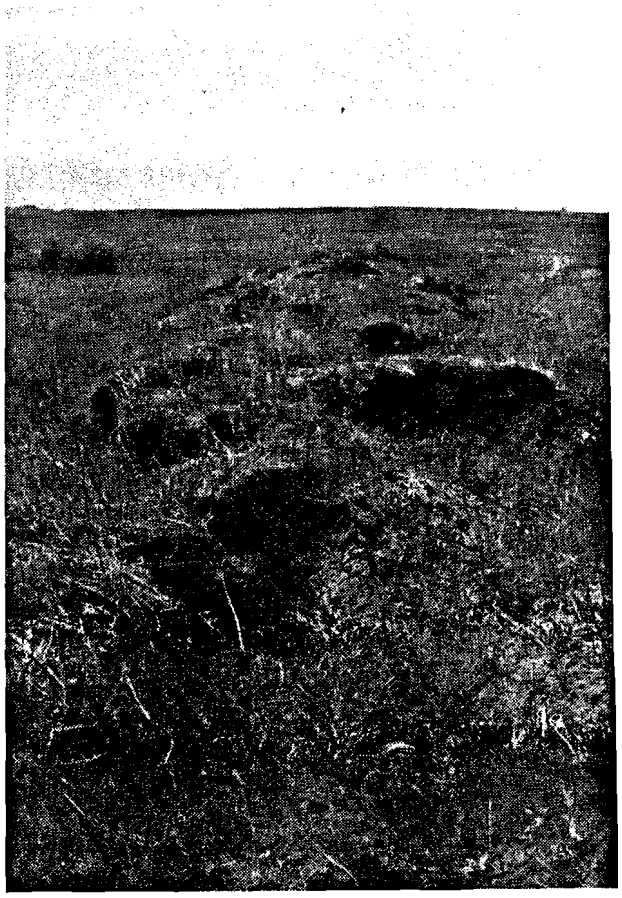

FIG. 4-16: POST-BLAST PROFILE OF BLAST #10 (S-1(N)), LOOKING SOUTH

4.3.1.10. Blast #10 : S-1(S) - Oct. 21, 1986

This blast took place at the western end of the south mining area, in some of the deepest cover encountered during the test blast program. The holes were very wet, with water and mud below 45 feet depth, corresponding to a clay seam immediately above the coal. Hole loading was therefore a very difficult process, as all of the slurry explosive had to be pushed down into the lower deck using loading poles.

Five explosive decks were used in some deeper holes. Closure holes 30 feet deep were drilled and blasted at each end of the southern panel entry accessing room S-1. These were loaded with two four-foot long explosive decks, with slurry employed in the lower of these.

The blast heaved up along almost the entire length, with the exception of the southern edge, where material was presumably able to move into the unblasted void to the south. The width of surface heaving approximated that of the room, and varied in height from 3 to 5 feet. This is illustrated in Fig. 4-16.

4.3.1.11. Blast #11 : S-12 - Oct. 24, 1986

This blast employed 15-foot spacing of 6-inch holes on rows 8 feet apart in fairly low overburden cover (38-42 feet). Holes were charged with 3 or 4 decks, the latter being the case when holes were deeper than 40 feet. The blast included 3 holes spaced 12 feet apart in the panel entry, plus 4 closure holes drilled to a depth of 25 feet, which were loaded with 2 decks.

This was the first major blast shot using 2 lines of holes in which ANFO was the only explosive used. It was very successful, with the entire room caving plus part of the adjacent panel entry to the west of the room.

The resulting sinkhole was 10-15 feet deep at the southern end, and sloped up to the original surface elevation at the intersection of room and panel entry. There was 3-4 feet of surface heave above the eastern limit of the blasted entry. In places the sinkhole created was up to 30 feet wide. The blast is illustrated in Fig. 4-17.

4.3.1.12. Blast #12 : N-8(N) - Oct. 28, 1986

This, the second blast taken on room N-8, was taken between a sinkhole and the northern limit of mining. It was shot using 15-foot hole spacing in two rows 8 feet apart, using only ANFO. It proved to be the most successful blast to date in deeper cover, which averaged 55-58 feet.

FIG. 4-17: POST-BLAST PROFILE OF BLAST #11 (S-12), LOOKING NORTH

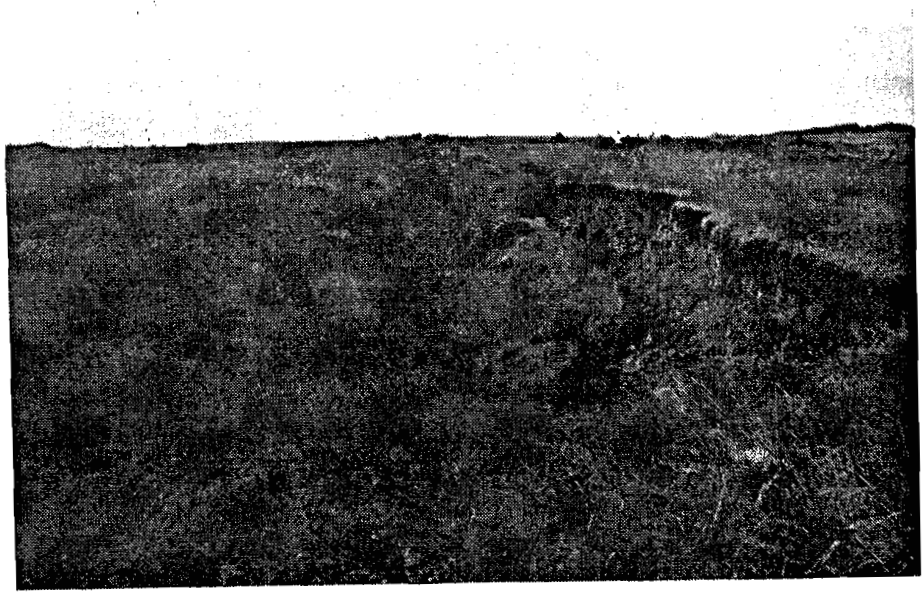

FIG. 4-18: POST-BLAST PROFILE OF BLAST #12 (N-8(N)), LOOKING SOUTH

There were 3-4 feet of surface heave at the southern end, with a general slope in the post-blast profile down to about 10 feet depth at the northern end of the room. The blast is illustrated in Fig. 4-18. There was considerable back-break into the eastern pillar, and the sinkhole was 28 feet across at its widest.

4.3.1.13. Blast #13 : N-10 - Oct. 30, 1986

This blast was very similar to that on room N-12, but in slightly deeper cover, around 54-56 feet. It caved over most of the room, with some surface swell experienced at the position of the panel entry. It was shot using ANFO explosive only.

4.3.1.14. Blast #14 : S-11(N) - Oct. 31, 1986

Room S-11 was drilled and blasted in two stages. The northern part included a small portion of panel entry, while the room itself was in overburden cover of about 43 feet. For the first time 50 milliseconds of delay period were allowed between explosive decks, which was twice the time that had previously been the case. The spacing on the holes in the panel entry was pulled in from 12 to 9 feet.

The blast caused caving over the entire length, with significant back-break, indicating that this room may have been somewhat wider than the planned 22 feet. The entry was also caved, but in the process a small but deep hole opened up at the junction of the entry with cross-cut SC-7, which undercut the cross-cut.

4.3.1.15. Blast #15 : S-1(S) - Nov. 3, 1986

This blast was the southern continuation of Blast # 10, employing two rows of 6 inch holes 8 feet apart, with 13 feet spacing between holes. The holes were wet, with mud from a wet clay seam filling most of them from 40 feet down. Slurry explosive was therefore used in the lower decks.

Considerable difficulty was experienced in hole loading due to the mud, which probably prevented the holes from being loaded to the correct density of explosive. In some cases extra boosters were employed in the lower two decks to ensure that all of the explosive was initiated.

The blast resulted in surface heave, about 3-4 feet in height, for almost all of its length; subsidence occurred at the southern end in the form of a 4 foot deep hole.

4.3.1.16. Blast #16 : S-11(S) - Nov. 5, 1986

This blast was taken on the southern end of blast # 14, in cover of around 41 feet in depth. It created a subsidence feature along almost all of its length. The original ground elevation was maintained at its northern end, adjacent to the previous blast on this room. This can be observed in Fig. 4-19, which is looking north towards the boundary between the two blasts taken on this room.

4.3.1.17. Blast #17 : S-10,SC-7 - Nov. 6, 1986

The blast on S-10 caved this room between a sinkhole and its southern limit of mining. In addition a blast was taken on SC-7, where an undercut sinkhole had developed as a result of blasts in the south panel entry adjacent to rooms S-12 and S-11. Six inch diameter holes were drilled in a single line at 9 foot spacings along this cross-cut between an original sinkhole and the one created by blasts.

In drilling these holes it was determined that the cross-cut was well on the way to caving naturally, with less than 20 feet of overburden remaining in places. As such, it was a potentially dangerous situation which required remedy. Two blastholes were drilled in the pillars to the north and south of SC-7 at its western end. It was determined that to position the drill rig over the undercut part of this development would be an unsafe practice.

The two blasts were connected using primaline, and the cross-cut was successfully blasted, thereby eliminating the hazard. The post-blast profile above the cross-cut is illustrated in Fig. 4-20.

4.3.1.18. Blast #18 : N-6 - Nov. 17, 1986

For this blast, experimentation with design was carried out to determine whether a single line of 8 inch holes, spaced 15 feet apart along the middle of the room, would cave a room in an area of fairly deep cover (55-60 feet).

It was determined from drilling and measurement that there was little or no void space remaining in the room below the coal, presumably due to influx of material from elsewhere in liquid form which had subsequently drained. It was also found that the room was narrower than average for the test site.

The surface elevation after the blast was basically unchanged, though the material was disturbed. Evidence of the recent extreme weather conditions was already visible from this blast, in the form of 18-24 inches depth of frozen ground below surface, which formed large slabs (see Fig. 4-21).

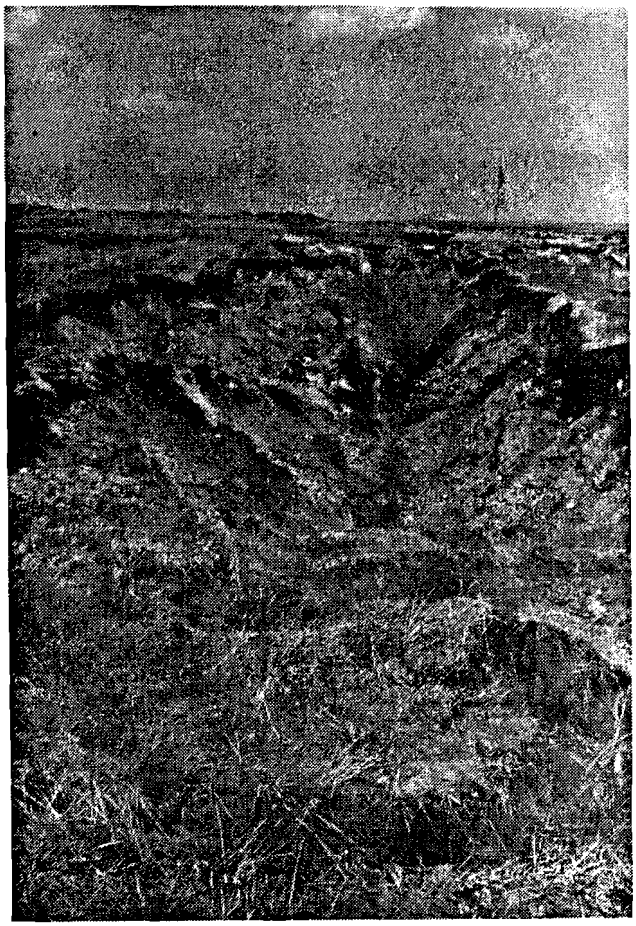

FIG. 4-19: POST-BLAST PROFILE OF BLAST #16 (S-11(S)), LOOKING NORTH

FIG. 4-20: POST-BLAST PROFILE OF BLAST #17A (SC-7), LOOKING NORTH

4.3.1.19. Blast #19 : S-9(N) - Nov. 18, 1986

Room S-9 was divided into two blasts to observe the effect of varying the blast pattern from a double to a single row situation in overburden conditions typical for the test site. Blast #19 was taken on the northern part of the room. Slurry explosive was used in the bottom two decks of each blasthole. The blast caved the room over its entire length, indicating the effectiveness of this explosive type even under the very cold temperatures experienced on this day.

Overburden depths of around 46 feet were encountered, and subsidence in the order of 15 feet was experienced. The depression created was between 25 and 35 feet wide, and is illustrated in Fig. 4-22. A deep hole, of around 20 feet, was initially created at the northern end, where lateral movement of material into the unblasted south panel entry was possible. This subsequently filled up by about 5 feet due to sloughing of material from the sides.

4.3.1.20. Blast #20 : N-4 - Nov. 19, 1986

During blasting operations in October a new sinkhole opened up on the northern end of room N-4. An exploration borehole in this area had indicated that the room had caved above the coal seam prior to the onset of blasting operations. It is possible that vibrations from nearby blasts accelerated the natural process of caving.

Ten holes of 20 feet depth were drilled around the sinkhole, which had near vertical sides. These holes were spaced 10 feet apart, and set back around 5 feet from the sinkhole, the sides of which were nearly vertical.

The blast succeeded in filling the sinkhole to within about three feet of the original surface elevation (see Fig. 4-23).

4.3.1.21. Blast #21 : S-9(S) - Nov. 20, 1986

The southern part of room S-9 was blasted using a single line of 6 inch holes spaced 12 feet apart. The same loading procedure was adopted as for blast #19, with slurry explosive used in the bottom two decks.

The purpose of this was for comparison of the effectiveness of a single versus a double line of blastholes in similar overburden conditions. This blast caved this portion of the room, and resulted in subsidence of between 10 and 15 feet. The width of the sinkhole created was about 20 feet on average; as would be expected this was less than that obtained using two rows.

FIG. 4-21: POST-BLAST PROFILE OF BLAST #18 (N-6), LOOKING NORTH

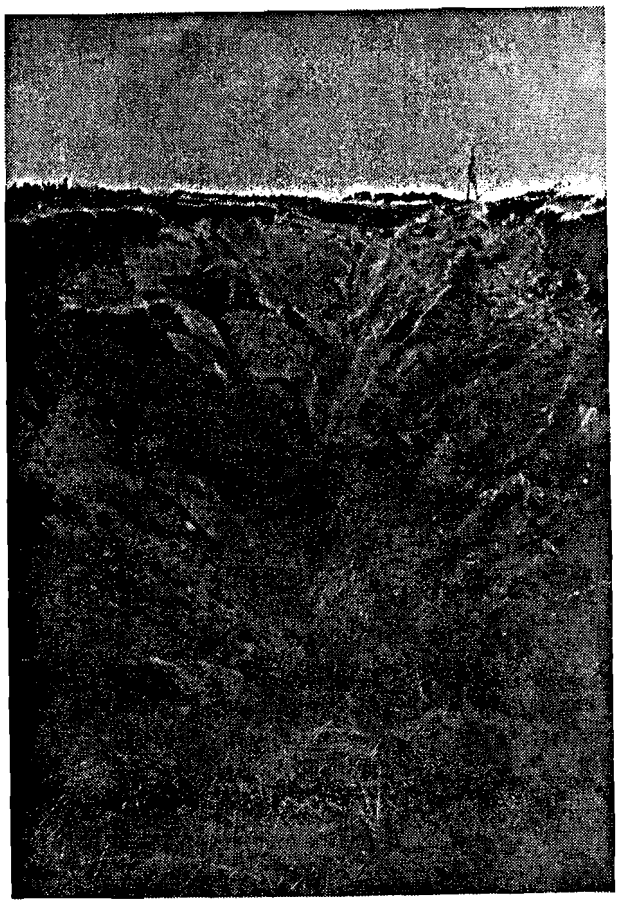

FIG. 4-22: POST-BLAST PROFILE OF BLAST #19 (S-9(N)), LOOKING NORTH

FIG. 4-23: POST-BLAST PROFILE OF BLAST #20 (SINKHOLE AT N-4), LOOKING NORTH, SHOWING SUCCESSFUL INFILLING OF AN INDIVIDUAL SINKHOLE

Figs. 4-24 and 4-25 show "before" and "after" views, respectively, of this blast, taken from the exact same position, looking towards the north. It can be seen that the ground elevation remained unchanged at the boundary between the two blasts.

4.3.2. Post-blast Exploration Drilling

A total of 613 feet of exploration drilling, from 9 holes, was carried out on November 4th. This was to determine, for five blasts where surface heave had resulted, whether the blast had successfully caved the openings.

The locations of these holes are shown on Fig. 4-26 for the site area. It was difficult to position the drill rig near enough to the blasted area to provide a safe site from which to intersect the blasted opening. In the case of 3 holes it was not possible to get near enough, and pillar was intersected. Where it was possible, variable results were obtained. Drilling results are summarized in Table 4-7.

In two cases (#2 and #4) it appears that the bottom two decks were successful in caving the opening, but that the upper two decks resulted in a "bridging" of material. One hole indicated that there was fill to within 3 feet of the roof. Elsewhere, it appears that the opening was filled up to the unbroken roof coal.

Comment on these results is made in Chapter 8 of this report.

TABLE 4-7 SUMMARY OF RESULTS FROM POST-BLAST EXPLORATION DRILLING

#	Room	TOC	BOC	Top of Void	Bottom of void	Hole Depth	Comments
1	N-8(N)	56	58	58.3	58.3	58	Void apparently filled
2	NC-7	-	-	22.0	35.0	50	Loose fill to 50 ft
3	N-1	62	79	-	-	79	Hole in pillar
4	N-1	-	-	38.0	46.0	60	Loose fill to 60 ft
5	N-1	60	78	-	-	78	Hole in pillar
6	N-1	60	65	65.0	68.0	75	Void nearly full
7	S-1(N)	60	78	-	-	78	Hole in pillar
8	S-1(N)	57	58	58.0	58.0	65	Void filled
9	S-1(S)	53	56	56.0	56.0	57	Void filled

FIG. 4-24: PRE-BLAST VIEW OF BLAST #21 (S-9(S)), LOOKING NORTH, PRIOR TO LOADING

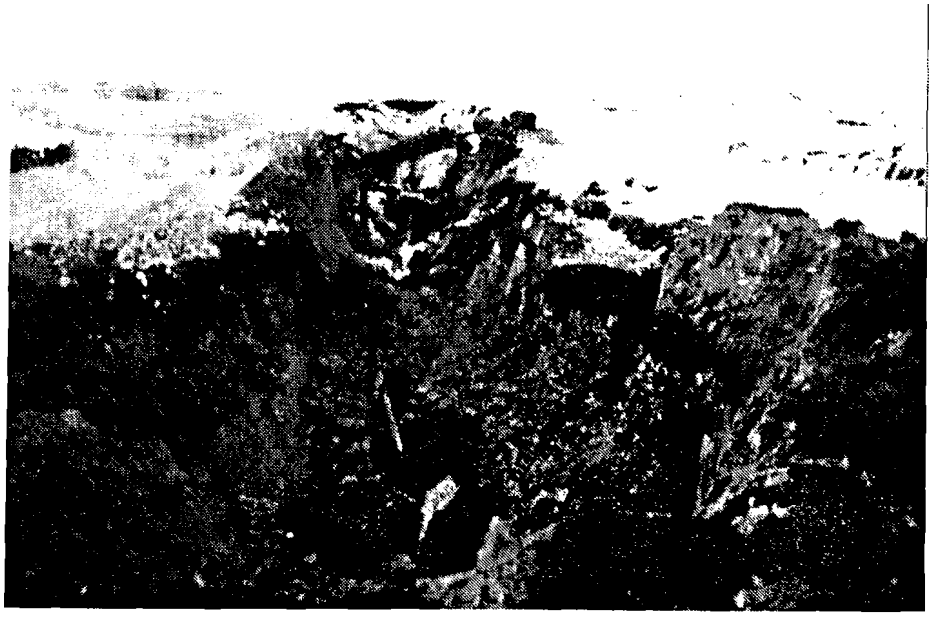

FIG. 4-25: POST-BLAST PROFILE OF BLAST #21 (S-9(S)), TAKEN FROM THE SAME POSITION AS FIG. 4-24

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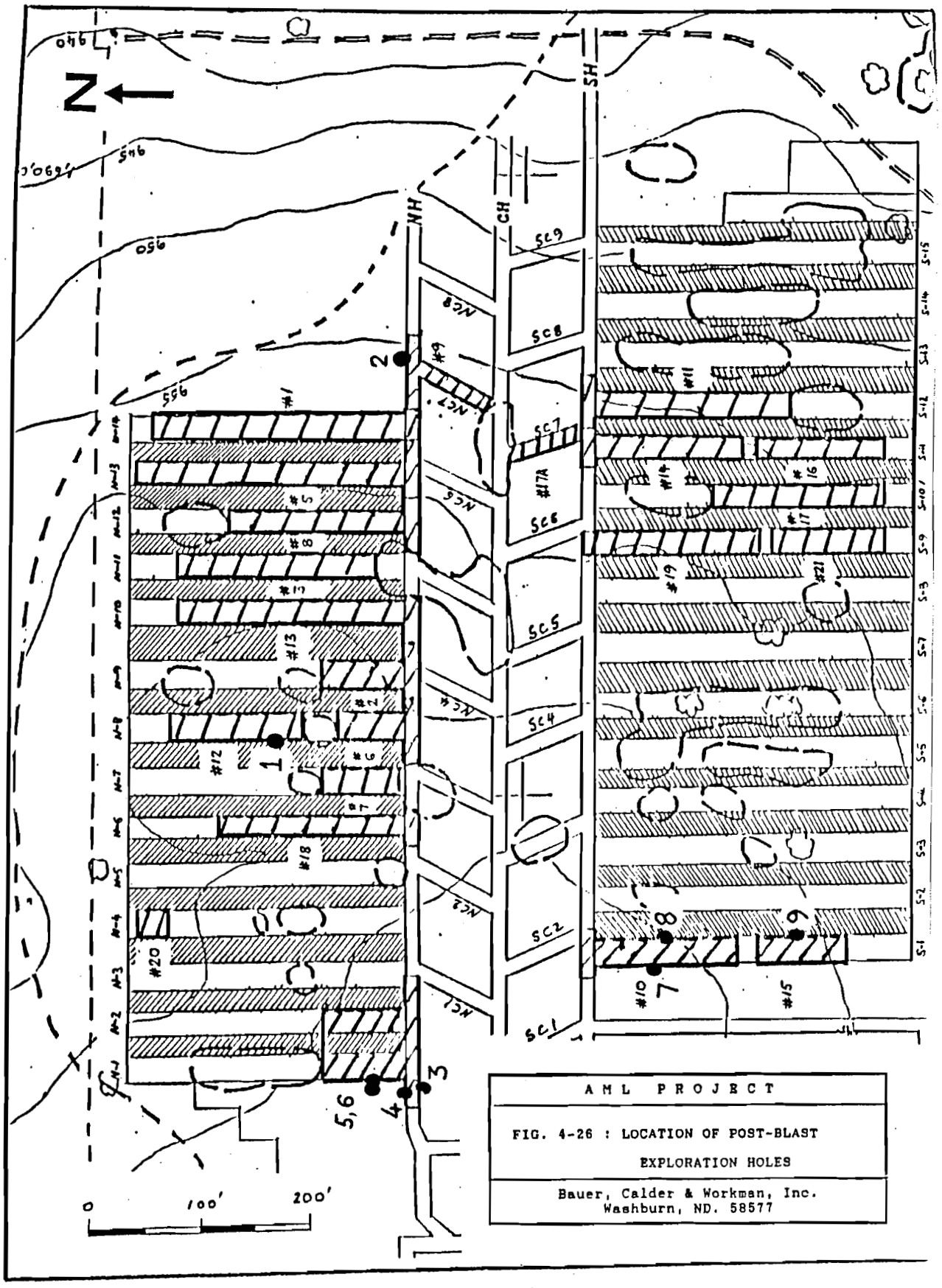

5. FIELD METHODOLOGY AND RECORD KEEPING

5.1. INTRODUCTION

The purpose of this chapter is to describe the basic practices developed during the fieldwork for the AML blasting project. It is intended that, in addition to reporting the work actually carried out, this should provide a useful guide to future operations of this type. Use is therefore made of photographs taken during the 1986 testwork. Some of these may contain indications of the targeting of Nonel trunkline for high-speed camera work. These targets should be ignored if this section is being used as a guideline for future AML blasting operations of a non-research nature.

Field methodology for site evaluation, including initial exploration drilling, was described in Chapter 3 of this report. In this chapter it is assumed that the selected site has been explored with respect to the general layout and extent of the underground openings underlying it.

A summary of typical "unit operations" for the work is contained in Table 5-1. This will in general act as a skeleton for the description given in this chapter. It should be appreciated, due to the novel nature of the work, that the field practices were being constantly developed and improved upon during the course of the testwork. This is especially true with regard to the record-keeping process, which is also described in this chapter.

An organized and systematic form of record keeping following the drilling of the blasthole pattern and prior to actual field loading is essential if holes are to be correctly loaded and blasting is to be successful. The type of information recorded and presented in this section is therefore of particular relevance to future AML reclamation work using blasting.

Although the drilling and blasting summary statistics presented here are essentially for the record, it is felt that they also would provide a guideline for future work. For this reason, the spreadsheets developed for presentation of the drilling and blasting statistics are described.

Table 5-2 contains a list of the field equipment used during the 1986 testwork. It is included because it may provide a useful "check-list" for future work of this type.

TABLE 5-1 : BLASTING FOR AML RECLAMATION - "UNIT OPERATIONS"

Room lineup:

- detailed exploration drilling to determine width and lateral extent of underground openings

Per blast:

- Pre-blast work:

  - design blast layout
  - layout blast pattern
  - drill blastholes
  - measure depths of voids and rooms
  - record depth of hole and of plug on stakes
  - plug holes
  - calculate actual loading data
  - estimate requirements for explosives and blasting accessories

- On blast date:

  - load powder and accessories from magazines and transport to site
  - unload powder alongside blastholes
  - make up boosters with appropriate DTH delays and place next to blastholes
  - place appropriate loading boards next to blastholes
  - back-fill blastholes to position of bottom deck
  - dewater if holes not drilled through to void and ANFO is to be used
  - re-plug any holes where plugs were lost
  - load decks in blastholes and stem collars
  - clean-up garbage and field equipment, remove vehicles and personnel
  - tie-in surface delays and bury primacord ends
  - set-up blasting seismograph
  - run out blasting cable
  - test cap and blasting cable using blasting galvanometer
  - install warning signs and road-blocks
  - connect cap and retreat to blasting position
  - sound siren, then blast
  - check misfires, sound siren again
  - reel in blasting cable; recover seismograph
  - return unused powder and accessories to magazines; make inventory

TABLE 5-2 : BLASTING FOR AML RECLAMATION - ESSENTIAL FIELD EQUIPMENT.

Major items - Rent or own:

- 2 x Pickups (Preferably 4-wheel drive) (should have snow-tires or chains for winter work)
- Tow-rope or chain
- Wooden protective pickup bed liners (required by regulations)
- Fire extinguishers (2 per pickup)
- 1 x Cap-box for pickup
- 1 x Dozer (depending on site conditions and accessibility)
- 1 x Bobcat (depending on availability of stemming materials)
- 1 x Powder magazine (semi-trailer of approved construction)
- 1 x Magazine for caps and fuses
- 1 x Magazine for Boosters, Primacord
- Padlocks for magazines (single key preferable)
- field office trailer (optional)

Other field equipment:

- Blasting machine (twist-type)
- Blasting galvanometer
- Blasting cable (500 feet)
- Siren
- Warning signs (number depending on access roads)
- 5lb hammer
- wooden stakes for blast layout and hole location
- 2 x shovels
- 2 x picks (if frozen stemming encountered)
- 2 x brass knives for cutting slurry explosive
- 2 x wire-stripper pliers
- Waterproof "magic" markers
- 2 x clipboards
- Electrical tape
- Spray paint
- 2 x 150 foot measuring tapes for layout of exploration and blast holes
- 2 x 100 foot blasting tapes
- 2 x 50 foot blasting tapes
- spare 100 and 50 foot blasting tapes
- lead weights for repair of blasting tapes
- Ten foot loading poles plus "stinger" point
- Baler twine (to retain plug at bottom of hole)
- Blasthole covers ("tin-hats")

5.2. ROOM LINEUP DRILLING

In order that a blast tie-in can be accurately located and centered above an underground opening, some form of line-up exploration drilling, additional to the initial site investigation drilling, will almost certainly be required. The amount of work involved will depend very much on the regularity of the distribution of rooms, pillars and entry-ways at the site. At the 1986 test site we encountered the fortunate combination of a rational mine plan, and an adherence to it. As such, therefore, the amount of lineup drilling required was relatively small.

There is no definite line that may be drawn between "exploration", "line-up" and actual blast pattern drilling for this type of work. For example, in the process of locating the existence of a room by exploration drilling, more than one hole may be drilled. If one hole is in the pillar, but a second attempt nearby encounters a void, then this information may be employed for room line-up. This was often the case during 1986 fieldwork, where the rooms were generally straight and of fairly consistent width.

If the pillar edge can be located with reasonable accuracy by exploration holes, then this knowledge could be used at the line-up stage to predict the position of the room center. One might also use a borehole TV camera for lining up the holes. However, this would only work if the TV image allowed an accurate determination of the distances involved. More research would be required before this would be practicable

In non-research applications, if the site is known to be fairly consistent with respect to underground layout, the possibility of drilling off the actual blasting pattern directly from the results of exploration drilling could be considered. For small discrepancies this may provide a cheaper alternative to the intermediate "line-up" drilling stage. However, very close control would be required for such a practice - the drilling of the blasting pattern would have to be stopped and modified immediately when there was indication of a blasthole being close to, or in, a pillar.

Acceptable practice under research or non-research conditions would be to use the actual blast pattern drilling to determine the longitudinal limit of a room which represented the limit of mining. It is far more economical to drill one blast pattern blasthole which does not encounter void than to drill a series of holes to pinpoint this.

During the 1986 testwork, despite the reasonable regularity and predictability of the location of underground openings, it was felt that line-up drilling was well justified. In part this decision was influenced by the essentially research nature of the project. It is obviously difficult, if not pointless, to draw meaningful conclusions with respect to the success of a

blast design if the blast was not placed at the optimum and planned location with respect to the underground opening.

As a result of this decision, however, it became apparent that even despite the regularity of the site, such a practice would have been justified irrespective of the research component of the project.

Almost all of the line-up drilling was carried out prior to the commencement of actual blasting operations. As such, some development was delineated which subsequently was not blasted. Conversely, one or two blasts were taken in areas which could have benefited from a little more line-up drilling. In a non-research oriented project it is possible that line-up drilling could be carried out as and when necessary. To a large extent this would depend on the commitment of the drill-rig to drilling off the required blasthole patterns, and the availability of a field engineer to supervise line-up drilling once blasting operations had commenced.

5.2.1. Line-up Drilling Practice

In many cases the location of the underground development had already been established during exploration drilling. The assumption was then made that the locating borehole was at the center of the opening. Since the typical width of the development type was known from maps, the practice was then to drill a first line-up hole to one side of the exploration drillhole at a distance of about one foot less than one-half of the development width.

If this hole also encountered a void, then a second hole was drilled the same distance away from the assumed room center on the opposite side. If this second hole encountered a void, then it could be deduced that the exploration hole was indeed central. If the second line-up hole encountered a pillar, then a third was drilled between it and the exploration hole. Once one lateral limit of the development has been determined in this manner, this is sufficient to locate a center line for the room, if its width is assumed to be consistent with that on the available mine plan.

In most cases a maximum of three line-up holes was sufficient to line-up each end of a room at the 1986 test site. Where the mine plan was closely adhered to, it was often not necessary to do this for every single room, but on alternate rooms.

It should, however, be well appreciated from the above that the time, effort and expense involved with line-up drilling is very dependent on the consistency of mining practice and the availability of reasonable mine plans which were actually followed.

5.2.2. Line-up Drilling Statistics

The results of line-up drilling carried out at the AML test site between September 24th and October 2 1986 are summarized in Table 5-3. A total of 82 line-up holes were drilled, for a total of 5061 feet. This represented about 45% of the total exploration and line-up drilling footage, and about 16% of the total drilling footage for the project.

With 21 blasts taken during the fieldwork period, this equates to about 4 line-up holes per blast. As mentioned in the previous section, rather more access development (main entries and connecting cross-cuts) was lined up than was subsequently blasted. An average of three line-up holes per blast would therefore be more typical.

The numbering principle for these holes is based on the same system used for exploration holes - for example, hole numbers N-14-3A, N-14-3B and N-14-3C relate to three line-up holes drilled on room N-14 at its southern end, near to exploration hole N-14-3.

5.3. PRE-BLAST FIELDWORK

General practice during the 1986 fieldwork was to lay out and drill off a blast pattern one or two days before a blast was to be taken. To a large extent this was controlled by the location of the blast with respect to other scheduled blasts. It is not wise to drill off a pattern close to a planned blast, thereby creating the risk that the vibration from the event could close off some holes, requiring re-drill. This was confirmed by one such instance during the fieldwork.

5.3.1. Blast Layout Design

During the 1986 testwork it was necessary to modify blast design philosophy as experience was gained. This is to be expected at any site, to successfully optimize blasting performance. Due to the experimental and research-based nature of this project, this was in fact not just necessary, but essential.

Thes blast design change may involve more than simply changing the number of decks employed as a function of depth (see Chapter 4). If one blast in a given area is less than successful, it may, for example, be necessary to decrease hole spacing, use a double rather than a single line of blastholes, move to a higher-density toe explosive, and so on.

Generally speaking the significance of this stage of AML blasting methodology will vary for different sites. In areas with fairly regular overburden conditions and a limited number

TABLE 5-3 : AML PROJECT - LINE-UP EXPLORATION DRILLING SUMMARY

NO	HOLE NO.	DATE	SURF.EL	TOC	BOC	BOS	VOID	CAVED	TOT.OB	TOT COAL	TOC ELEV	HOLE DEP.	DRIL. DEP
85	N-14-9A	24 SEP	957.0	41.5	60.0		60.0		41.5	18.5	915.5	60.0	60
86	N-14-9B	24 SEP	957.0	41.5	51.0		51.0		41.5	9.5	915.5	51.0	60
87	N-14-9C	24 SEP	957.0	42.0	49.0		49.0		42.0	7.0	915.0	49.0	50
100	N-9-9A	26 SEP	966.0	53.5	73.0	73.0			53.5	19.5	912.5	73.0	73
101	N-9-9B	26 SEP	966.0	54.5	59.0		59.0		54.5	4.5	911.5	59.0	60
102	N-9-9C	26 SEP	966.0	53.0	72.5	72.5			53.0	19.5	913.0	72.5	73
103	N-9-9D	26 SEP	965.0	52.5	57.5		57.5		52.5	5.0	912.5	57.5	60
104	N-9-9E	26 SEP	965.0	51.0	56.0		56.0		51.0	5.0	914.0	56.0	60
105	N-9-9F	26 SEP	966.0	52.5	72.0	72.0			52.5	19.5	913.5	72.0	73
106	N-1-9A	30 SEP	971.0	59.5	65.5		65.5		59.5	6.0	911.5	65.5	67
107	N-1-9B	30 SEP	971.0	59.0	66.0		66.0		59.0	7.0	912.0	66.0	70
108	N-1-9C	30 SEP	971.0	59.7	78.5	78.5			59.7	18.8	911.3	78.5	80
109	N-1-9D	30 SEP	971.0	60.0	67.5		67.5		60.0	7.5	911.0	67.5	70
110	NH-8C	30 SEP	971.0	59.7	65.7		65.7		59.7	6.0	911.3	65.7	66
111	NH-9A	30 SEP	972.0	60.0	68.5		68.5		60.0	8.5	912.0	68.5	75
112	NH-9B	30 SEP	972.0	60.0	68.0		68.0		60.0	8.0	912.0	68.0	80
113	N-9-9A	30 SEP	970.0	57.5	76.5	76.5			57.5	19.0	912.5	76.5	80
114	N-9-9B	30 SEP	970.0	57.0	74.0				57.0	17.0	913.0	75.0	75
115	N-9-9C	30 SEP	970.0					42.0				42.0	50
116	N-5-2A	30 SEP	968.0	55.0	55.2		55.2		55.0	0.2	913.0	55.2	60
117	N-11-2A	30 SEP	963.0	52.0	58.0		58.0		52.0	6.0	911.0	58.0	60
118	N-11-9A	30 SEP	963.0	49.5	69.0	69.0			49.5	19.5	913.5	69.0	70
119	N-11-9B	30 SEP	963.0	48.0	58.5		58.5		48.0	10.5	915.0	58.5	60
120	S-1-1A	30 SEP	970.0	56.0	63.8		63.8		56.0	7.8	914.0	63.8	65
121	S-1-1B	30 SEP	970.0	56.0	59.5		59.5		56.0	3.5	914.0	59.5	60
122	S-1-2A	30 SEP	968.0	53.5	58.0		58.0		53.5	4.5	914.5	58.0	60
123	S-1-2B	30 SEP	968.0	53.0	58.0		58.0		53.0	5.0	915.0	58.0	60
124	SH-8A	30 SEP	972.0	58.5	67.0		67.0		58.5	8.5	913.5	67.0	70
125	SH-8B	30 SEP	972.0	59.3	67.0		67.0		59.3	7.7	912.7	67.0	70
126	SH-7A	30 SEP	970.0	57.5	63.5		63.5		57.5	6.0	912.5	63.5	65
127	SH-7B	30 SEP	970.0	57.0	65.5		65.5		57.0	8.5	913.0	65.5	70
128	S-8-1A	30 SEP	962.0	45.0	47.0		47.0		45.0	2.0	917.0	47.0	50
129	S-8-1B	30 SEP	962.0	44.0					44.0		918.0	60.0	60
130	S-8-1C	30 SEP	962.0	44.0					44.0		918.0	60.0	60
131	S-8-2A	30 SEP	959.0	43.3	45.5		45.5		43.3	2.2	915.7	45.5	50
132	S-8-2B	30 SEP	959.0	43.8	63.8				43.8	20.0	915.2	60.0	60
133	S-8-2C	30 SEP	960.0	44.0	64.0				44.0	20.0	916.0	60.0	60
134	S-10-2A	1 OCT	956.0	41.5	59.7	59.7			41.5	18.2	914.5	59.7	60
135	S-10-2B	1 OCT	956.0	38.8	42.6		42.6		38.8	3.8	917.2	42.6	45
136	S-10-9A	1 OCT	955.0	40.5					40.5		914.5	60.0	60
137	S-10-9B	1 OCT	955.0	40.3	59.8				40.3	19.5	914.7	59.8	60
138	S-10-9C	1 OCT	955.0	40.0	59.5				40.0	19.5	915.0	59.5	60
139	SH-2A	1 OCT	951.0	36.0	41.0		41.0		36.0	5.0	915.0	41.0	45
140	SH-2B	1 OCT	951.0	36.0	43.8		43.8		36.0	7.8	915.0	43.8	50
141	CH-2A	1 OCT	952.0	37.0	44.5		44.5		37.0	7.5	915.0	44.5	50
142	CH-2B	1 OCT	952.0	38.0	43.6		43.6		38.0	5.6	914.0	43.6	45
143	SCC9-1	1 OCT	950.0					30.0				30.0	40
144	CH-6A	1 OCT	968.0	54.5	58.0		58.0		54.5	3.5	913.5	58.0	60
145	CH-6B	1 OCT	968.0	55.3	61.0		61.0		55.3	5.7	912.7	61.0	61
146	CH-8A	1 OCT	972.0	61.0	67.0		67.0		61.0	6.0	911.0	67.0	70
147	CH-8B	1 OCT	972.0	61.0	67.5		67.5		61.0	6.5	911.0	67.5	70
148	NCC1-1	1 OCT	972.0	60.7	80.0				60.7	19.3	911.3	80.0	80
149	NCC1-1A	1 OCT	972.0	60.0	79.5	79.5			60.0	19.5	912.0	79.5	80
150	NCC1-1B	1 OCT	972.0	59.3	78.6	78.6			59.3	19.3	912.7	78.6	80
151	NCC1-1C	1 OCT	972.0	59.6	67.5		67.5		59.6	7.9	912.4	67.5	70
152	NCC1-1D	1 OCT	972.0	59.0	67.0		67.0		59.0	8.0	913.0	67.0	70
153	NCC2-1A	1 OCT	969.0	57.0	64.0		64.0		57.0	7.0	912.0	64.0	65
154	NCC2-1B	1 OCT	969.0	56.0	64.0		64.0		56.0	8.0	913.0	64.0	65
155	NCC2-2A	1 OCT	970.0	57.3	76.5	76.5			57.3	19.2	912.7	76.5	80
156	NCC2-2B	1 OCT	970.0	57.8	67.0		67.0		57.8	9.2	912.2	67.0	70
157	NCC2-2C	1 OCT	970.0	58.0	65.0		65.0		58.0	7.0	912.0	65.0	65
158	NCC1-2A	1 OCT	972.0	60.5	67.0		67.0		60.5	6.5	911.5	67.0	70
159	NCC1-2B	1 OCT	972.0	60.3	68.0		68.0		60.3	7.7	911.7	68.0	70
160	N-12-2A	2 OCT	962.0	50.0	69.0	69.0			50.0	19.0	912.0	69.0	70
161	N-12-2B	2 OCT	962.0	48.5	57.0		57.0		48.5	8.5	913.5	57.0	60
162	N-13-2A	2 OCT	960.0	47.0					47.0		913.0	60.0	60
163	N-13-2B	2 OCT	959.0	44.5	50.5		50.5		44.5	6.0	914.5	50.5	51
164	N-12-9A	2 OCT	961.0	47.5					47.5		913.5	60.0	60
165	N-12-9B	2 OCT	961.0	46.0	57.5		57.5		46.0	11.5	915.0	57.5	60
166	N-13-9A	2 OCT	961.0	44.5					44.5		916.5	60.0	60
167	N-13-9B	2 OCT	959.0	44.5	51.5		51.5		44.5	7.0	914.5	51.5	55
168	N-13-9C	2 OCT	958.0	44.0					44.0		914.0	60.0	60
169	N-13-9D	2 OCT	958.0	43.5	55.0		55.0		43.5	11.5	914.5	55.0	55
170	NH-2A	2 OCT	953.0	42.0	45.0		45.0		42.0	3.0	911.0	45.0	45
171	NH-2B	2 OCT	953.0	42.0	48.0		48.0		42.0	6.0	911.0	48.0	50
172	NCC7-1A	2 OCT	956.0	40.0	45.0		45.0		40.0	5.0	916.0	45.0	45
173	NCC7-1B	2 OCT	956.0	40.0	46.0		46.0		40.0	6.0	916.0	46.0	46
174	NCC8-1A	2 OCT	953.0	38.5	48.0		48.0		38.5	9.5	914.5	48.0	50
175	NCC8-1B	2 OCT	953.0	38.0					38.0		915.0	60.0	60
176	NCC8-1C	2 OCT	953.0	38.0	48.0		48.0		38.0	10.0	915.0	48.0	50
177	NCC8-2A	2 OCT	953.0	39.8	45.0		45.0		39.8	5.2	913.2	45.0	45
178	NCC8-2B	2 OCT	953.0	38.0	46.0		46.0		38.0	8.0	915.0	46.0	46
82	TOC = Top of Coal			BOC = Bottom of coal		BOS = Bottom of seam					TOTAL:	4892.3	5061

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of different development types, the establishment of a "standardized" blast design for different hole depths should come fairly early on in the project. At other sites blast design may have to be an ongoing process, related to the logging of different geological conditions as encountered when drilling off the blast patterns.

Minor changes in design, related to the ends of a blast, or areas where different underground development types intersect, can generally be carried out in the field at the layout stage, without a separate major design effort.

5.3.2. Layout of Blast Pattern

Once a line indicating the center of the development to be blasted has been established, the field layout of blastholes is relatively simple. For single row blasting, two stakes are driven along the center line, a tape is stretched between them, and small wooden stakes or pegs are driven in at the appropriate positions. These pegs are numbered for identification purposes, and for correlation with the driller's logs.

The position of the first blasthole in a single row situation will be critical if the blast is up against one or more underground excavation limits. In the 1986 testwork common practice was to locate the first blasthole position in the panel entry at its center line, and subsequent blastholes according to the required hole spacing. In other cases the first hole would be located according to what was considered a safe and practical distance from an existing natural sinkhole.

In double row blasting the practice employed was to position two tall stakes on either side of an established center line location for a room. The distance between these stakes was set to the designed row spacing; in many cases, therefore, they were 4 feet on either side of the center line. This was repeated at the other end of the blast, and a tape was stretched between stakes along a line corresponding to the first row. Holes were positioned along this line, and then the process was repeated, with the appropriate offset if staggered rows were used, for the second line.

One of the line-up stakes for layout of a double-row blast can be observed in the foreground of Fig. 5-1.

5.3.3. Drilling of a Blast Pattern

The drilling of a blast pattern was, in the 1986 testwork, generally left to the driller, without engineering supervision. The driller was instructed to log major geological features, such as presence of rock layers, change from sand to clay, and of course the depths at which coal and void were encountered.

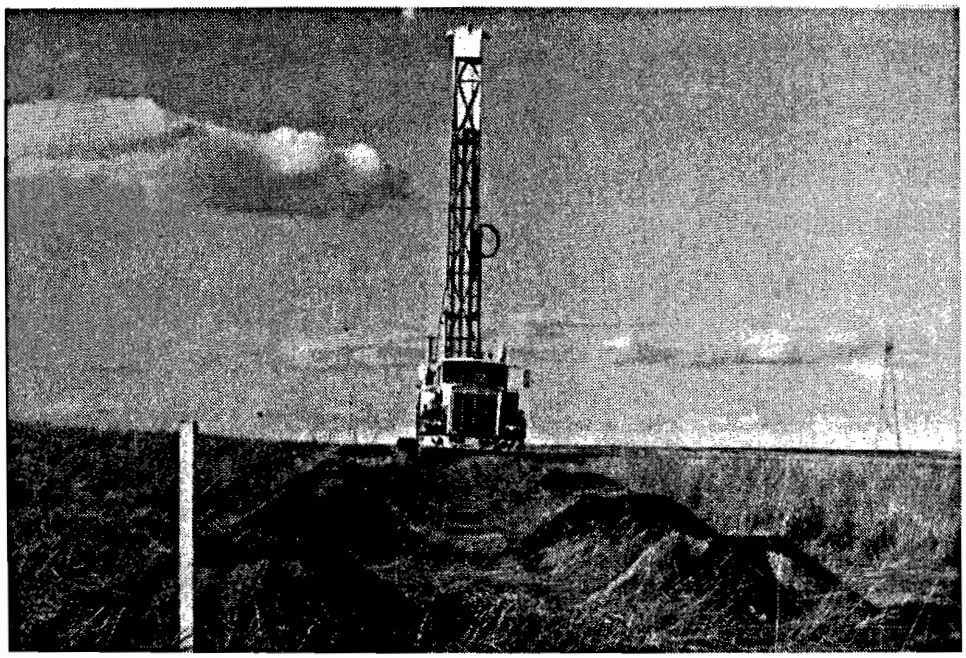

FIG. 5-1: DRILLING OF A TWO-ROW PRODUCTION BLAST HOLE PATTERN

In the event that a hole was found to fall in a pillar, the driller was to inform the Field Engineer before proceeding further. As mentioned earlier, blasthole drilling was generally used to determine the end of a development - the order in which holes was drilled was therefore controlled in such a way that this could be achieved.

Fig. 5-1 shows the drilling of a double-row blasthole pattern in the field. The rig has completed one row to its longitudinal limit, and is drilling the second row to complete the pattern.

5.3.4. Blasthole Measurement

On completion of a drilling pattern it is necessary to make certain measurements prior to the plugging and loading of the holes. This consisted of measuring:

- depth to bottom of room
- depth to void

This was achieved using a blasting tape - a 50 or 100 foot tape weighted at the zero end with lead weights. The first measurement is made by dropping one end of the tape down the hole until the tape goes slack. The tape is then pulled up from the hole until its weight is felt - this is the maximum depth to the bottom of the void. In many cases the rooms were found to contain water. In one area the room was filled with mud, and it was virtually impossible to determine the bottom of the holes since the tape became stuck and coated with mud.

The usefulness of such measurement is discussed in Chapter 8 of this report. For AML work of a non-research nature it would generally be possible to omit this. However, a knowledge of the void depth can help with the interpretation of results from the blasting program.

The most important measurement is obviously the depth to the void; this is recorded by the driller, but must be checked prior to plugging. It is determined by pulling the tape slowly upwards from the bottom of the hole until it can be felt to "snag" against the roof of the opening. This is not always easy - it may be necessary to swing the tape from side to side, and repeat the upwards movement several times until the hole bottom can be detected. It is a measurement technique that improves with practice! The measurement is less difficult when the driller records accurate hole depths, as was the case for the field test program.

The measurement of depth of void was, when different from that recorded by the driller, consistently less. This is probably due to some spalling of the roof coal (which generally formed the hole bottom) after the drill string was retracted. It is a very important point, and indicates that shortcuts should not be made with respect to this measurement, as will be explained in the next section when hole plugging is described.

These measurements were recorded on special field sheets designed for this purpose (described later in Section 5.5.1.). At the same time, it was found to be beneficial to record the depth to void on the stake which was previously employed to mark and identify the blasthole position.

5.3.5. Plugging of Blastholes

Since all blastholes should bottom out into underground workings, it is necessary to plug the holes at their bottom in order to contain the column of explosive and stemming. In the VCR mining method[6,7] this is achieved using two wooden wedge-shaped plugs, one of which is pushed onto the other.

For AML blasting work, however, it was found that plastic "seismic" type hole plugs (sometimes called "tiger paws" in the seismic drilling business) were adequate. One of these is illustrated in Fig. 5-2. It consists of a plastic cone which is somewhat less in diameter than the blasthole, to which are attached plastic fins. These cause the entire plug to be a couple of inches in diameter greater than the hole, thereby holding the plug at a desired position once pushed down the hole.

Doubt existed as to the capability of these plugs to hold up the vertical component of the weight of explosive and stemming in the hole, so they were secured using baler twine. Figure 5-2 illustrates that the twine was doubled, threaded through the plug and knotted several times so that it did not pull through the hole at the vertex of the plastic cone.

The cone was inserted in the blasthole using a wooden attachment which screws into the bottom of a loading pole. This is shown in Fig. 5-3. For another type of hole plug employed, a brass "stinger point" attachment was used to secure the plug to the bottom loading pole while it was being pushed to the required depth.

On average the hole plug was set about six inches above the measured void depth, to ensure that it did not fail to give a tight seal for backfill stemming.

Figs. 5-4 and 5-5 are included to illustrate the use of loading poles. Ten foot lengths of pole are attached by means of brass connectors. The hole plug is pushed to the required depth.

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FIG. 5-2: ILLUSTRATING USE OF BALER TWINE TO HOLD PLASTIC HOLE PLUG IN POSITION IN BLASTHOLE

FIG. 5-3: ILLUSTRATING USE OF "STINGER POINT" TO POSITION PLASTIC HOLE PLUG USING LOADING POLES

FIG. 5-4: ILLUSTRATING USE OF LOADING POLES TO PUSH PLASTIC HOLE-PLUG INTO POSITION

FIG. 5-5: ILLUSTRATING COUPLING OF LOADING POLES

It is a good idea to put graduated marks on one pole and use it as the last in the sequence, to simplify the accurate placement of the cone.

Once the plug is in position, the loading poles are retracted. The string attached to the plug is tied off at surface to the wooden identification stake.

5.3.6. Estimation of Explosives Requirements

A final task to be performed prior to the day of a blast is the estimation of the quantity of explosives that will be required. This estimate will be based on experience, using previous magazine inventory sheets, a knowledge of requirements from previous blasts of a similar size, and the specific blast design.

The quantity and number of each down-the-hole delay should be established prior to the blast. This will be determined in part by the blasthole depths, which affect the number of decks required. The actual choice of delay numbers will be that of the Blasting Engineer in charge. Once the number of holes and number of decks per hole is known, it is also possible to estimate the required number of boosters.

It is obviously preferable to overestimate rather than underestimate the requirements of explosives, boosters and delays, especially if the blasting site is an appreciable distance from magazine locations. It is a good idea to include an extra electric blasting cap, also. Unused products should be returned to the magazines as soon as possible after blasting to prevent loss, and for obvious safety reasons.

5.4. PROCEDURE ON BLAST DAY

The "unit operations" that should be performed on the actual day of a blast are described in the following sections. It is important to realize that once explosives have been loaded into a blasthole, it is necessary either to complete the blast that day, or to post a guard overnight.

Bearing this fact in mind, it is advisable to take stock of weather conditions and labor availability prior to opening the magazine. Once hole loading is underway, it is a good idea to start at one end of the blast and work toward the other, even if more than one loading crew is available. If this is done, it is then possible simply to shorten the blast if operational conditions arise which make it impossible to complete the planned job.

5.4.1. Transport of Explosives

During 1986 testwork explosives magazines were sited about half a mile away from the blast site. Bagged ANFO and slurry products were stored in an approved semi-trailer type magazine rented from a local supplier. This is illustrated in Fig. 5-6. In addition, there were two smaller aluminum magazines, one for storage of boosters and primacord, and the other for storing caps and delays. One of these is illustrated in Fig. 5-7; it can be seen that this is grounded to earth to eliminate any potential hazards from static electricity.

At the start of each blast day, the appropriate powder requirements were loaded from the semi-trailer magazine into a pickup. The pickup had a specially constructed plywood bed, so that at no point did explosives come into contact with the metal bed of the vehicle. Seams and nail positions in the wood were caulked.

Wooden stakes were fixed to the inside of the pickup bed at the front, on either side, for mounting fire extinguishers. The pickup used is illustrated in Fig. 5-8; the fire extinguishers are not mounted here, as it was being used to transport field supplies other than explosives at the time. When carrying explosives, these should not attain a level higher than the pickup box during loading or transportation.

A portable cap box, of wooden exterior and interior construction, containing layers of sheet rock and steel, was used in the pickup to transport delays and caps. Magnetic signs indicating the carriage of high explosives were attached to the sides, front and back of the pickup during explosives transport.

Explosives were unloaded alongside the blastholes in the field; Fig. 5-9 illustrates bags of ANFO and slurry adjacent to one blasthole, ready for loading.

5.4.2. Preparation of Boosters and Down-the-hole Delays

Booster and down-the-hole delay combinations can be made up in advance of actual blasthole loading. This should be carried out at the actual blasthole locations, and not prior to transportation from the magazines. Boosters used during the 1986 testwork had a cardboard tube taped onto the side in order that they could be slid down a single primaline.

The connection of the DTH delay to the booster for the bottom deck number consisted simply of running the delay down the center hole, looping the Nonel tube at the bottom, and inserting it vertically upward into the booster core. The slider tube was not required in this case. For the other boosters, the delay was threaded down through the cardboard tube, inserted vertically into the bottom of the cast booster and then doubled

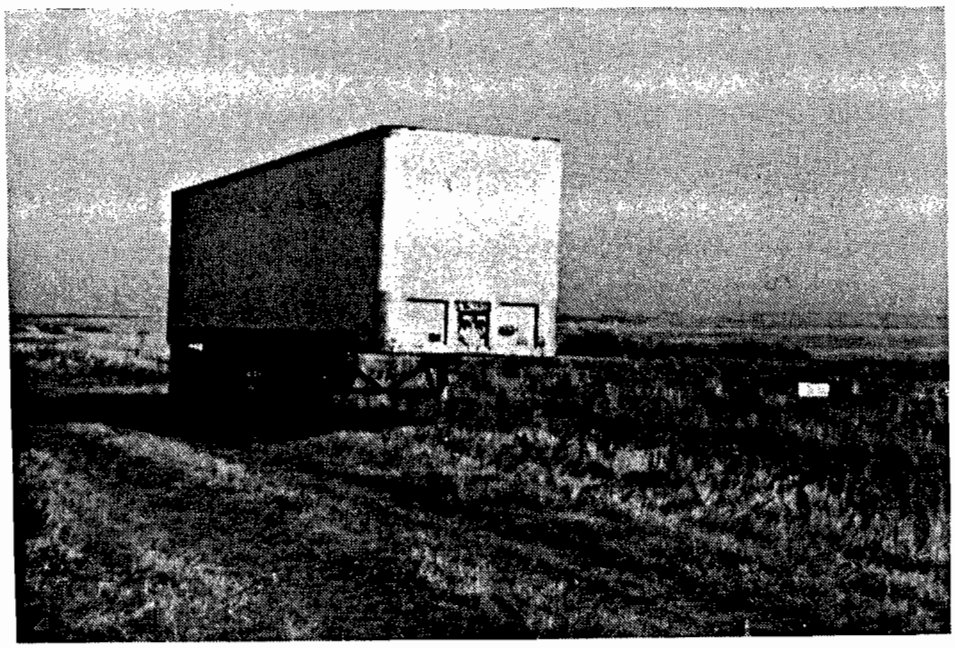

FIG. 5-6: TRAILER-TYPE EXPLOSIVES MAGAZINE USED DURING FIELD BLASTING TESTWORK

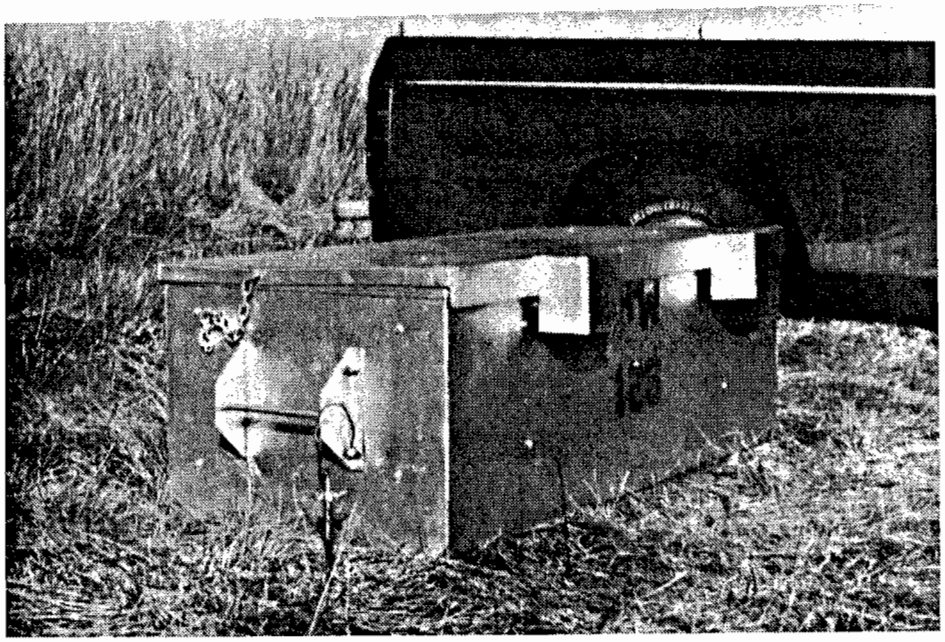

FIG. 5-7: ILLUSTRATING GROUNDED MAGAZINE USED FOR STORAGE OF DELAYS AND CAPS DURING FIELD BLASTING TESTWORK

FIG. 5-8: ILLUSTRATING PICKUP USED FOR TRANSPORT OF EXPLOSIVES AT TEST SITE

FIG. 5-9: ILLUSTRATING EXPLOSIVES AND PREPARED PRIMER AND D-T-H DELAY COMBINATIONS ADJACENT TO BLASTHOLE READY FOR LOADING

back into the cap well. Actual connection procedures will be very product-dependent. Once the booster/delay combinations were made up, they were placed adjacent to each blasthole ready for loading. This is illustrated in Fig. 5-10.

5.4.3. Deck Loading

The first stage of deck loading is a check, using the blasting tape, that the hole plug is still securely in place. The hole is then backfilled up to the position of the bottom explosive column. If there has been some tilting of the plastic hole plug, such that it does not form a tight seal at the bottom of the hole, then this will become apparent at the backfill stage. It may then be necessary to re-plug the hole before proceeding; it is for this reason that spare hole plugs should be included on the blast day itself.

Fig. 5-11 illustrates backfill of a blasthole using gravel stemmings prior to explosive loading. The user of the blasting tape must check constantly that the stemmings level does not come higher than the planned position of the lower deck.

It was found to be impractical to drop a 5 inch diameter slurry bag directly into a six inch hole without creation of time-consuming hang-ups in the blasthole, which had to be cleared using the blasting poles. Approximately one half of the explosive for the lower deck is poured into the hole; this is illustrated in Figs. 5-12 and 5-13 using the Iregel slurry.

This product was found to be very difficult to handle, with the best method being to slit the sack open as shown, and pour the slurry into the hole from the inner plastic bag. The second slurry type used, Energel, was much stiffer, and could be cut into chunks which were then manually dropped down the hole.

The bottom booster/delay combination was tied to the primaline and lowered down the hole. This is shown in Fig. 5-10; the cord already visible in the hole on this picture is the baler twine holding the hole plug in place.

The spool of primaline is illustrated in Fig. 5-14. this also shows that once the bottom booster is in place, the primaline can be cut off at surface, leaving about 2-3 feet for the subsequent blast tie-in. The other boosters, in the appropriate order with respect to their delay numbers, are threaded onto the primaline ready for dropping down the hole, and the line is tied off at surface to the wooden stake (see Fig. 5-14.)

Explosives and stemming are in turn filled to the appropriate levels in the hole. In each case the practice was to put about half the explosive column in, slide the booster/delay down the primaline into position, and to top the deck up to the

FIG. 5-10: ILLUSTRATING BOOSTER AND D-T-H DELAY FOR BOTTOM EXPLOSIVE DECK CONNECTED TO PRIMACORD DOWNLINE

FIG. 5-11: BACKFILLING A BLASTHOLE TO POSITION OF LOWERMOST EXPLOSIVE DECK USING CRUSHED ROCK STEMMING MATERIAL

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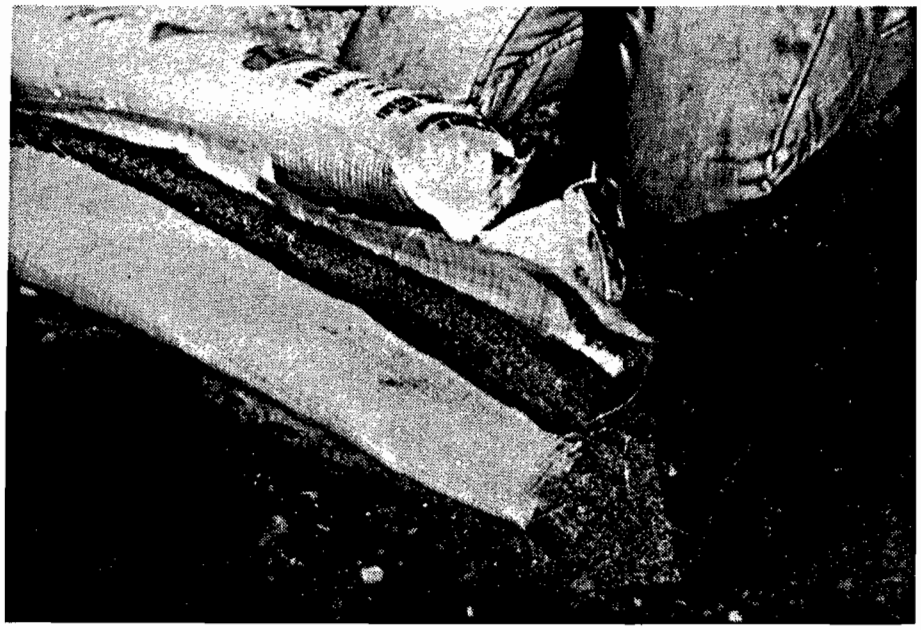

FIG. 5-12: ILLUSTRATING CUT BAG OF IREGEL SLURRY EXPLOSIVE PRODUCT READY FOR LOADING

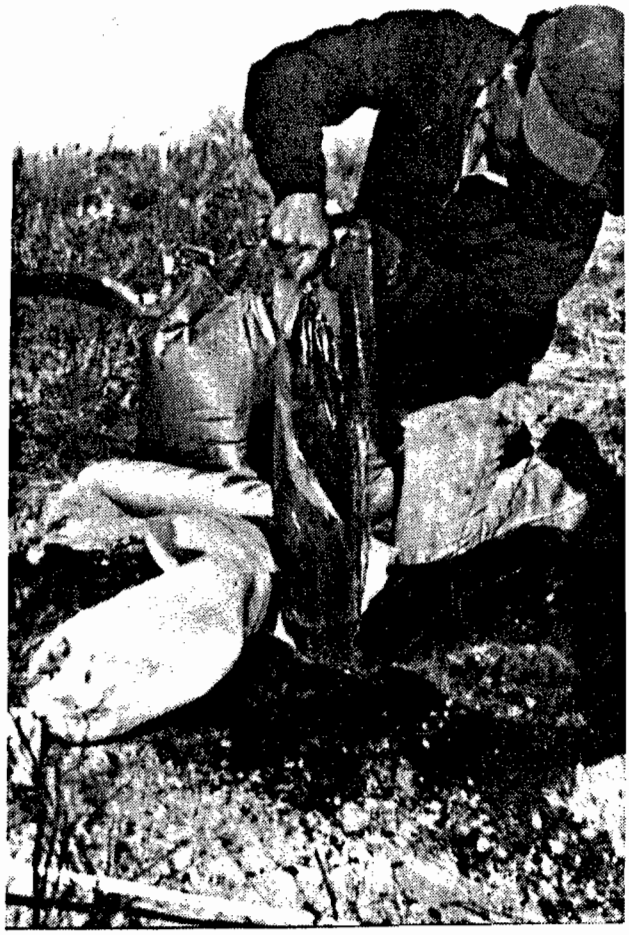

FIG. 5-13: LOADING OF IREGEL SLURRY EXPLOSIVE IN BLASTHOLE

FIG. 5-14: ILLUSTRATING REEL OF PRIMACORD DOWNLINE, AND PRIMER/DELAY COMBINATIONS STRUNG ON DOWNLINE READY FOR USE DURING HOLE LOADING

FIG. 5-15: ILLUSTRATING USE OF WEIGHTED BLASTING TAPE TO MEASURE HEIGHT OF EXPLOSIVE DECK DURING HOLE LOADING WITH BAGGED ANFO EXPLOSIVE

height indicated on the loading board. Fig. 5-15 shows ANFO being poured into a blasthole while the height of the total column is being monitored using the blasting tape.

When the top explosive deck has been filled, the hole collar is stemmed to surface, and the loading crew moves onto the next blasthole.

5.4.4. Blast Tie-in

The surface tie-in of the blast was carried out using 42 millisecond delay caps and Nonel trunkline. The Nonel/delay combination is a single unit. One end of the Nonel tube consisted of a plastic J-hook; the other end was connected to the delay. The delay end was contained in a plastic "bunch block" into which the primacord downline could be connected.

The connection procedure is illustrated in Fig. 5-16. The incoming Nonel tube is shown to the rear of the photograph. The white plastic bunch-block is located at the center of this photo. The primacord downline is looped inside the bunch block such that it surrounds the delay cap. The primacord was cut off about 6 inches from the top of the hole. The J-hook for the next Nonel tube was clipped to the primacord downline just above the hole stemming as shown in Fig. 5-16.

It can be appreciated that detonation of the incoming Nonel tube is carried out by the primacord from the previous blasthole. Forty-two milliseconds later the blasthole illustrated is initiated, including the outgoing Nonel tube via the J-hook. The next hole in the sequence is detonated 42 milliseconds later, as the delay is located at the other end of the outgoing Nonel tube.

For single row blasts the tie-in was very simple; the blast was sequenced from one end of the row to the other. In double row blasting, a "zig-zag" tie in was employed. This can be seen in numerous examples in the blast sketches given in Appendix A, and is illustrated in Fig. 5-17. Though loops, and even knots, in the Nonel tube are not generally regarded as a problem to their efficient function, care was taken to avoid this when finalizing the tie-in. It was also ensured that one Nonel tube did not touch another.

Once the surface tie-in was complete, and had been checked, the J-hook, bunch-block and exposed primacord were buried with stemmings to reduce noise during blasting.

5.4.5. Connection and Blasting

Since the delay and bunch block end of the surface delay was always situated at the initiation point of the blasthole, it

FIG. 5-16: ILLUSTRATING BUNCH BLOCK AT ONE END OF INCOMING NONEL NOISELESS SURFACE DELAY, CONNECTED TO END OF PRIMACORD DOWNLINE, AND J-HOLE CONNECTOR FOR OUTGOING SURFACE DELAY

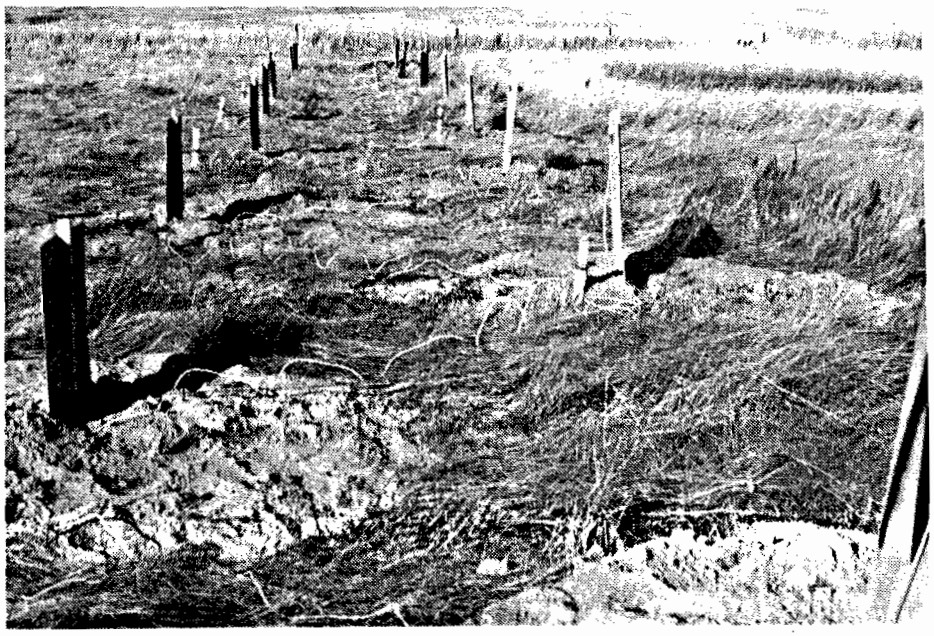

FIG. 5-17: ILLUSTRATING SURFACE TIE-IN OF A DOUBLE-ROW BLAST USING NONEL NOISELESS SURFACE DELAYS

follows that the initiation point of the blast itself was the J-hook end of the first delay.

Once the surface tie-in was completed, the blasting cable was run out from a selected safe position to the start of the blast. Also at this time it was generally the practice to set up the engineering seismograph, and the high-speed camera if employed.

The two wires at the end of the blasting cable furthest from the blast were twisted together. At the blast end the electrical continuity of the cable was tested using a blasting galvanometer.

The electric blasting cap was then laid on the ground some distance from the blast itself, pointing away from the operator. The wires running into the cap were connected to the blasting galvanometer to test the cap. The two wires on the cap were disconnected from the galvanometer, and the ends twisted together to short the circuit out in the cap. The cap was connected to the first Nonel by taping it with electrical tape. The cap should always point in the direction of the blast. This is illustrated in Fig. 5-18.

While this was being done the roadblocks were being set up, and the warning signs installed. A final check was made of the blast tie-in. The site was cleared of all non-essential personnel, who were stationed at strategic points around the blast site to observe if all was clear. These observers should be in sight of the Blasting Engineer, or if not, the operator of the blasting siren. Otherwise radio communication will be necessary.

The siren employed ran off a 12 volt vehicle battery. When the siren operator was satisfied from his observation and from that of other observers in his vision that all was clear, he flashed the vehicle lights. On the indication from the Blast Engineer, who was stationed at the position from which the blast was to be set off, the warning siren was sounded.

At this time the Field Engineer connected the blasting cap to the incoming blasting cable, which was still shorted out at the other end. The blasting cap was buried, and the Field Engineer joined the Blasting Engineer at the blasting point.

The blasting machine was connected to the cable, and after a final check that all was clear, the blast was set off. In many cases during the 1986 testwork, the Field Engineer operated, either directly or using a remote switch, the high-speed camera. The Blasting Engineer in these cases gave a countdown from five in order that the camera could be activated.

A typical view from the blasting point is illustrated in Fig. 5-19; the high-speed camera is visible in the middle

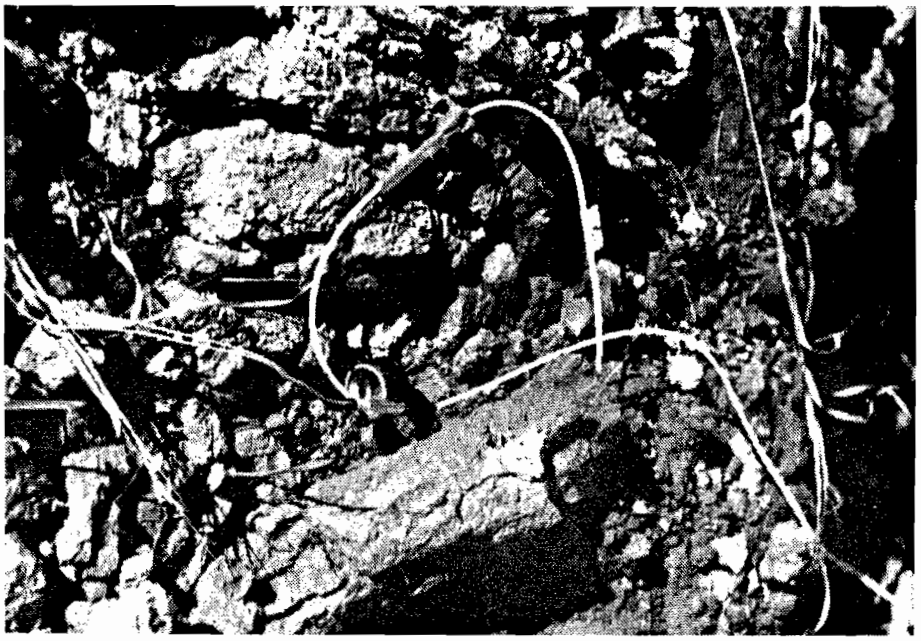

FIG. 5-18: ILLUSTRATING CONNECTION OF ELECTRIC BLASTING CAP TO NONEL SURFACE DELAY LEADING TO FIRST HOLE IN A BLAST

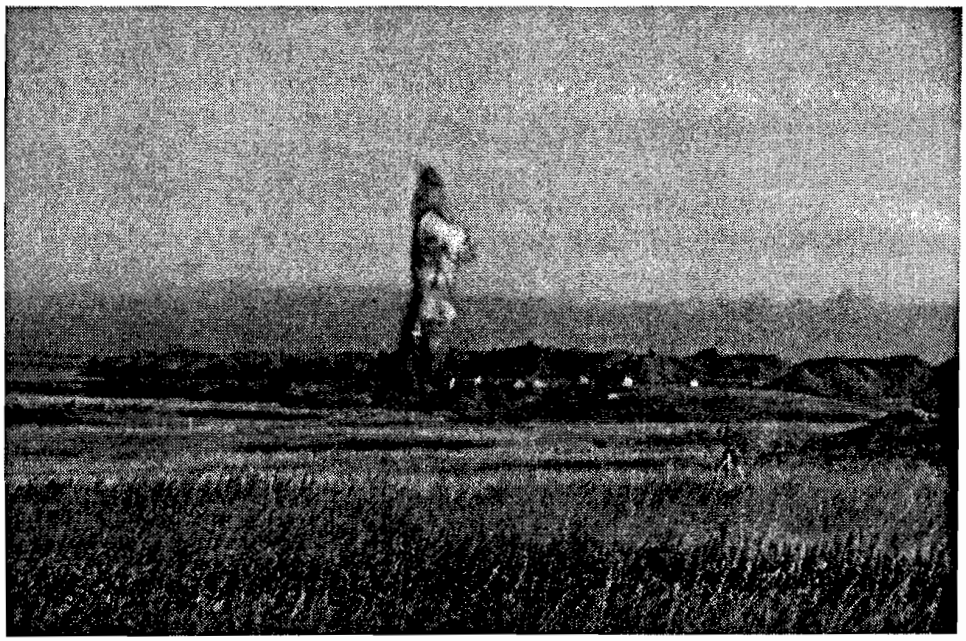

FIG. 5-19: AML BLAST VIEWED FROM VANTAGE POINT; HIGH-SPEED CAMERA VISIBLE IN FOREGROUND

distance, and was activated by using the remote cable from the same position that this photograph was taken.

5.4.6. Post-blasting Procedures

Following the blast a period of about 10-15 minutes was allowed to elapse for any blast fumes to dissipate, and then the blast was checked for misfires. Once it was established that all blastholes had fired, the signal was given to the siren operator, who sounded the all-clear. Care must be taken during the post-blast inspection because of potential ground instability in the vicinity of the blast.

Post-blast cleanup then commenced. The blasting cable was reeled in, the engineering seismograph and high-speed camera were recovered and packed away, and other miscellaneous field equipment stored in the trailer.

Unused explosives were returned to the magazine, and inventory sheets were filled out.

5.5. BLAST RECORD KEEPING

Blast record keeping is a vital part of any operation using blasting. It was particularly essential for this project, due to its research nature. It is felt that procedures developed during the testwork may be used as a guideline for record keeping in non-research applications of AML blasting also.

5.5.1. Field Blasthole Measurements

The types of field measurement carried out on blastholes prior to blasting was described in Section 5.3.4. of this report. It is necessary to have a consistent form of record keeping, both for use in the field and, if required, for later analysis. The record sheet developed during the testwork is presented as Table 5-4. Actual field records made using these sheets are included as Appendix B.

Each blast is identified by a number, location, and by the day on which it occurred. The hole diameter and the number of holes blasted are also important records. For each blast, holes are identified by their number (in the column marked #); these numbers should correspond to those on a blast sketch.

From the driller's logs, the depth to the top of coal seam (column "TOC") and to the void as drilled should be transferred. As noted earlier, it is important to measure this void depth again prior to hole plugging. The measured depth is inserted in the next column ("VOID MEAS") and a depth is then assigned for the hole plug (see column marked "PLUG" in Table 5-3).

AML PROJECT - BLAST SUMMARY

BLAST #: _____

LOCATION: _____

DATE _____

HOLE DIAM _____

NO. HOLES _____

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	ROOF COAL	BOTTOM	VOID DEPTH	ROCK LAYERS	COMMENTS

TABLE 5-4: EXAMPLE OF FIELD BLAST SUMMARY SHEET FOR AML BLASTING WORK

Sometimes the drilled depth to void differed by more than six inches from the measured void depth. Obviously the plug could not be placed any lower than the position of the void as indicated by the measuring tape, otherwise it would be hanging in the roof of the room and not be a plug at all. However, the depth at which the driller encountered the void is an important result, as it reflects the total depth, and, importantly, the thickness of roof coal in the area of the blasthole.

The column headed "DEPTH" was thus included in the field measurement sheet, which represented the Field Engineer's estimation of the true depth of overburden around that blasthole. This was the depth which was used when loading the decked explosive columns (that is, the depth of the appropriate loading board consulted for loading). In most cases it was the same as the depth of the void as reported by the driller.

The importance of the roof coal thickness has been discussed elsewhere in this report; in many cases it resulted in a modification of the height of the lowermost explosive deck. This thickness was calculated by subtracting the "TOC" column value in Table 5-4 from the "DEPTH" value and entered in the column headed "ROOF COAL".

As mentioned previously, the depth to the floor of the underground opening was also determined at the blasthole measurement stage. This is recorded in the column marked "BOTTOM" on the Blast Summary Sheet. The void depth is thus this depth minus the depth to void. The significance of the "VOID DEPTH" result will be described in Chapter 8 of this report.

5.5.2. Magazine Inventory

Throughout the country there exists a legal obligation on the part of users of explosives to keep, and regularly update, magazine inventories.

During the 1986 AML blasting testwork magazine inventory was kept for all of the product types used. An example of a typical magazine inventory sheet is given in Table 5-5. This is the actual sheet employed to inventory bagged ANFO during the testwork period.

Regulations require that bulk explosives inventory be recorded in pounds; for practical purposes, this was achieved by use of a bag count at start and completion of a blast day.

Items such as delays and boosters were counted on an individual basis. It is advisable to check that the actual contents of a "full" box of delays agrees with the number printed on the carton. If there was a shortfall from the packing at the manufacture point, this can cause problems later for the user of the explosives when the discrepancy becomes apparent.

PRODUCT ECONO-MIX SIZE 50# BAGS

Job Site	Supplier	Date	Code	Received	Used	Balance	Remarks
				LBS	LBS/BAGS	LBS	BAG BALANCE
BEULAH AML	Schwab Supply Inc	9-23-86	SEP12861	36000	—	36000	720
		9-25-86	"	—	1325/26½	34675	693½
		9-26-86	"	—	1800/36	32875	657½
		10-7-86	"	—	2900/58	29975	599½
		10-9-86	"	—	5725/114½	24250	485
		10-13-86	"	—	3000/60	21250	425
		10-14-86	"	—	1450/29	19800	396
		10-16-86	"	—	1550/31	18250	365
		10-17-86	"	—	2400/48	15850	317
		10-20-86	"	—	1150/23	14700	294
		10-21-86	"	—	4250/85	10450	209
		10-24-86	"	—	3450/69	7000	140
		10-28-86	"	—	3200/64	3800	76
BEULAH AML	Schwab Supply	10-30-86	NOV4198.5	6000	5200/104	4600	92
		10-31-86	"	—	3350/67	1250	25
BEULAH AML	Schwab Supply	11-3-86	OCT31 1986	-11000	1450/29	10,800	216
		11-5-86	"	—	1400/28	9400	188
		11-6-86	"	—	2200/44	7200	144
		11-17-86	"	—	2950/59	4250	85
		11-18-86	"	—	1300/26	2950	59
		11-19-86	"	—	450/9	2500	50
		11-20-86	"	—	750/15	1750	35

TABLE 5-5: TYPICAL EXPLOSIVES MAGAZINE INVENTORY SHEET FOR USE WITH AML BLASTING WORK

Details from these inventory sheets were also entered in the Blast Summary sheets described in the previous section of this report.

5.5.3. Field Sketches and Notes

In a research application there is an obvious requirement to keep field notes relating to specific details about each blast which were not recorded on other field record sheets. This would include information such as the following:

- any holes which were not loaded as planned due to caving of the hole after plugging, or hang-ups of explosive, etc.

- any errors in loading which were subsequently noticed (for example, use of incorrect delay number in a particular deck)

- changes in "standard" loading procedure for a hole of that depth due to the presence of rock layers, high roof coal thickness, etc.

- a sketch of the blast layout, showing the relative positions of blastholes identified by their hole numbers, the burden and spacing of blastholes, the tie-in sequence, and the blast initiation point.

- any indication of the room blasted being of non-uniform width (for example, indication during drilling that a blasthole was in, or near to, a pillar)

- after each blast, some general notes should be made regarding the apparent direction of caving, and once the blast has been inspected, notes concerning the overall effect of the blast, whether it caved over its entire length, whether there was surface heave, the approximate depth of the sinkhole created, etc.

In a non-research application the need for record keeping is equally important. While records may not be as exhaustive, all of the points mentioned above are of importance. Records are necessary in order that any less successful blasts or portions of blasts can be rationalized in terms of local variations in overburden conditions or blasting practice. Having done this, there is a better basis from which to consider variations in blast design to remedy any problems.

5.5.4. Project Blast Summary

Any AML blasting project will require some form of summary

showing the basic statistics for each blast. A blasting summary sheet for an AML project is shown in Table 5-6.

Each blast is identified by its number and location, and the date it was shot. The number of blastholes shot is recorded, as is the hole diameter. The consumption of explosives and explosives accessories is also tabulated.

Most blasting projects will require use of an engineering seismograph; columns are included to indicate the location (distance) of the seismograph relative to the blast, and the recorded maximum peak particle velocity (PPV).

Table 5-6 included columns to indicate the location of the high-speed camera relative to the blast, the target down-the -hole delay number, and the framing speed at which the camera was set. This will not normally be required for non-research applications.

AML PROJECT - BLAST SUMMARY

#	DATE	LOCATIONS	HOLE DIAM	NO.OF HOLES	EXPLOSIVES USED: ANFO BAGS LBS	EXPLOSIVES USED: SLURRY BAGS LBS	EXPLOSIVES USED: BOOSTERS	DELAYS USED: 42 MS	DELAYS USED: #4	DELAYS USED: #5	DELAYS USED: #6	#7	#8	#9	HIGH SPEED FILM: CAMERA	HIGH SPEED FILM: DELAY #	HIGH SPEED FILM: FPS	SEISMOGRAPH: SEISMOG	SEISMOGRAPH: RPPV
1																			
2																			
3																			
4																			
5																			
6																			
7																			
8																			
9																			
10																			
11																			
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16																			
17																			
18																			
19																			
20																			

TABLE 5-6: TYPICAL BLAST DATA SUMMARY SHEET FOR AML BLASTING WORK

6. HIGH-SPEED CAMERA STUDIES

6.1. INTRODUCTION

From the discussion of the theoretical aspects of AML blasting technique made elsewhere in this report it should be apparent that the correct sequencing of events within a blast is critical. This applies particularly to the correct order of initiation of decked charges within a blasthole such that successful cratering takes place. In addition, as is the case with just about all blasting operations, it is important to ensure that the blastholes go off in the correct sequence. From the point of view of controlling blast vibrations within acceptable limits, it is also necessary that delay intervals between, and within, blastholes are correct.

Due to the speed at which these events take place, the only practical and quantitative method available to study them is the use of high-speed photography. A brief description of the principles and methodology involved will be given in this chapter, together with presentation and discussion of the results of the high-speed camera study carried out as part of the 1986 AML blasting testwork.

The aims of this study are outlined briefly below:

1. Determination of actual blast sequence.

   This simply consists of a check that the blastholes went off in the planned order, and that there were no cross-propagation or misfire situations. It can be quantitative, with the time for the onset of surface movement (or vertical movement on targets if used) determined from the high-speed film. In many cases a visual check along the direction of the blast using a slow-motion run through the film is sufficient. Misfires or incorrect blasthole sequencing may affect the overall success of a blast, especially in a case such as this where cratering theory is so dependent on the existence of successive blast "free faces".

2. Determination of the accuracy of surface delays.

   This involves the quantitative measurement, using a stop-frame film projector, of the time interval between observed surface delay flashes from the high-speed film. The observed spread of delay times is especially important when the minimization of blast vibrations is required, since it will control the weight of explosive charge that is initiated at any given instant.

3. Determination of the accuracy of down-the-hole delays.

This consists of a quantitative study of the time which elapses between the initiation of the blasthole (as indicated by the surface delay flash) and that of a targeted deck delay within the hole. A "spread" of down-the-hole delay times for one delay interval that overlaps the previous or next delay in the hole will almost certainly lead to the unsuccessful crater blasting of that hole.

4. Determination of the rate of surface movement.

This is the measurement of the rate of surface heave from the top deck of a blasthole. It is achieved by measuring the vertical displacement that took place within a known time interval.

5. Study of the general timing of the blast-caving sequence.

In some of the films shot at a lower frame speed there was sufficient film available to record all of the blast, plus the time interval before the onset of caving, and the caving action itself. Where this is the case, a semi-quantitative measure of the timing of the blast-collapse sequence is possible.

6.2. FIELD TECHNIQUES

6.2.1. Targeting Down-the-hole Delays

In order to obtain on surface an indication of when a single deck within a blasthole is initiated, it was necessary to tie a strand of Nonel noiseless trunkline to the booster in that deck. This was then dropped down the hole with the booster, with the other end tied off at surface. Once the hole loading was complete, a masonite target was fixed in the ground at the blasthole location by means of a wooden stake; these targets were painted a bright color for ease of visibility during the high-speed film analysis. The Nonel trunkline target was then fixed to the target using tape.

In order that the accuracy of down-the-hole delays may be successfully measured, it is essential that the target Nonel trunkline length be initiated only by the booster connected to the targeted delay, and not by the Primacord or by any other lengths of Nonel in the hole. During hole loading, therefore, it is essential that the Nonel trunkline be kept separate from the main Primacord initiation line.

It was initially attempted to target every deck in a blasthole, and run each length of Nonel trunkline to surface. This is illustrated in Fig. 6-1. However, this proved to be extremely time-consuming, and considerable doubts existed as to he ability to separate each strand from the other, and from the main Primacord initiation line. From then on the practice employed was the targeting of a single delay period for any given blast.

By running the Primacord and the Nonel target line on opposite sides on the blasthole, separated by the explosive and stemming columns, it was felt that there was the greatest chance of success. The Nonel trunkline running to a lower deck would not be initiated by a higher deck, since the higher deck would always be detonated later.

It was found from the earlier blasts that were studied in this way that the flash given off by the trunkline was not very visible when it was run straight up the face of the masonite target. For this reason, subsequent targeting was carried out using a longer length of Nonel, which was coiled and fixed to the masonite as shown in Fig. 6-2.

6.2.2. High-speed Camera Setup

The instrument used to take high-speed films of some of the blasts in this project was a LOCAM Model 51 (DC model) manufactured by the Redlake Corporation. It was capable of taking high-speed films of up to 400 feet in length at shutter speeds of up to 500 frames per second. This is fairly typical of high-speed cameras applicable to field studies.

It is not intended in this report to enter into any details regarding the operation of such cameras - these are available in the appropriate instruction manuals. Nor is it an aim to describe in any great detail the principles and techniques for high-speed photography; these are available from various sources[14].

One of the problems encountered at the test site was the positioning of the camera so that a good view of the entire blast was available. This was due to the generally flat topography, and the height of the vegetation in some areas. Where possible, the camera was located so that the line of vision was perpendicular to the length of the blast. For the type of study carried out this was for reasons of convenience. In studies where quantitative measurement of ground movement is required the angular relationship between camera and blast is rather more important.

Fig. 6-3 illustrates a set of targets which were oriented towards the high-speed camera at an angle oblique to the blast direction. Figure 6-4 shows the camera set-up on its tripod,

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best available copy.

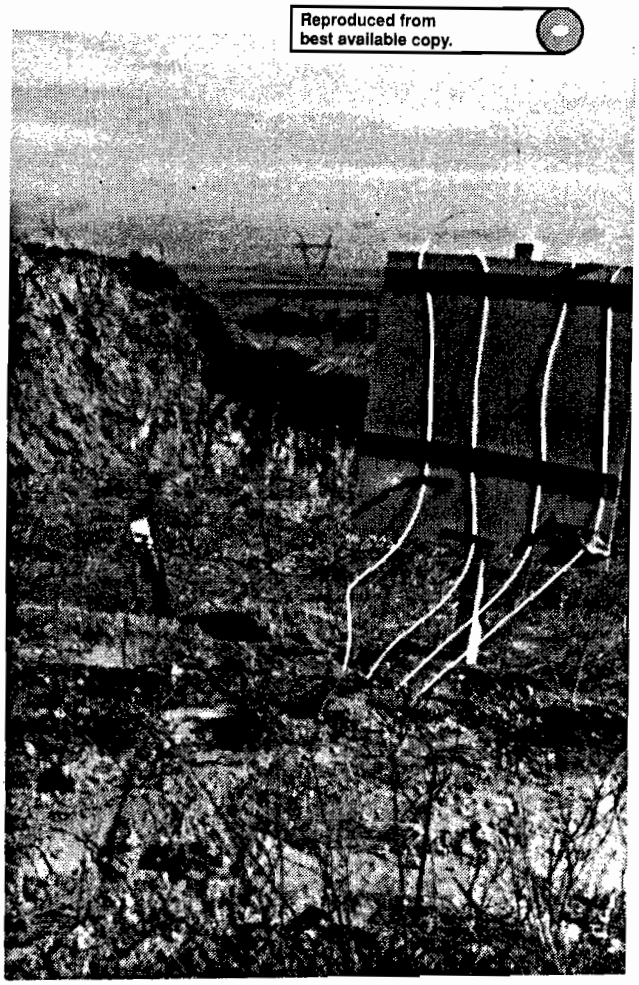

FIG. 6-1: ILLUSTRATING USE OF FOUR NONEL TRUNKLINES USED TO TARGET EACH EXPLOSIVE DECK IN A BLASTHOLE, FIXED TO SURFACE TARGET

FIG. 6-2: ILLUSTRATING COMMON PRACTICE FOR HIGH-SPEED CAMERA TARGETS - SINGLE COIL OF NONEL TRUNKLINE ATTACHED TO ONE DECK IN A BLASTHOLE

with the DC battery pack evident in the foreground. In many cases a remote control switch and cable were employed to move the camera operator back from the immediate proximity of the blast. This cable can be observed running from the battery pack towards the bottom of the photo in Fig. 6-4.

Where possible the fastest framing speed, 500 frames-per-second (fps) was employed, to ensure the greatest possible accuracy of measurement. In some cases, subdued light conditions made the use of lower framing speeds necessary. A light meter was used in conjunction with tabulated data provided with the camera to set the correct f-stop on the camera. A 12-70mm zoom type lens was employed; in some cases relative proximity of camera to blast, combined with blast length, made it difficult to bring all of the targets clearly into the field of view.

Whether the camera was set off remotely, or from the battery pack unit itself, it was started up about 3 seconds before the blast was initiated. This was to ensure that the motor had accelerated to the speed appropriate to the number of frames per second that were set on the dial by the operator. Whenever possible, therefore, the camera operator was located at the same place as the man operating the blasting machine.

The film used for the study was 16mm color video news film (daylight type), ASA 160. This was loaded and developed in the same way that one would expect for conventional movie film - the essential difference is the much shorter filming time duration available from a 100 foot film due to the speed at which frames were shot by the LOCAM camera.

6.3. ANALYSIS OF HIGH-SPEED FILMS

6.3.1. Summary of Films Taken

A total of 14 high-speed films were taken during the 1986 AML blasting testwork. In some cases the targeting of down-the-hole delays was not carried out due to time constraints - in these cases it is still possible to obtain general information about the blast sequence, and about surface delay intervals.

It was found that 9 of the films taken yielded useful information. Of the remaining 5 films, some were adversely affected by some initial problems experienced with the camera battery pack during operation. Others were shot rather late in the day, and although flashes from delays are visible, it is very difficult to correlate the flashes with actual hole locations in the blast tie-in. Some problems were experienced in the last blast filmed due to the extreme cold - it appears that the battery pack and/or the switch mechanism of the camera do

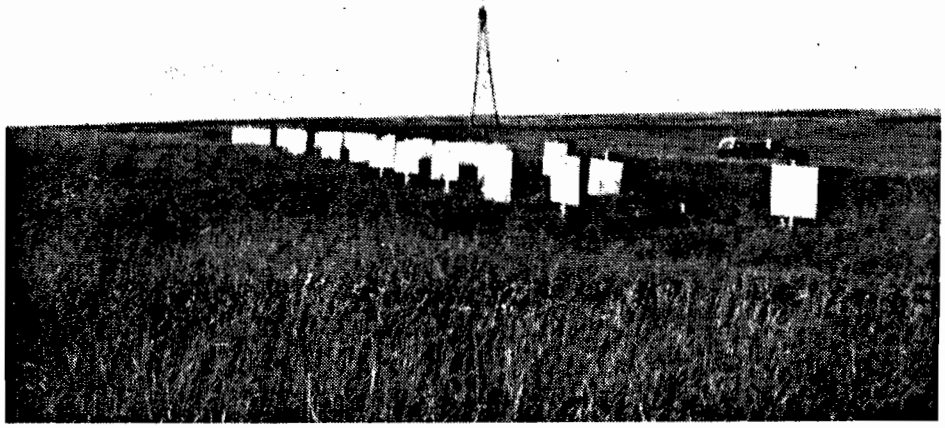

FIG. 6-3: ILLUSTRATING SET OF SURFACE TARGETS EMPLOYED FOR HIGH-SPEED CAMERA STUDIES

FIG. 6-4: HIGH-SPEED CAMERA SETUP, SHOWING TRIPOD, DC BATTERY PACK AND REMOTE CABLE (LEADING TOWARDS THE PHOTOGRAPHER)

not function correctly at sub-zero temperatures.

Several of the films were shot at 500 frames per second (fps), though in some cases poor light forced a reduced framing speed. A summary of pertinent data regarding high-speed camera work carried out for this project is presented in Table 6-1.

For each film analyzed, up to two events may be recorded per blasthole: the flash from the 42 ms surface delay linking the hole to the next, and the flash of the Nonel noiseless trunkline connected to a particular deck in the hole. In one case it was possible to observe the flash of the actual blast initiation using an electric blasting cap, and in two cases there was sufficient film available to record part or all of the blasting and caving sequence.

TABLE 6-1 : SUMMARY OF HIGH-SPEED FILMS TAKEN DURING 1986 AML BLASTING TESTWORK.

Blast #	Blast Location	Camera Location	Film Speed	Target Delay	Comments
5	N-13	300' SW	500	-	Poor light. No useful data
6	N-8(S)	300' W	225	7	Good data
7	N-7	250' W	220	7	Good data
8	N-12	300' E	265	4	Good data for N. part
9	NC-7	300' SW	500	5	Battery/switch malfunction
10	S-1(N)	250' S	200	-	Poor light and position
11	S-12	400' NW	110	6	Surface delays + caving
12	N-8(N)	300' W	500	8	Excellent data
13	N-10	400' E	300	7	Good data
14	S-11(N)	350' NW	500	9	Good data
15	S-1(S)	200' W	200	9	Poor light and position
16	S-11(S)	300' E	400	4	Excellent data
17	S-10	350' E	310	6	Good data
21	S-9(S)	400' E	500	8	Battery/switch malfunction

6.3.2. Use of Stop-frame Projector

High-speed films may be viewed using a conventional 16mm movie projector. Since these operate typically at about 24 fps, it is possible to observe these films, slowed up 80-90 %, and make some overall qualitative judgements regarding blast sequence and misfires. However, meaningful analysis can only be carried out using a special stop-frame projector.

The instrument employed for this analysis was a Photo-optical Data Analyzer, Model 224A, manufactured by L-W International. Again it is not appropriate in this report to describe the construction or operation of this projector in any detail.

The machine allows the frames to be viewed one at a time, or at varying speeds up to 24 fps. It incorporates a counter, which can be zeroed at an appropriate position in the film, and which registers one count for each frame passing through the instrument. It has the ability to run in forward and reverse motion, and the image may be projected onto a screen in exactly the same way as a conventional movie projector.

6.3.3. Analysis Methodology

Each film was initially viewed at 24 fps to obtain an overall idea of the quality of the information obtained. Frames that were shot during camera testing, and during the 3 second run-up to blast initiation were cut from the beginning of the film such that about 200 frames remained before the blast start. A "zero" position was then marked onto the film approximately 50 frames before initiation.

The film was projected onto a rigid screen, to which was fixed a sheet of graph paper. The projection was "frozen" at a frame in which most of the masonite targets showed up clearly. The positions of these were marked on the graph paper, and they were identified by hole number according to the blast sketch (see Appendix A).

The frame counter on the stop-frame projector was zeroed at the position marked on the film, and then run at one frame per second through the firing sequence. The frame number of each observed event (surface delay or targeted down-the-hole Nonel flashes) was marked on the graph paper. The film was reversed and run again to check this, and to pick up any additional events that escaped notice during the first run.

In most cases the 100 feet of film shot was enough to record all of the flashes in the blasting sequence, the ground surface heave resulting from initiation of the top deck, and most of the subsequent fall of this ground heave. In one or two cases where the film speed was lower it was possible to observe the onset of ground caving. Where this was the case, the approximate frame number where this occurred was also recorded.

For two blasts where the camera was oriented perpendicular to the length of the shot, and the film was more sharply focussed, a semi-quantitative measurement of the rate of ground surface movement was taken. This involved plotting on graph paper the outline of one of the targets, the size of which was known and could therefore be used for scaling purposes. The

position of the top edge of various targets was marked, with the corresponding frame number, at different points during its ascent following initiation of the upper explosive deck.

The measurement of the displacement over a known number of frames (which can be equated to a time interval) allows a calculation of the approximate rate of ground heave to be calculated. Results of this measurement are presented in section 6.3.6. of this chapter.

The final stage in the high-speed film analysis was the measurement, from the film itself, of the average spacing between timing marks. The significance of this is explained in the next section of this report.

6.3.4. Analysis Principles

Detailed description of the analysis principles for high-speed films may be found elsewhere[14]. A brief summary of those principles relevant to this analysis are presented here.

During filming of a blast the high-speed camera places timing marks on the edge of the 16mm film at a frequency of 100 per second. It can be appreciated therefore that the faster the film speed, the greater will be the spacing of these timing marks. The relationship between timing mark frequency, timing mark separation and film speed is given by the following expression:

$$\text{Time interval between frames} = \frac{\text{Width of one frame} \times 1000}{\text{Timing mark freq.} \times \text{Timing mark separation}}$$

The width of a single frame is constant, at 7.605 mm, and the timing mark frequency is known to be 100 Hz. As such, it is possible to simplify the expression to the following:

$$\text{Time between frames (ms)} = \frac{76.05}{\text{Timing mark separation (mm)}}$$

It is then possible to check that the film speed during the blast corresponded to that for which the camera was set, according the following relationship:

$$\text{Film speed (fps)} = \frac{1000}{\text{Time between frames}}$$

The delay interval between two events observed in the film is calculated by multiplying the difference in frame numbers by the time interval between frames. It is possible for the filming speed to change slightly during a blast; this is especially the case if insufficient time is allowed for the camera motor to get up to speed prior to initiation. For this reason, the timing mark separation was measured at various points on the film during the blast duration.

The accuracy of the calculated time interval between events is very dependent on the film speed. The greater the film speed, the shorter the time interval between frames, and the greater the degree of accuracy that may be obtained. If the separation of timing marks is variable, indicating that the film speed was varying during the blast, then calculated delay intervals will be suspect. In general, though, the calculated delay interval, and its associated degree of error, can be expressed as follows:

$$\text{Delay interval} = \left[\frac{\#\text{ frames}}{\text{b/t events}} \times \frac{\text{Time}}{\text{b/t frames}}\right] \pm \text{Time between frames}$$

6.3.5. Spreadsheet for Data Analysis

For means of calculation and tabular presentation, a spreadsheet was designed using the Lotus 1-2-3 package for input of the data obtained from work with the stop-frame projector. An example of this spreadsheet is presented as Table 6-2, in order to assist with the following description of its function.

6.3.5.1. Input Data

Each blast is identified by its number and location. The number of holes in it is recorded, and the target down-the-hole delay number where relevant. The film speed set on the camera is indicated, together with the approximate camera location. In cases where it is required to make accurate measurements of ground movement rates, it is necessary to record very accurately the distance from camera to blast, the angle of sight, and the focal length used for the lens. In general this was not carried out for this study, as the major interest was only in the accuracy of delays.

In the column headed "Hole #" the blasthole numbers, as identified on the appropriate blast sketch (see Appendix A) are entered, in the order that they were initiated. In some cases the blast configuration required that two holes be initiated simultaneously, but in general one blasthole per delay was aimed for.

As stated earlier, two events were studied: the flash of the surface delay indicating initiation of the blasthole, and

AML PROJECT - HI-SPEED CAMERA STUDIES

BLAST # : 12 LOCATION : N-8(N)

NO. HOLES : 17 TARGET DTH DELAY # : 8

FPS (SET) : 500 CAMERA : 300' W LENS : ? MM

		SURFACE DELAYS				DOWN-THE-HOLE DELAYS			
HOLE #	FILM SPEED (FPS)	FRAME #	MARK SEPRN. (MM)	DETN. INTVL (MSEC)	CUM. TIME (MSEC)	FRAME #	MARK SEPRN. (MM)	DETN. INTVL (MSEC)	CUM. TIME (MSEC)
INIT	504.9	62	38.40		0.0				0.0
1	504.9	80	38.40	35.6	35.6				
18									
2									
17	503.6	141	38.30	40.3	156.7	141	38.30	0.0	156.7
3	503.6	160	38.30	37.7	194.3	262	38.40	202.3	396.1
16	504.9	180	38.40	39.7	233.7	282	38.40	202.0	435.7
4	506.2	199	38.50	37.6	271.0	303	38.50	205.4	476.7
15	506.2	218	38.50	37.5	308.6	326	38.50	213.3	522.2
5	504.9	238	38.40	39.6	348.6	238	38.40	0.0	348.6
14	504.9	257	38.40	37.6	386.2	361	38.40	206.0	592.2
6	504.9	274	38.40	33.7	419.9	380	38.40	209.9	629.8
13									
7	506.2	313	38.50	38.6	496.5	419	38.40	209.7	707.0
12	506.2	330	38.50	33.6	530.1	331	38.50	2.0	532.1
8						453	38.50		
11									
9						492	38.50		
AVG	505.2			37.4				206.9	
N	12	12				12			

TABLE 6-2: EXAMPLE OF SPREADSHEET USED FOR CALCULATION OF RESULTS FROM ANALYSIS OF HIGH-SPEED FILMS

the flash of the Nonel trunkline representing initiation of the targeted down-the-hole delay. As explained in Chapter 5, the Primacord downline for a blasthole was fixed to the end of the 42 ms surface delay at the end containing the surface delay cap. The J-hook of the next surface delay was connected to the primaline at the hole collar. Therefore, it is the surface delay Nonel tube running AWAY from the blasthole that records when the hole was initiated.

The frame number for each event is recorded in the appropriate columns marked "Frame #". The average timing mark separation for the part of the film containing that frame number is entered in the adjacent column titled "Mark Seprn.".

Where blanks exist in the spreadsheet, this represents blastholes for which events were not observed for one reason or another. In the case illustrated in Table 6-2 it was possible to observe the initiation of the first surface delay, that which was set off by the blasting cap and which led to the initiation of the first blasthole 42 ms later. Such cases are indicated by "INIT" in the hole number column. In one case the actual flash from the electric blasting cap was observed - this is indicated by "FIRE" in the first column of the spreadsheet (see Appendix C).

6.3.5.2. Calculated Data

The actual film speed is calculated, as described in the previous section of this report, for the frame numbers at which surface delay events were recorded. The column for "Film Speed" is included in the spreadsheet, therefore, to indicate periods where the film-speed may be fluctuating significantly, and where the delay periods calculated may thus be suspect.

The calculation of the detonation interval for surface delays takes into account possible film-speed variation by employing the following formula:

$$\text{Detn. Intvl.} = (F\#_n - F\#_{n-1}) \times \frac{76.05}{(MS_n + MS_{n-1})/2}$$

where $F\#_n$ = Frame number of current event

$F\#_{n-1}$ = Frame number of previous event

MS_n = Timing Mark separation for current event

MS_{n-1} = Timing Mark separation for previous event

The average time interval between frames is thus calculated, and multiplied by the number of frames between events.

The cumulative time for each event is also calculated. Where the initiation of the first surface delay was visible, this is established as the "zero" for cumulative time calculation. When the initiation point is not visible from the film, the first visible event is established as the zero time. The cumulative time interval for surface delay initiation is thus calculated by the spreadsheet as follows:

$$\text{Cum. Time} = (F\#_n - F\#_{zero}) \times \frac{76.05}{(MS_n - MS_{zero})/2}$$

where $F\#_n$ = Frame number of current event

$F\#_{zero}$ = Frame number of zero time event

MS_n = Timing Mark separation for current event

MS_{zero} = Timing Mark separation for zero time event

Again this method tends to "smooth out" any irregularities in the film-speed between the zero time and calculated events. This same calculation is carried out to obtain the cumulative time for down-the-hole delay events.

Down-the-hole delay intervals are calculated by subtracting the frame number associated with the target flash from the frame number at which the blasthole was initiated. It thus gives a direct measure of the down-the-hole millisecond delay period. The formula employed is:

$$\text{Detn. Intvl.} = (DHF\#_n - SF\#_n) \times \frac{76.05}{(DHMS_n + SMS_n)/2}$$

where $DHF\#_n$ = Frame number of D-T-H event

$SF\#_n$ = Frame number of surface event for same B/H

$DHMS_n$ = Timing Mark separation for D-T-H event

SMS_n = Timing Mark separation for surface event

There were numerous cases where the surface delay flash and that from the target Nonel were observed simultaneously. When this occurs,the down-the-hole detonation interval is calculated to be zero and the down-the-hole cumulative time is equal to that for the surface delay.

The spreadsheet finally calculates the average value for film-speed, and the average surface and down-the-hole delay times, in the row marked "AVG". It also records the number of events, N, that went into each average value. In some of the

tabulated results in Appendix C it was decided to manually omit some obviously erroneous values when taking these averages. This is discussed in the next section.

6.3.6. Results of Analysis of High-speed Films

General results from the analysis of high-speed films taken during field testwork for this project are summarized in Table 6-3. Detailed results are included as Appendix C.

TABLE 6-3 : SUMMARY OF RESULTS FROM ANALYSIS OF HIGH-SPEED FILMS

Blast #	Blast Location	Av. Film Speed (fps)	Surface Delays		Down-the-hole Delays		
			# Events	Av. Int.	No.	# Events	Av. Int.
6	N-8(S)	226	5	34.4 ms	7	2	171.9 ms
7	N-7	218	7	39.2 ms	7	3	176.9 ms
8	N-12	263	9	38.5 ms	4	1	133.4 ms
11	S-12	109	8	43.0 ms	6	0	-
12	N-8(N)	505	11	37.4 ms	8	7	206.9 ms
13	N-10	302	15	39.2 ms	7	10	181.4 ms
14	S-11(N)	490	9	37.4 ms	9	2	255.6 ms
16	S-11(S)	410	12	35.4 ms	4	2	100.1 ms
17	S-10	309	11	34.7 ms	6	4	168.3 ms

6.3.6.1. Determination of Blast Sequence

It was established, from a combination of qualitative and quantitative measurement, that the blast sequence for each of the 9 blasts studied in detail was as planned. There were no apparent misfires, nor was there any evidence of holes going off out of sequence.

6.3.6.2. Accuracy of Surface Delays

A considerable amount of data was obtained which allows the accuracy of the 42 ms surface delays used during testwork to be determined. A statistical analysis of all of the observed surface delay events, with one or two suspect values omitted, gave the following results:

# events	Mean	Standard Deviation
85	37.3 ms	3.61 ms

Thus it can be seen that the surface delays were, on average, about five milliseconds faster than claimed.

6.3.6.3. Accuracy of Down-the-hole Delays

Rather less information was obtained for determining the accuracy of the down-the-hole delays used. There are a number of reasons for this, including the following:

- only one delay number per blast was studied

- Nonel noiseless trunkline was not used to "target" a down-the-hole delay number in every blast filmed, but only where time permitted.

- in many cases it appears that the Nonel trunkline used to indicate when the deck went off was in fact set off by the primacord, not by the initiation of the deck itself. As such, therefore, the flash from the targeted Nonel coincided with that from the surface delay for a particular hole.

- the burying of surface delays, which was used as a measure to reduce noise, caused in some cases a small plume of stemming, kicked up by the initiation of the 42 ms delay cap, to obscure the target and the coil of Nonel trunkline.

Nevertheless, data was obtained for down-the-hole delay numbers 4,6,7,8 and 9 (delay periods 100, 150, 175, 200 and 250 ms respectively).

Results from this analysis are summarized in Table 6-4. From these results it can be seen that the down-the-hole delay numbers 7 and above were reasonably accurate, though two few data are available for a statistical analysis of standard deviation to be meaningful. Thus it is not possible to estimate the probable "spread" on down-the-hole delay times.

The results from #4 delay are confusing; however, two observations indicated a high accuracy on this 100 ms interval. The #6 delays definitely seem to be inaccurate, and approached the theoretical #7 delay period. In blasts where #7 and #6 delays were used for the top and second decks, respectively, these decks may have gone off almost simultaneously.

Unfortunately, the films taken when #5 delay was targeted did not yield useful results.

TABLE 6-4 : STATISTICAL ANALYSIS OF DOWN-THE-HOLE DELAY PERIODS ANALYZED FROM HIGH-SPEED FILMS.

Target Delay #	Theoretical delay interval	# Events observed	Actual aver. delay interval	Standard deviation
4	100 ms	4	124.3 ms*	30.6 ms
5	125 ms	0	n/a	n/a
6	150 ms	5	176.0 ms**	18.0 ms
7	175 ms	15	179.2 ms	5.1 ms
8	200 ms	7	206.9 ms	4.2 ms
9	250 ms	2	255.6 ms	1.8 ms

* Highly variable results from #4 delay. Two out of the four observations gave average of 100.2 ms.

** If one suspect value is omitted, average interval for #6 delay from 4 observations is 168.3 ms, standard deviation 5.9 ms.

6.3.6.4. Rate of Surface Movement

In two of the high-speed films a quantitative study was made of the rate of surface movement from the top deck. This was achieved by measuring the vertical displacement of a target for a given blasthole between a known number of frames.

In addition, profiles were drawn showing the surface level at successive periods during the blast. These indicated that the surface vertical displacement was even along the blasts, and provides further evidence that the blast sequence was as planned and that there was no misfiring of the top deck.

This analysis is semi-quantitative only; results are presented on the following page. The positions of targets for the studied blastholes for each of the studied blasts, and the approximate ground profiles, are illustrated in Fig. 6-5.

Measurement of surface movement can generally be regarded as being less accurate than that of targets adjacent to boreholes, which present "sharp" lines which can readily be identified and plotted during study. However, it is possible that target movement could be faster than true surface movement if there occurs a "rifling" action - that is, if the wooden target was actually ejected from the ground by venting explosive energy near to the surface.

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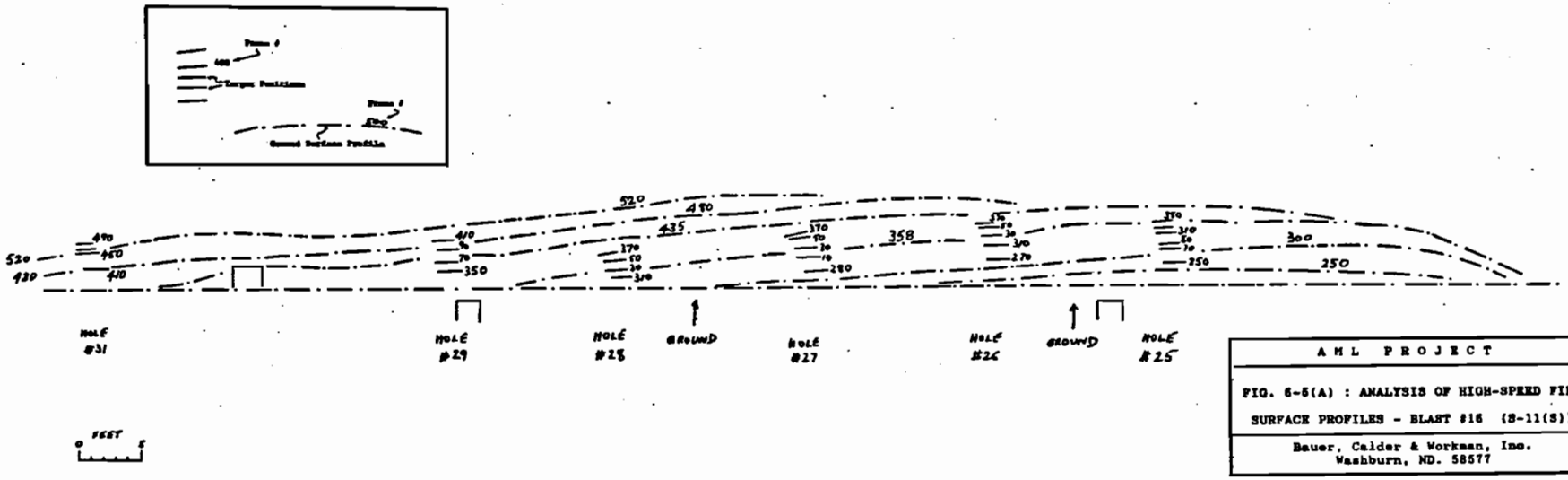
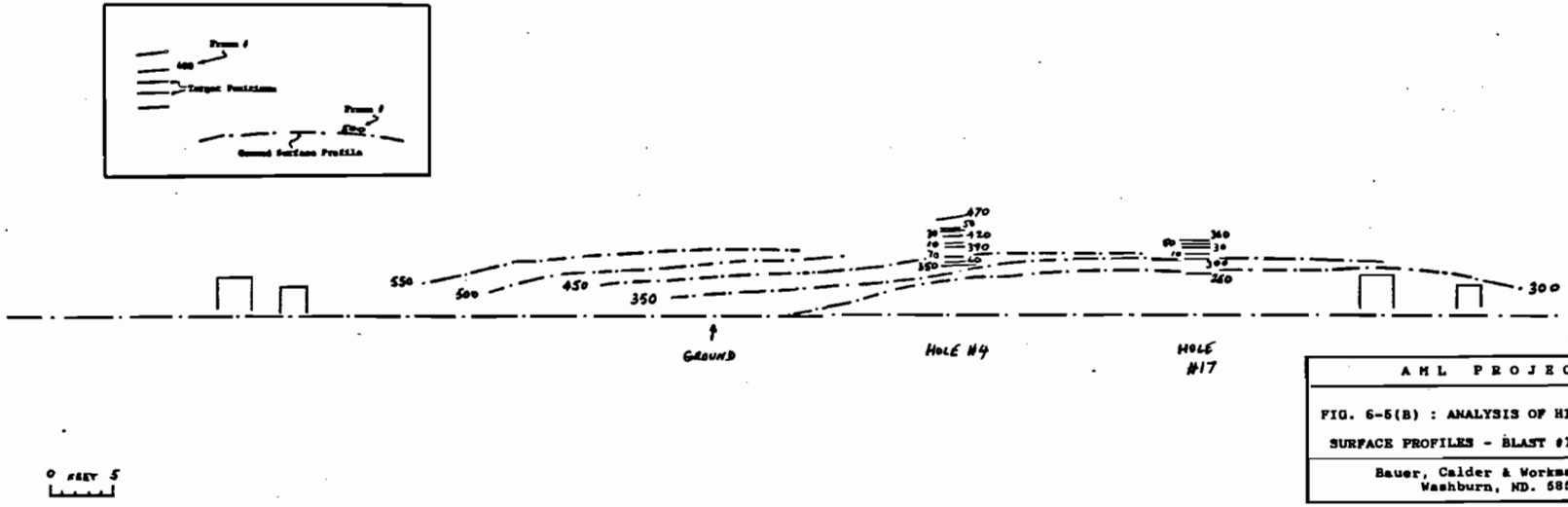


AML PROJECT

FIG. 6-5(A) : ANALYSIS OF HIGH-SPEED FILMS
SURFACE PROFILES - BLAST #16 (S-11(S))

Bauer, Calder & Workman, Inc.
Washburn, ND. 58577



AML PROJECT

FIG. 6-5(B) : ANALYSIS OF HIGH-SPEED FILMS
SURFACE PROFILES - BLAST #12 (N-8(N))

Bauer, Calder & Workman, Inc.
Washburn, ND. 58577

TABLE 6-5 : SUMMARY OF ANALYSIS OF RATES OF GROUND SURFACE MOVEMENT

Blast	Hole # /Event	Delay #'s in top 2 decks	Rate of movement (ft/sec)		
			Min.	Max.	Average
S-11(S)	#31	8, 9	8.4	16.8	13.4
	#29	6, 8	20.1	25.2	21.8
	#28	6, 8	15.1	15.1	16.8
	Surface	-	-	-	16.6
	#27	6, 8	11.8	21.8	17.9
	#26	6, 8	8.4	20.2	15.5
	Surface	-	-	-	14.5
	#25	6, 8	10.1	30.2	13.8
N-8(N)	Surface	-	-	-	8.9
	#4	7, 8	9.9	19.9	19.3
	#17	7, 8	8.8	19.9	13.3

Results from the above table would tend to indicate that this was not a problem for the first blast studied, where both ground surface and target movement rates were in the order of 15 to 20 feet/second on average.

Rather less information was obtained from the film of blast N-8(N), however, similar results to S-11(S) were obtained.

6.3.6.5. General Timing of Blast-caving Sequence

In three of the studied films it was possible to obtain information regarding the timing of the blast-caving sequence. The results of this are presented in Table 6-6.

From these results it can be seen that there is a delay of between 1.7 and 2.1 seconds, after the actual blast movement has stopped, before the onset of caving. The caving action itself took about 6 seconds in one case, and 4.5 seconds in another. These times can be linked to a certain extent with the lengths of these blasts, which were 185 feet and 145 feet respectively.

6.3.7. Conclusions with respect to High-speed Camera Studies

The purpose of this section is to attempt to relate the results from these high-speed camera studies to some of the practical and theoretical aspects of AML reclamation by blasting. In particular, it is important to consider the effect that any unexpected results may have had on the successful

TABLE 6-6 : GENERAL TIMING OF BLAST-CAVING SEQUENCE FROM THREE HIGH-SPEED FILMS

BLAST LOCATION	END OF BLAST MVMT.	START OF CAVING	END OF CAVING
S-12	2.9 sec	5.0 sec	11.2 sec
S-11(S)	1.5 sec	3.3 sec	n/a
S-10	1.6 sec	3.3 sec	7.8 sec

caving of the underground openings for specific blasts, or in general.

6.3.7.1. Blast Sequence and Misfires

In none of the high-speed films studied was there any evidence of blastholes firing out of sequence, nor was there of any blastholes not firing at all. This is significant, in that it shows that the overall field methodology with respect to use of surface delays and tie-in was sound. It provides, on the other hand, no explanation for the less successful blasts.

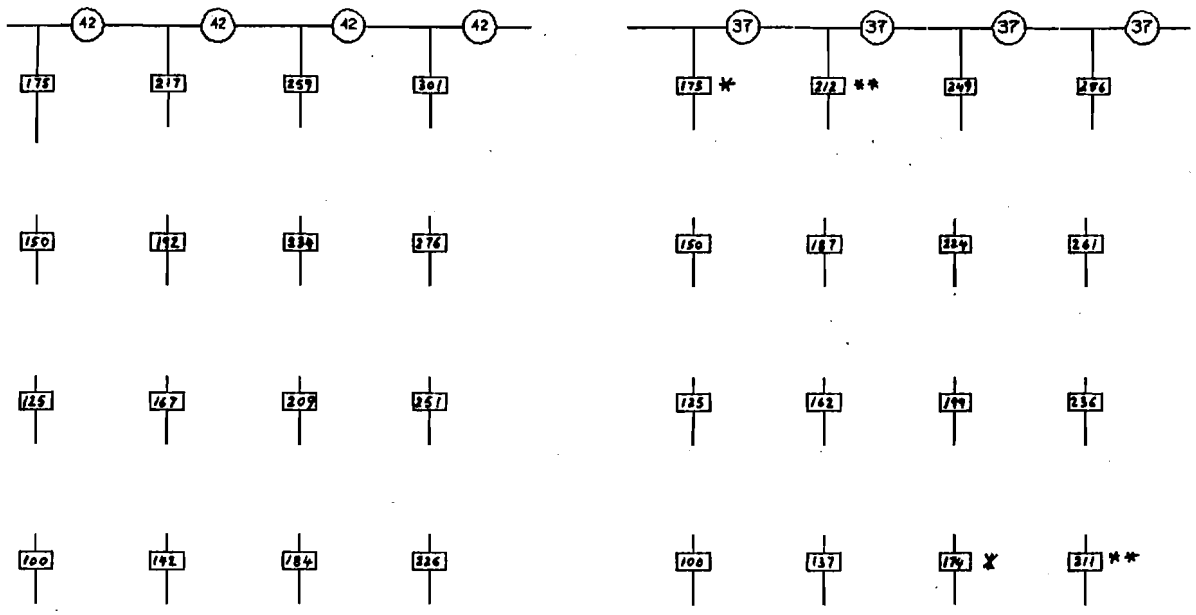
6.3.7.2. Surface Delays

Problems with surface delays as indicated by high-speed photography could cause two types of major problem : incorrect blast sequencing and/or misfires, and blast vibration problems. The first of these was discounted above.

When decked charges are used, as is the case for the AML blasting testwork, blast vibrations resulting from the relatively small explosive weights which are detonated instantaneously are generally not going to be a concern. More comments on this are made in Chapter 7 of this report.

Fig. 6-6 has been prepared to illustrate, however, the potential effect of variable surface delay times on the down-the-hole delay times when decked charges would be initiated. It considers a four hole section of a blast which employs four explosive decks per hole. The average measured interval indicated by the high-speed studies, 37 ms, and the plus and minus one standard deviation values (33 ms and 41 ms) are used in the analysis.

When the 42 ms delays are accurate, and the down-the-hole delays are accurate and separated by 25 ms, there are no two explosive decks that are initiated within 5 ms of one another.



* NEAR-SIMULTANEOUS DETONATION OF BLASTHOLES
**

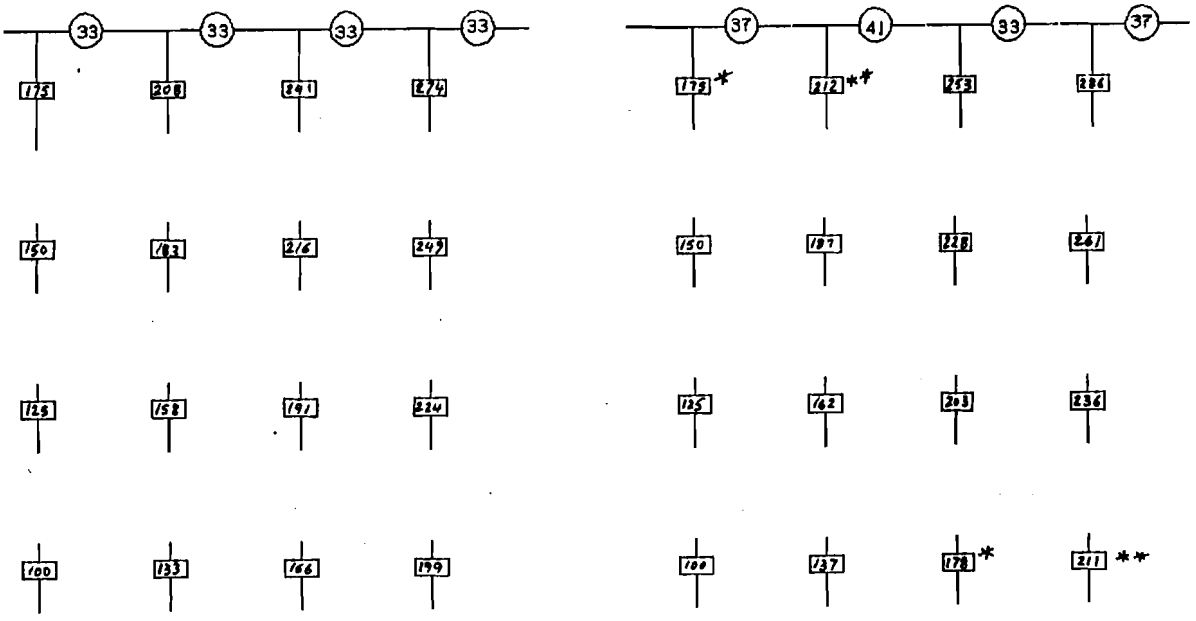


FIG. 6-6: ILLUSTRATING THE EFFECT OF VARIATIONS IN SURFACE DELAY PERIODS ON DOWN-THE-HOLE DELAY TIMES

This is also the case when all intervals are 33 ms. However, if the delay is 37 ms, it can be seen that two decked charges can be initiated almost simultaneously. This is observed in Fig. 6-6 for the top deck of blasthole #1 and the bottom deck of blasthole #3, and for the top deck in blasthole #2 and the bottom deck of blasthole #4. This pattern will in fact repeat itself throughout the blast. The same can happen when a combination of delay times within the observed spread is present.

In some cases blast vibrations may be a concern for AML blasting work, due perhaps to relatively close proximity of buildings. In these instances it is important to consider the possible spread of surface delay times when selecting surface and down-the-hole delays, and designing a blast layout.

6.3.7.3. Down-the-hole Delays

In general the results of high-speed camera work carried out to investigate the accuracy of down-the-hole delays during the AML field testwork were disappointing, and inconclusive, for delay periods 6 and below.

There is reasonable evidence to suggest, however, that delay numbers 7, 8 and 9 were reasonably accurate, though in the latter case only two observations were obtained. Number 4 delay would, in half of the observed cases, appear to be accurate, but there was also wide variation in observed results. Very limited data is available for #6 delay. However, if this data is representative there would definitely appear to be a problem with this delay. There were no results available for # 5.

In any blasting method employing crater theory and decked explosive charges, it is absolutely essential that the down-the-hole delays be accurate. The whole principle, as explained in Chapter 4 of this report, requires the formation of a free-face by the decked charge below the initiating charge.

The fact that some of the numbers 6 and 7 delays appeared to go off together about 175 ms after primacord initiation would thus appear to present a potentially serious problem. This is especially the case where #7 was used in the top deck, and #6 in the next deck down. Two of the blasts resulted in surface heave, with definite evidence of "bridging" of the top two decks (see Chapter 4).

Blast numbers 2 (N-9), 4 (N-1,N-2), 6 (N-8(S)), 7 (N-7), 9 (NC-7) and 10 (S-1(N)) all produced surface heave. All of these blasts employed delay numbers 7 and 6 in the top two decks. There was definite evidence of "bridging" of these two decks for blast numbers 4 and 9. This may possibly have been related to the overlap of delay periods 6 and 7. Other reasons for the surface heave are given in Chapter 8 of this report.

It may be further argued that the evidence for problems with #6 delay (5 observations) is not conclusive by any means. It is suggested that in future work of this kind some effort be made to study further the accuracy and spread of values relative to the claimed manufacturers' specifications for down-the-hole delays.

It is recommended that if conditions are such that noise level is not a major concern, surface delays should not be buried in a future high-speed camera study of this type of work, for reasons described in the above mentioned section.

It is difficult to recommend means by which the initiation of the Nonel target trunkline by Primacord can be avoided, especially in a small diameter blasthole such as 6 inch. Possibly different stemming types could be tried, for example, gravel may result in less cross-propogation between primaline and Nonel trunkline than drill cuttings. HD Primaline (7.5 grain) could be used instead of RX Primaline (15 grain). However, E cord pigtails would be needed at surface to connect with the Nonel NTL delays. Use of Nonel Primadets would solve the problem, but 4 to 5 would be needed for each hole, each of different lead length, which would create complication in the tie-in. In any event, this is a research related problem.

6.3.7.4. Rate of Surface Movement

There is little or no documented information on typical rates of vertical ground movement from six-inch diameter blastholes in material of the types encountered in the AML testwork. Results from hard-rock applications indicate velocities in excess of 30 feet per second to be typical. One would expect, however, unconsolidated "soil-type" materials such as those encountered at the test site to move more slowly.

Rates of vertical ground movement were typically in the range of 13 to 20 feet per second for the two blasts analyzed. The general lack of evidence of "rifling" of blasthole stemming from any of the high-speed films would suggest that collar heights for this work were chosen correctly.

6.3.7.5. Timing of Blast/Caving Sequence

There is no available information with which to compare the results of our analysis of the timing of the blast and caving sequence for this work. The relatively long delay between the end of the blast action and the visible onset of caving is at first surprising. However, it should again be remembered that this was a "soil-type" material, and one might expect different effects in hard-rock applications. Comments regarding the timing of the caving sequence itself were made in Section 6.3.6.5. of this report.

7. BLAST VIBRATION

7.1 INTRODUCTION

Blast vibration was monitored carefully during the field test work. All but four blasts yielded vibration data. The four not measured resulted from equipment malfunction.

The seismograph initially used measured peak particle velocity only. Subsequently a blasting seismograph that measured the particle velocity in each mode, resultant peak particle velocity, vibration frequency and airblast was employed. The basic instrument was used for the first four blasts and the more sophisticated equipment was used for all other blasts recorded.

The seismograph used for most of the field testwork was a Safeguard Seismic Unit 1000D, a microprocessor-based digitizing unit developed by NOMIS Computer Systems Corporation. The unit is illustrated in Fig. 7-1. The seismic head is shown in the foreground, which is fixed to the ground by a metal stake, levelled by means of a surface bubble, and oriented in the direction of the blast using an arrow enscribed on the top.

Relevant information is entered using a touch-pad type keyboard. This included the date, identification, blast location, seismograph location with respect to the blast, and the "trigger" levels for vibration and airblast which would automatically set off the machine. In Fig. 7-2 the instrument is shown from above, with the microphone for airblast detection oriented towards the blast.

The unit was placed at varying distances from the blast. These ranged from 300 to 1,450 feet from the event. The equipment was also placed at varying orientations to the blast to account for differences that might result from the presence of the mined voids.

The nearest residences were along a north-south road west of the property. The closest home was 2,500 feet from the site. Therefore, no problems associated with ground vibration were expected because the scaled distances were quite high.

It was considered important to keep noise and airblast to a minimum. These blasting phenomenon were believed to have a greater potential for distressing local residents and, as such, should be well controlled. For this reason all surface delays, caps and detonating cord pigtails were buried under drill cuttings. Further NONEL surface products were used, which generate little noise.

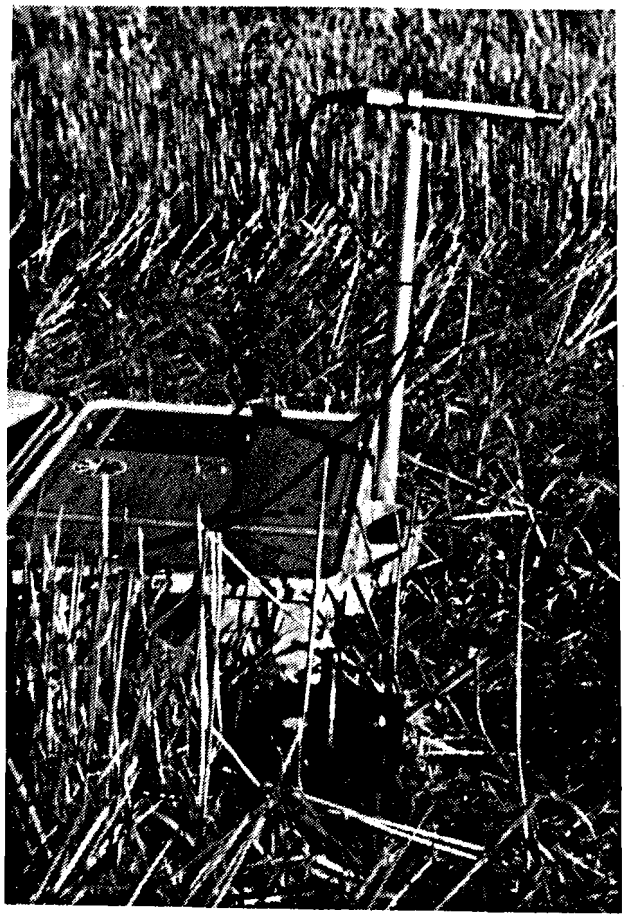


FIG. 7-1: SSU 1000D MICRO-PROCESSOR BASED DIGITIZING SEISMOGRAPH USED DURING BLASTING TESTWORK

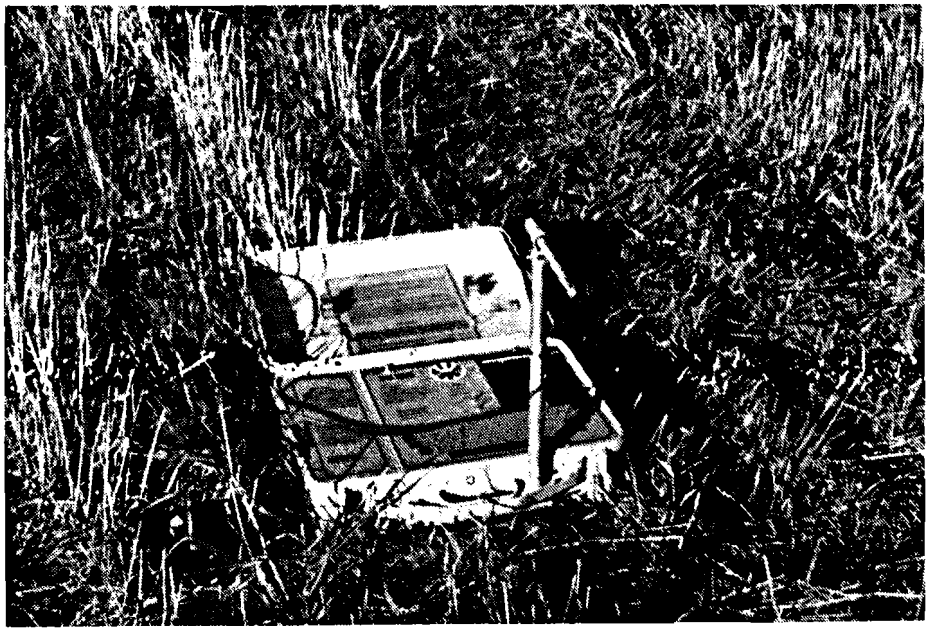


FIG. 7-2: ILLUSTRATING SEISMOGRAPH WITH AIRBLAST MICROPHONE DIRECTED TOWARDS BLAST

7.2 TEST BLAST RESULTS

The blast vibration results are recorded in table 7-1. The vibration levels were invariably modest. Only one result exceeded 0.5 in/sec and in this case the seismograph was positioned only 300 feet from the blast.

Scaled distances are reported for both square root scaling and cube root scaling of the weight. Square root scaling is typically reported in vibration work. However, since the charges were designed as spherical cratering charges with appreciable delays between individual charges it was thought that cube root scaling might be more pertinent in this case.

The minimum scaled distance to a residence was 155.9 $ft/lb^{1/2}$, or 393 $ft/lb^{1/3}$. Referring to table 7-1 scaled distances of this magnitude resulted in peak particle velocities of less than 0.10 inches/second. Such levels of vibration will not result in damage to buildings and are below the levels generally considered to result in the onset of human response.

A table was presented in Chapter 2 that showed the onset of window damage from airblast to occur at 0.03 psi of overpressure. The maximum airblast pressure recorded at the test site was 0.00468 psi. This is an order of magnitude less than the onset of damage. The maximum recording occurred at a scaled distance of 25.8 $ft/lb^{1/2}$ (58.4 $ft/lb^{1/3}$). For scaled distances near to that of the closest house the maximum overpressure was .00204 psi and was more typically two orders of magnitude less than the onset of window damage.

The ground motion frequency range was from 4.7 to 21.3 hertz. The frequency reported is the value for the mode having the greatest particle velocity. These values are typical of the lower frequencies that predominate at larger values of scaled distance. These are near to the typical natural response frequency of most structures and are therefore potentially more damaging. The fact that the method of blasting reported here minimizes charge weights per delay and leads to very low vibration levels is a definite advantage.

Figure 7-3 is a plot of peak particle velocity versus scaled distance, where the scaled distance is the distance to the point of interest divided by the square root of the weight per delay period. The upper limit line is shown, being a line parallel to the trend of the data but drawn such that all measured data lies below the limit line. The line has been fitted by visual inspection since there is insufficient data to warrant a statistical regression analysis.

TABLE 7-1 : RESULTS OF BLAST VIBRATION MONITORING FOR THE BEULAH TEST BLASTING PROGRAM

BLAST #	DATE	LOCATION	DISTANCE TO MEASUREMENT POINT (FT)	MAXIMUM EXPLOSIVE WEIGHT PER DELAY PERIOD (LBS)	SCALED DISTANCE FT/(LB)^1/2	SCALED DISTANCE FT/(LB)^1/3	PEAK PARTICLE VELOCITY (IN/SEC)	FREQUENCY (HZ)	AIRBLAST (PSI)
1	SEP. 25	N-14	1450 SW	41.7	224.5	418.1	0.03	—	—
2	SEP. 26	N-9	1300 SW	57.4	171.6	337.0	0.03	—	—
3	OCT. 7	N-11	1375 SW	73.6	160.3	328.1	0.06	—	—
4	OCT. 9	N-1,N-2	1150 SW	257.0	71.7	180.9	0.23	—	—
5	OCT. 13	N-13	400 E	73.6	46.6	95.4	no reading	no reading	no reading
6	OCT. 14	N-8(S)	675 SE	171.8	51.5	121.4	0.10	21.3	0.00083
7	OCT. 16	N-7	425 S	147.2	35.0	80.5	0.24	6.0	0.00150
8	OCT. 17	N-12	450 E	61.3	57.5	114.1	no reading	no reading	no reading
9	OCT. 20	NC-7	400 N	98.2	40.4	86.7	0.31	18.3	0.00204
10	OCT. 21	S-1(N)	400 W	184.0	29.5	70.3	0.27	4.7	0.00424
11	OCT. 24	S-12	300 N	135.6	25.8	58.4	0.73	13.5	0.00468
12	OCT. 28	N-8(N)	575 E	67.8	69.8	141.0	0.13	9.8	0.00083
13	OCT. 30	N-10	640 W	156.4	51.2	118.8	0.10	9.8	0.00094
14	OCT. 31	S-11(N)	550 N	114.8	51.3	113.2	0.10	11.1	0.00116
15	NOV. 3	S-1(S)	700 E	73.0	81.9	167.5	0.16	12.2	0.00105
16	NOV. 5	S-11(S)	425 W	83.4	46.5	97.3	0.32	9.1	0.00210
17	NOV. 6	S-10	1400 W	62.6	176.9	352.6	0.02	—	0.00204
18	NOV. 17	N-6	1100 SW	129.8	96.6	217.3	no reading	no reading	no reading
19	NOV. 18	S-9(N)	1200 SW	46.9	175.2	332.8	0.10	9.8	0.00050
20	NOV. 19	N-4	—	—	—	—	—	—	—
21	NOV. 20	S-9(S)	300 E	52.1	41.6	80.3	0.07	16.0	0.00083

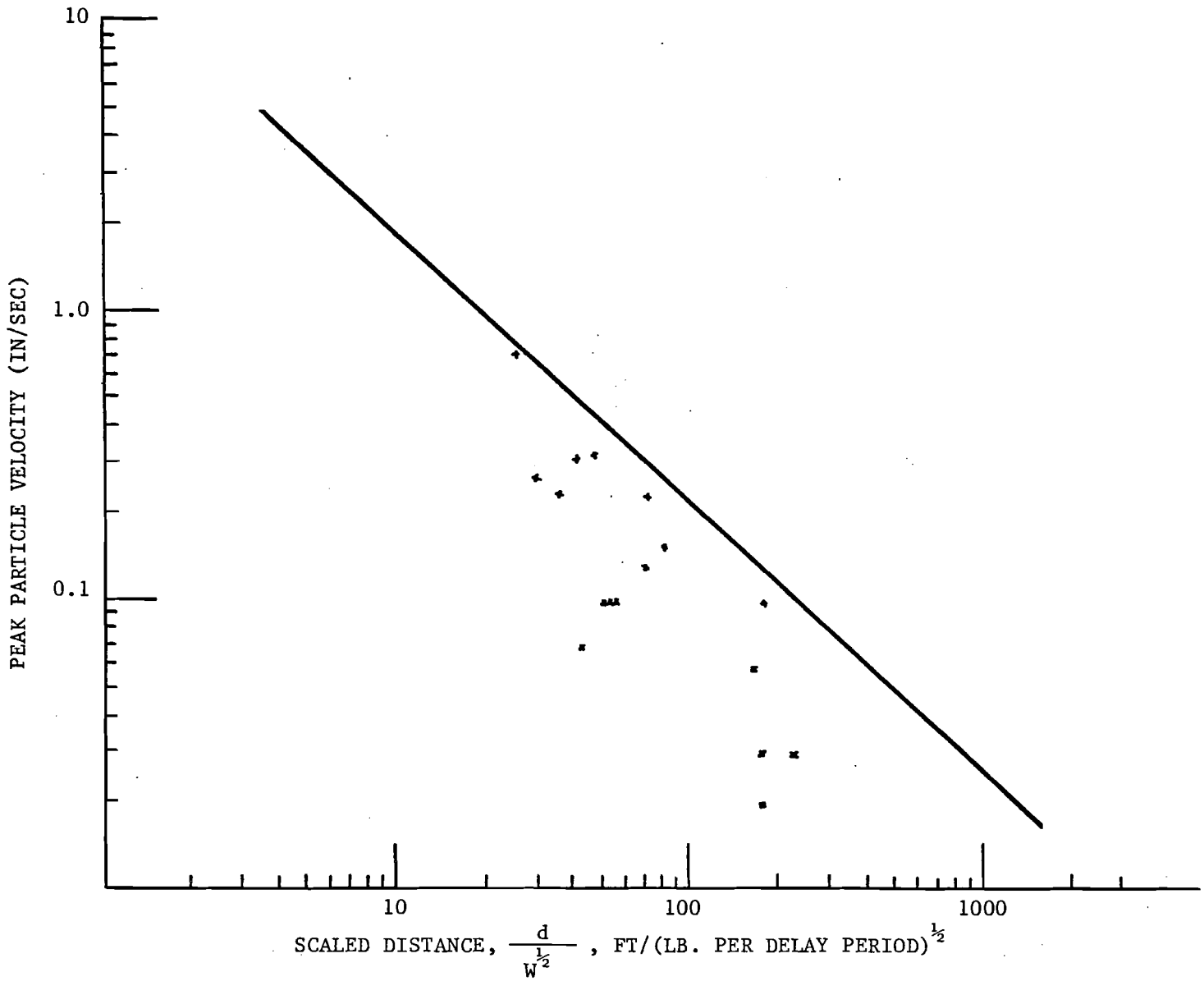


FIG. 7-3: PLOT OF PEAK PARTICLE VELOCITY VS. SCALED DISTANCE FOR BLASTS AT THE BEULAH TEST SITE

7.3 DISCUSSION OF RESULTS

The data in table 7-1 clearly shows that only modest peak particle velocity was experienced at the test site. Even when the seismograph was close to the shot the particle velocity levels were not of concern.

To achieve a peak particle velocity of 2.0 ins/sec, that was long considered the threshold of damage, a scaled distance of 9.0 ft/(lb per delay)$^{1/2}$ is required. For a peak particle velocity of 1.0 ins/sec (the allowable limit in operating coal mines) a scaled distance of 19 ft/(lb per delay)$^{1/2}$ is necessary according to the graph.

For this type of work it is recommended that a peak particle velocity of 0.5 ins/sec be the design limit. Then a minimum scaled distance of 42 ft/(lb per delay)$^{1/2}$ must be maintained. This vibration limit would allow for more recent findings concerning vibration and damage.

Reducing the vibration to this level will eliminate most citizen complaints about the blasting. It is a level at which no damage will result. One can eliminate all citizen concern by reducing the vibration to 0.1 ins/sec which leads to a minimum scaled distance of 240 ft/(lbs per delay)$^{1/2}$.

At the test site the closest house was 2,500 feet away. To insure that no vibration exceeded 0.5 ins/sec would require that no more than 3,543 pounds of explosive per delay be detonated. This is far in excess of the weight per delay required. To maintain no more than 0.1 ins/sec at the nearest house requires that the maximum weight per delay be 108.5 pounds.

The largest weight detonated per delay during the test blasts was 257 pounds. For the nearest house the scaled distance is then 156 ft/(lbs per delay)$^{1/2}$. From Fig. 7-3 no more than 0.15 ins/sec peak particle velocity would have been expected at this residence.

It was thought that the scaled distance for these charges should be scaled to the cube root of the weight. However, when the data was plotted using cube root scaling it led to weights per delay period that were far too optimistic. For example, for a PPV of 0.5 ins/sec a scaled distance of 90 ft/(lbs/delay)$^{1/3}$ was found. For this scaled distance a charge weight per delay of 21,400 pounds could be shot. Experience in predicting vibration levels leads to the conclusion that this is much too great and would have led to vibration levels over 1.0 ins/sec. Therefore, it was concluded that square root scaling was the better approach. This may result from multiple decks behaving like a continuous column charge.

Review of table 7-1 and Fig. 7-3 leads to the conclusion that AML reclamation by blasting can be achieved quite close to

buildings without problem. Experience at Beulah suggests that blasting within 400 feet is possible if the weight per delay does not exceed 90 pounds. This is certainly feasible if the hole diameter does not exceed 6 inches. At 90 pounds vibration should not exceed 0.5 ins/sec. The predicted level here is for a geology consisting of overconsolidated clays and may vary for other geology.

When the distance to the nearest building is no less than 1000 feet then up to 560 pounds per delay could be detonated. This is quite adequate for most purposes. It is therefore concluded that blasting is not a problem for distances of 1,000 feet or more and that distances as little as 400 feet can be achieved with careful design and field operation.

Airblast results were an order of magnitude below that which can cause damage. It is concluded that these favorable results were largely due to careful blast design and field practice.

The blasts were designed to heave the surface but to avoid bursting of the gases through the surface. Each hole was adequately stemmed. Drill cuttings were used for stemming and +1/4 - 3/4-inch stone was also tried. Both performed well with little or no advantage seen to using the gravel.

Of prime importance was the time taken to bury all the surface delays, detonating cord and blasting cap. These elements are noise creating but by burying each under drill cuttings most of this noise was negated. Surface delays were selected with airblast in mind. For this reason NONEL noiseless trunkline delays were used. The NONEL tube, detonating at about 6000 ft/second is very quiet and substantially minimizes the noise relative to that of detonating cord. It is also a safe product being less sensitive to stray currents and thunderstorms then electric hookups.

Therefore, from both a ground vibration and airblast perspective blasting to within 400 feet of structures will often be possible provided flyrock is also adequately controlled. Thus, blasting may be more generally applicable to AML reclamation than is often thought.

8. RESULTS FROM TEST BLASTING PROGRAM

8.1. INTRODUCTION

The purpose of this chapter is to compile the results of the test blast program and provide an overall assessment of the technical feasibility of blasting as an AML reclamation method. General results from each test blast were described in Chapter 4. In the following section a detailed technical analysis of blast data is carried out.

In addition results from measurement of pre-blast void size and post-blast profiles are presented, and an attempt is made to correlate these and make some observations regarding material swell during blasting.

In the final section of this chapter some general conclusions are made regarding the effectiveness of blasting for the collapse of underground development. It draws on results presented in Chapters 4, 6 and 7, and discusses the ways in which experimentation during the field testing program contributed to the optimization of blast design.

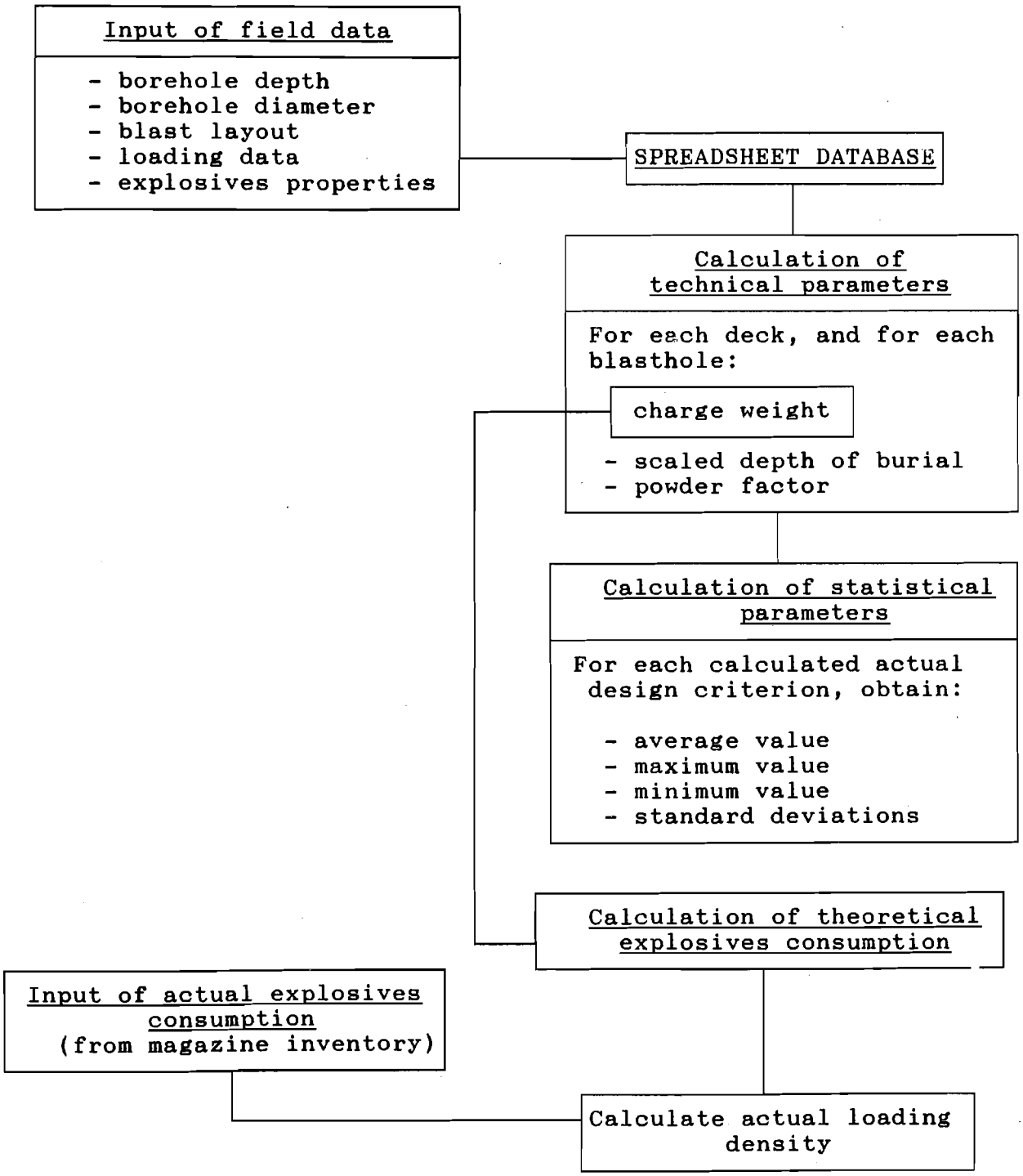
8.2. TECHNICAL ANALYSIS OF BLAST DATA

A schematic for the technical analysis of actual blast data is presented in Table 8-1. Due to the large amounts of data requiring analysis and the very repetitive nature of calculations, the decision to use a spreadsheet was an obvious one.

A spreadsheet for technical analysis of each of the 20 test blasts carried out on underground openings is contained in Appendix D. Blast #20 (N-4) was not included in the technical analysis as it was employed only to fill a sinkhole. An example of the spreadsheet is presented in Table 8-2, which will be used to illustrate the following description of the data and calculations contained therein. Two blasts, #2 (N-9) and #10 (S-1(N)) contained blastholes with 5 explosive decks. In these cases the spreadsheet contained extra columns for Deck Number 5.

In some cases a blast was employed primarily to collapse an opening, but was also used for an additional purpose. When this occurs the spreadsheet includes the blasthole depths, and calculates explosives consumption for the secondary blast. It does not calculate the other technical parameters. This is the case for blast #7 (N-7) where some of the holes were located around a sinkhole, and blast #17 (S-10) where part of the blast was used to collapse a short cross-cut that was considered to be an imminent hazard.

TABLE 8-1 : SCHEMATIC FOR TECHNICAL ANALYSIS OF BLASTS



AML PROJECT - BLAST SUMMARY : TECHNICAL DATA

EXPLOSIVE TYPE	ANFO	IREO	EMER
EXPL. DENSITY	0.85	1.00	0.85
WT. STRENGTH	1.00	0.96	0.95

BLAST # 5 LOCATION N-19 HOLE DIAM 6 ROOM WIDTH 22 FT SPACING 13 FT ROW SEPARATION 8 FT

		DECK			NUMBER 1						DECK			NUMBER 2				
#	DEPTH	DEL #	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SDOB	SDOB	P.F.	DEL #	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SDOB	P.F.
1	49.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
2	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
3	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
4	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
5	48.0	7	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.37	6	18.5	22.5	ANFO	4.0	41.7	2.45	0.75
6	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
7	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
8	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
9	48.0	7	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.37	6	18.5	22.5	ANFO	4.0	41.7	2.45	0.75
10	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
11	54.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.45	0.39	6	20.0	24.5	ANFO	4.5	46.9	2.43	0.81
12	59.0	7	9.5	15.5	ANFO	6.0	62.6	3.15	2.27	0.55	6	21.5	27.5	ANFO	6.0	62.6	2.39	0.95
13	47.5	7	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.37	6	18.5	22.5	ANFO	4.0	41.7	2.45	0.75
14	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
15	50.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
16	50.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
17	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
18	51.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.45	0.39	6	20.0	24.0	ANFO	4.0	41.7	2.45	0.75
19	49.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
20	49.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
21	49.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
22	47.5	7	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.37	6	18.5	22.5	ANFO	4.0	41.7	2.45	0.75
23	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
24	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
25	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
26	48.5	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
27	49.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.5	ANFO	4.5	46.9	2.43	0.81
28	47.0	7	9.5	12.5	ANFO	3.0	31.3	3.50	2.06	0.34	6	17.5	21.5	ANFO	4.0	41.7	2.45	0.75
29	50.0	7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.41	6	19.0	23.0	ANFO	4.0	41.7	2.45	0.75
30	48.0	7	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.37	6	18.5	22.5	ANFO	4.0	41.7	2.45	0.75
AV.	49.37	30				3.95	41.2	3.33	2.19	0.41	30				4.32	45.0	2.44	0.78
MIN	47.0					3.0	31.3	3.15	2.06	0.34					4.0	41.7	2.39	0.75
MAX	59.0					6.0	62.6	3.50	2.45	0.55					6.0	62.6	2.45	0.95
STD	2.21					0.46	4.81	0.05	0.09	0.03					0.40	4.22	0.01	0.04

EXPLOSIVES CONSUMPTION :	ANFO	EMER	IREO
THEOR.	3869	0	1509
ACTUAL	3000	0	1800
	0.78		1.19
	0.66		1.19

BURDEN 10.3 FT WT/UNIT LENGTH 12.27 PFAREA 143.0

	DECK			NUMBER 3						DECK			NUMBER 4				TOTAL		
DEL #	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SDOB	P.F.	DEL #	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SDOB	P.F.	LENGTH FT	WEIGHT LB	P.F.
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.05	0.94	16.0	174.2	0.66
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	29.0	33.0	ANFO	4.0	41.7	2.31	0.79	4	39.0	43.0	IREO	4.0	49.1	1.91	0.99	15.5	169.0	0.66
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	29.0	33.0	ANFO	4.0	41.7	2.31	0.79	4	39.0	43.0	IREO	4.0	49.1	1.91	0.99	15.5	169.0	0.66
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	31.0	36.0	ANFO	5.0	52.1	2.41	0.86	4	42.5	47.5	IREO	5.0	61.3	2.28	0.97	18.5	202.1	0.71
5	34.0	40.0	ANFO	6.0	62.6	2.39	0.95	4	46.5	52.5	IREO	6.0	73.6	2.27	1.07	24.0	261.3	0.84
5	29.0	33.0	ANFO	4.0	41.7	2.31	0.79	4	39.0	43.0	IREO	4.0	49.1	1.78	1.05	15.5	169.0	0.67
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.19	0.89	16.0	174.2	0.66
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.19	0.89	16.0	174.2	0.66
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	30.5	34.5	ANFO	4.0	41.7	2.45	0.75	4	41.0	45.0	IREO	4.0	49.1	2.19	0.89	16.0	174.2	0.64
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.05	0.94	16.0	174.2	0.66
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.05	0.94	16.0	174.2	0.66
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.05	0.94	16.0	174.2	0.66
5	29.0	33.0	ANFO	4.0	41.7	2.31	0.79	4	39.0	43.0	IREO	4.0	49.1	1.78	1.05	15.5	169.0	0.67
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.78	1.05	16.5	179.4	0.70
5	30.0	34.0	ANFO	4.0	41.7	2.31	0.79	4	40.0	44.0	IREO	4.0	49.1	1.91	0.99	16.5	179.4	0.69
5	28.0	32.0	ANFO	4.0	41.7	2.31	0.79	4	38.0	42.0	IREO	4.0	49.1	1.91	0.99	15.0	163.8	0.66
5	29.5	33.5	ANFO	4.0	41.7	2.45	0.75	4	40.0	44.0	IREO	4.0	49.1	2.19	0.89	16.0	174.2	0.66
5	29.0	33.0	ANFO	4.0	41.7	2.31	0.79	4	39.0	43.0	IREO	4.0	49.1	1.91	0.99	15.5	169.0	0.66
30				4.10	42.8	2.35	0.79	30				4.10	50.3	1.95	0.99	16.5	179.3	0.68
				4.0	41.7	2.31	0.75					4.0	49.1	1.78	0.89	15.0	163.8	0.64
				6.0	62.6	2.45	0.95					6.0	73.6	2.28	1.07	24.0	261.3	0.84
				0.40	4.20	0.07	0.04					0.40	4.94	0.16	0.05	1.55	16.84	0.03

TABLE 8-2: EXAMPLE OF SPREADSHEET USED FOR CALCULATION OF TECHNICAL PARAMETERS FROM ACTUAL BLAST DATA

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8.2.1. Data Input

The following basic data for each blast is input:

- blast number and location
- blasthole diameter
- width of opening (estimated from line-up drilling)
- spacing of holes along row
- row separation (zero for single-row blasting)
- densities and weight strengths for explosives used
- actual explosives consumption (inventory sheets)

For each blasthole the following data input is required:

- identifying blasthole number
- depth to void

Data input for each deck in each blasthole:

- down-the-hole delay number used for deck initiation
- explosive type used (IREG = Iregel, ENER = Energel)
- down-hole distances to top and bottom of each deck

Much of the above information is obtained from field sketches, blast summary sheets and field notes as described in Chapter 5 of this report. Unless otherwise specified in blast summary sheets or field notes, the standard loading distances as set-up for a given blasthole depth were used in this analysis.

8.2.2. Calculation of Technical Parameters

There follows a brief description of the technical parameters calculated by the technical data analysis spreadsheet.

8.2.2.1. Burden between Blastholes

For single row blasts this does not apply, as the burden is equal to the hole spacing. For double-row blasts this is obtained by the following expression:

$$\text{Burden} = \frac{\text{Row separation}}{\cos \Phi}$$

where $\Phi = \text{TAN}^{-1}$ [Hole spacing / (2 x row separation)]

8.2.2.2. Weight per Unit Length of Charge Column

This is the weight, in pounds, of each charged vertical foot of blasthole, for a given blasthole diameter. It assumes that the explosive has unit specific gravity. The specific gravity of individual explosives is factored in later.

$$\text{Wt./unit length} = \frac{[\text{Hole diameter (inches)}]^2 \times \pi \times 62.4}{2 \times 12}$$

This formula is entered in the top of the spreadsheet, and used in each calculation of charge weight.

8.2.2.3. Area of Influence of Blastholes

As explained earlier, blast design in this type of work is carried out according to crater theory. However, it is felt that the calculation of powder factors will be beneficial to workers who are more familiar with this concept. It is very important to understand, though, that successful AML blasting of the type described in this report should be based primarily on crater theory. Use of powder factor alone will not be an adequate design approach.

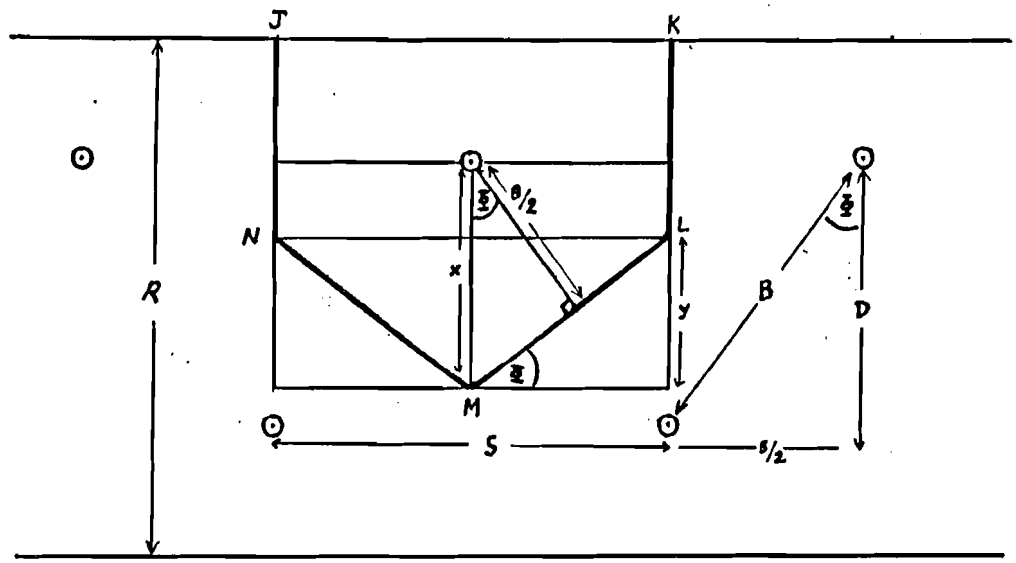
In order to calculate powder factor it is necessary to estimate the volume of ground affected by each explosive deck, and for each blasthole. For this model it is assumed that each blasthole affects a portion of the volume of ground which lies immediately above the open underground workings. The problem is thus resolved by the calculation of a "polygonal area of influence" for each borehole, as indicated in the diagram on Fig. 8-1. During powder factor calculations this area is multiplied by the appropriate depth of overburden over which each blasthole deck has influence.

In reality, the volume influenced by the blast will consist of a series of overlapping crater volumes, and the concept of a block of overburden with vertical walls for powder factor is a crude one. This is illustrated in Fig. 8-2, which also shows how the distances employed in volume calculations for each deck are obtained.

This calculated area is entered in the spreadsheet with the title "PFAREA". It should be noted that for blasts which took place in more than one development type, this factor is calculated for the dominant type only. In other words, if a blast took place on a room and part of the adjacent panel entry, the dimensions of the room, not the entry, are used to calculate the area of influence of a blasthole deck.

FIG. 8-1: CALCULATION OF POLYGONAL AREA OF INFLUENCE FOR A BLASTHOLE FOR POWDER FACTOR DETERMINATION

A. TWO LINES OF BLASTHOLES

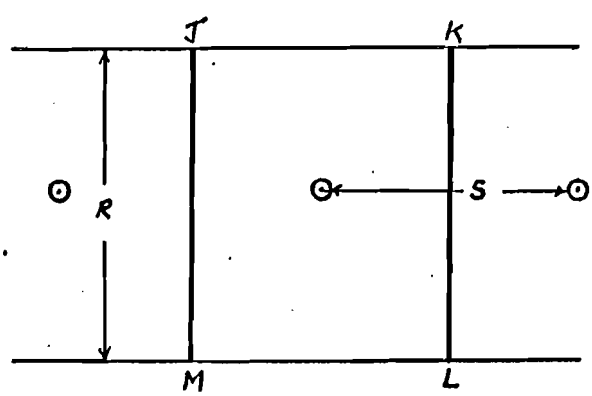


S = hole spacing on row
D = distance between rows
R = room width
⊙ = blasthole location

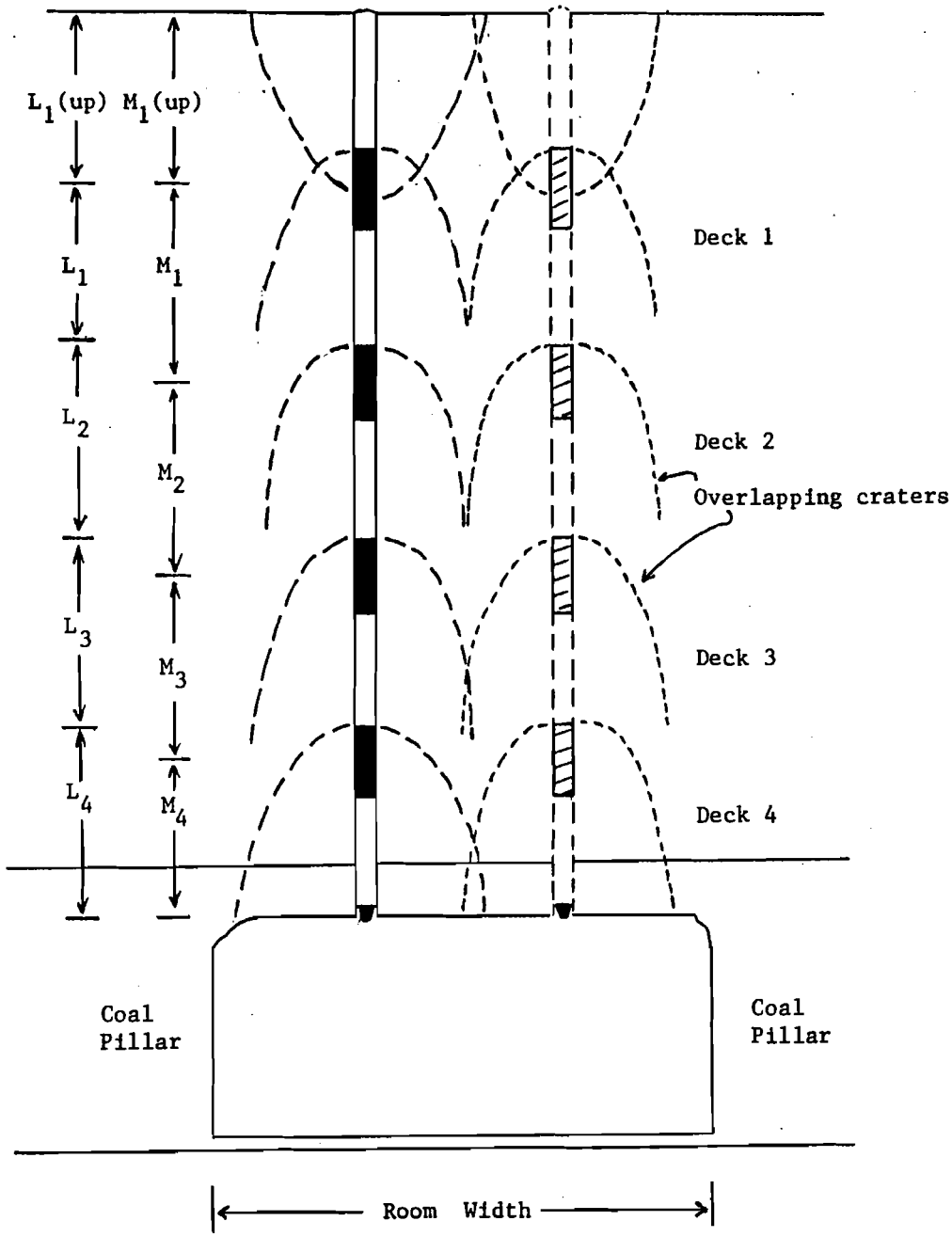
Φ = ATAN [S/2D]
B = D / COS Φ
x = B / 2 COS Φ
y = S TAN Φ / 2

AREA JKLMN = S [(R-D)/2 + (x - y)] + S/2. y = S [(R-D)/2 + x - y/2]

B. SINGLE LINE OF BLASTHOLES



AREA JKLM = R x S



$$\text{Rel. P.F.}_n = \frac{\text{Charge Wt.}_n \times \text{Wt. Strength} \times 27}{\text{Polygon Area} \times L_n}$$

$$\text{SDOB}_n = \frac{M_n}{\text{Charge Wt.}^{1/3}}$$

FIG. 8-2: ILLUSTRATING COLUMN DEPTHS EMPLOYED FOR CALCULATION OF RELATIVE POWDER FACTORS AND SCALED DEPTHS OF BURIAL

8.2.2.4. Weight of Explosive Column

For each explosive deck in a blasthole the length of charge column is obtained by subtracting the value in the "TOP" column from that in the "BOTTOM" distance column. The charge weight is obtained according to the following:

Charge wt. = Wt./unit length x charge length x explosive S.G.

The spreadsheet determines the appropriate explosive specific gravity by checking the contents of the "EXPL. TYPE" column, and using the value from the small table located at the top of the spreadsheet.

8.2.2.5. Scaled Depth of Burial (SDOB)

The scaled depth of burial for an explosive deck is the distance from the center of the explosive charge column to the "free-surface" created by the deck below, scaled by the cube-root of the weight of the explosive deck. This is shown below:

$$\text{SDOB} = \frac{\text{Distance from charge center to free face}}{(\text{Charge weight})^{1/3}}$$

The way in which these distances are obtained is shown in Fig. 8-2

For the top deck (Deck Number 1) there are two SDOB values, one down to the deck below, and another one up to surface. The first of the SDOB values in the spreadsheet (see Table 8-2) refers to the scaled depth of burial of deck number 1 below surface. Since collar stemming distances were invariably greater than between-deck separations, this first SDOB is usually greater than those for decks 2,3,4 and 5.

8.2.2.6. Powder Factor

Powder factor is calculated as the weight of explosive in a deck per unit volume of overburden over which it may be considered to act. The appropriate volume is calculated by multiplying the polygonal area of influence, described earlier, by the distance from the top of the explosive column to the free-face (see Fig. 8-2).

The powder factor must be weighted by the relative weight strength of the explosive in order to calculate a "relative powder factor". On a per weight basis, for example, a slurry explosive may have only 85% of the explosive energy of ANFO. This is due to the water used to concentrate the Ammonium Nitrate into a liquor (usually 14-16% by weight).

In a given charge deck length there will be a greater weight of slurry than ANFO, due to its higher density. The explosive energy, however, must be factored by the lower weight strength of the slurry. If, for example, the slurry has a S.G. of 1.2, then the explosive energy per unit volume may well be greater than that of ANFO. However, if the slurry has a weight strength of only 0.85, there will be less available explosive energy per unit weight than if ANFO had been used.

Since powder factor is a weight-based measure of explosives consumption it is necessary to relate the powder factors of individual explosives to a base. This is usually done by quoting the unit weight of ANFO (per cubic yard or per ton) that is necessary to generate the same energy output on detonation. This is designated the "relative powder factor".

The explosive charge in the top deck affects the material both above and below it. Consequently the powder factors obtained for the top deck are lower than for the others. The relative powder factor is calculated as follows:

$$\text{Rel. P.F. (lb/cu.yd)} = \frac{\text{Charge wt. x Wt. strength x 27}}{\text{Area of influence x length of influence}}$$

Powder factor is not calculated for blastholes whose area of influence is different from those in the dominant development type for the blast.

For each blasthole the total charge length is calculated, together with the appropriate explosive weight, and entered into the last columns of the technical analysis spreadsheet (see Table 8-2). The powder factor calculated here is not a relative powder factor. It is obtained by dividing the total weight of explosive in the blasthole by the total volume affected:

$$\text{Total P.F. (lb/cu.yd)} = \frac{\text{Total Charge Wt. x 27}}{\text{Area of influence x hole depth}}$$

8.2.2.7. Statistical Analysis of Technical Parameters

For each blast the average values for borehole depth, and the length and weight of charge for each deck in each borehole are obtained by the spreadsheet. Average values for scaled depth of burial and powder factor are also calculated. For each parameter, spreadsheet functions were also employed to indicate the maximum and minimum value in each column, and the standard deviation of the values.

The calculated technical parameters are generally a function of the borehole depth, as the loading instructions were standardized during the course of the test blast program. If borehole depths were consistent, then a low standard deviation is expected for these. If blasthole depths were very variable, due perhaps to some prior caving in the area which had not yet shown up as a surface feature, then this will be reflected in the "MIN", "MAX" and "STD" rows of the spreadsheet (see Table 8-2).

In some cases, where the SDOB was considered to be not typical, or not relevant, it was not calculated, and therefore omitted from the statistical analysis. This is the case, for example, of blastholes which did not intersect a void, such as those used around sinkholes, or closure holes at either end of a blast.

Variations in development type are not reflected in the statistical analysis of powder factors as these are only calculated in the dominant development type for that blast.

As stated earlier, actual variations in loading practice from the standard loading instructions are recorded at the data input stage. Therefore, factors such as the presence of rock layers, or the presence of thicker roof coal, will be reflected in the statistical analysis of technical parameters.

8.2.2.8. Theoretical and Actual Loading Density

It was explained above how the specific gravity of each explosive type was used to calculate the charge weight for each deck and each blasthole. It follows that, if these weights are totalled for the blast as a whole, there should be a reasonable correlation between these and the actual field consumption of explosives as indicated by magazine inventory sheets.

The theoretical explosive densities of slurry products Iregel and Energel are located in the small table at the top of the technical analysis spreadsheet (see Table 8-2). Theoretical and actual consumption for each explosive type is shown in the small table at the bottom of the spreadsheet entitled "Explosives consumption". These are recorded as a weight in pounds. A ratio may then be calculated of actual to theoretical weights; this is the third figure in each column of this part of the spreadsheet (see Table 8-2).

In initial analysis work carried out, the values used for theoretical explosive density were the manufacturers' claimed densities for the bagged product. The ratios of actual to theoretical consumption for slurry explosives were found to be significantly lower than unity. There are are four possible operational reasons why this situation could arise:

- unusual loading practices in the field
- inaccurate measurement of deck column heights in the field
- actual blasthole diameter less than planned
- loading densities were lower than expected

Very seldom were mistakes made with field loading practice. This was, after all, a research oriented project, and most of the labor was carried out by engineering personnel well aware of the importance of interpreting results in the light of design criteria.

Some inaccuracy inevitably occurs when measuring down-hole distances to charge and stemming interfaces. These tended to be systematic, however, in their form, with a tendency to put a few inches too much, rather than too little, in the explosive column. As such, one would expect the ratio of actual to theoretical explosives consumption to be greater than one.

There was close agreement in many cases between theoretical and actual ANFO weights. This suggested that there was not a significant problem with blasthole diameter.

There definitely appeared to be discrepancy between these values for the bagged slurry products. It had become apparent during field operations, especially for the Energel product, that it was taking less weight of explosive than would have been expected to fill a given charge column height.

It was observed, for example, that one and a half bags of the Energel product, representing 45 lbs of explosive, was consistently occupying a 4 feet 4 inch high deck in a six inch hole. As shown earlier, the weight of explosive in a deck can be calculated as follows:

$$\text{Wt.(lbs)} = \pi \times \frac{[\text{Hole diam.}/2]^2}{12} \times \text{column ht.} \times 62.4 \times \text{S.G.}$$

In the above example, the apparent S.G. of the loaded product is given by:

$$\text{S.G.} = \frac{45}{12.25 \times 4.33}$$

$$= 0.848$$

The practice of cutting open slurry bags in field loading operations (see Chapter 5) may result in a lower density in the hole than that achieved during manufacture by compressing the product into plastic lined tubular bags. This is especially the case for the Energel product, which was quite stiff and had to be cut into chunks for loading purposes.

Theoretical explosives densities for the slurry explosives were therefore reduced by 10-15% from manufacturers' claimed densities for the bagged product. These are the values assumed for all of the results of technical analysis work as presented in Appendix D of this report. The final figure in each column of the small table entitled "Explosives Consumption" on the spreadsheet is the apparent actual loading density indicated from the analysis. It is obtained according to the following expression:

$$\text{Apparent loading density (SG)} = \frac{\text{Actual consumption}}{\text{Theoretical consumption}} \times \text{theoretical loading density (SG)}$$

Data summarizing the relationship between actual and theoretical explosives consumption for the test blast program is contained in Table 8-3. It can be seen that the indicated, or apparent specific gravity for the first batch of ANFO used in testwork would appear to be a little lower than expected, by about 6%. There is close agreement for the second batch of bagged ANFO, however.

The actual loading density for Iregel slurry explosive is calculated in Table 8-3 to be 0.97 g/cc. The value of 1.00 g/cc assumed for the analysis of technical blast data would appear to be reasonable, therefore.

In calculating the apparent loading density for Energel it was decided to disregard suspect values from blast #17. This blast was a complex one, and it is likely that more slurry was actually loaded than indicated in the analysis spreadsheet. The indicated loading density for Energel is thus 0.87 g/cc, as shown in Table 8-3. The value of 0.85 g/cc initially assumed, which as stated earlier was a value indicated by actual field observations, was thus a reasonable one.

The significance of these observations regarding explosive loading densities, and how they affect blast design, is discussed later in this Chapter.

8.2.3. Technical Data Summary

A general summary of the field test blasts was included in Chapter 4 as Table 4-5. Technical data such as charge weight,

TABLE 8-3 : COMPARISON OF ACTUAL AND THEORETICAL EXPLOSIVES CONSUMPTION FOR FIELD BLAST PROGRAM

	EXPLOSIVE S.G. ANFO#1	ANFO#2	IREGEL	ENERGEL
CLAIMED	0.85	0.85	1.15	1.10
USED	0.85	0.85	1.00	0.85

BLAST #	LOCATION	ANFO#1 THEORET.	ANFO#1 ACTUAL	ANFO#1 RATIO	ANFO#1 APPARENT S.G.	ANFO#2 THEORET.	ANFO#2 ACTUAL	ANFO#2 RATIO	ANFO#2 APPARENT S.G.	IREGEL THEORET.	IREGEL ACTUAL	IREGEL RATIO	IREGEL APPARENT S.G.	ENERGEL THEORET.	ENERGEL ACTUAL	ENERGEL RATIO	ENERGEL APPARENT S.G.
1	N-14	1481	1325	0.89	0.76												
2	N-9	2086	1800	0.86	0.73												
3	N-11	3040	2900	0.95	0.81					1282	930	0.73	0.73				
4	N-1,N-2	5700	5725	1.00	0.85					2582	2640	1.02	1.02				
5	N-13	3869	3000	0.78	0.66					1509	1800	1.19	1.19				
6	N-8(S)	1585	1450	0.91	0.78					736	690	0.94	0.94				
7	N-7	1710	1550	0.91	0.77					528	480	0.91	0.91				
8	N-12	2711	2400	0.89	0.75					1172	1200	1.02	1.02				
9	NC-7	1126	1150	1.02	0.87					589	630	1.07	1.07				
10	S-1(N)	4322	4250	0.98	0.84					1926	1650	0.86	0.86				
11	S-12	3682	3450	0.94	0.80												
12	N-8(N)	3608	3200	0.89	0.75												
13	N-10	5099	5200	1.02	0.87												
14	S-11(N)	3420	3350	0.98	0.83												
15	S-1(S)					1481	1450	0.98	0.83					1648	1770	1.07	0.91
16	S-11(S)					1319	1400	1.06	0.90					626	720	1.15	0.98
17	S-10,SC-7					2164	2200	1.02	0.86					* 991	* 1200	* 1.21	* 1.03
18	N-6					2873	2950	1.03	0.87					1363	1320	0.97	0.82
19	S-9(N)					1345	1300	0.97	0.82					1663	1620	0.97	0.83
20	N-4																
21	S-9(S)					720	750	1.04	0.89					880	930	1.06	0.90
TOTALS/AVERAGES		43439	40750	0.94	0.797	9902	10050	1.01	0.863	10324	10020	0.97	0.971	6180	6360	1.03	0.875

* Values for Blast #17 suspect.
Discounted for calculation of average apparent loading density

scaled depth of burial and powder factor are very dependent on blast design. Since blast design was essentially standardized, in conjunction with the use of loading boards, these parameters will essentially be a function of blasthole depth.

Data of the type shown in Table 8-2, and more extensively in Appendix D, relates to actual blast conditions. Local variations such as overburden depth and roof coal thickness were designed for in actual loading practice. To a certain extent average values, and the associated standard deviations around these, will be of some use in indicating the degree of variability of site factors.

In other cases, however, these will be less meaningful. A good example is where the overburden depth varies around the critical depth at which a change in the number of required explosive decks occurs. The case of 40-41 feet of overburden, for example, where the number of decks increases from 3 to 4, is illustrated in table 8-4.

TABLE 8-4 : TECHNICAL PARAMETERS ASSOCIATED WITH 40 AND 41 FOOT BLASTHOLES.

Hole depth (ft)	# decks	DECK 1				DECK 2			
		Col.Ht (ft)	Col.Wt (lbs)	SDOB	P.F.	Col.Ht (ft.)	Col.Wt. (lbs)	SDOB	P.F.
40	3	4	41.7	2.45	0.34	4	41.7	2.31	0.68
41	4	3	31.5	1.75	0.31	3	31.3	2.22	0.60

It was felt, therefore, that a summary of the technical data analysis for the field test blasting program should take the form of tables showing the variation of technical parameters with blasthole depth under a number of typical loading conditions.

In each case a room width of 22 feet is assumed, and two rows of six inch blastholes spaced at 15 feet intervals on rows 8 feet apart. Five different loading options are considered:

- use of ANFO only
- use of Iregel slurry in bottom deck
- use of Energel slurry in bottom deck
- use of Iregel slurry in bottom 2 decks
- use of Energel slurry in bottom 2 decks

An example of one such analysis, the case where the bottom

deck of a blast was loaded with Iregel explosive, is presented in Table 8-5. The complete summary analysis is contained in Appendix E.

Each one of these tables presents a list of the charge weight, SDOB and relative powder factors for each deck in blastholes of depths ranging from 35 to 66 feet. In other words, this is an analysis of the technical characteristics of the standard loading procedures.

Explosives specific gravities of 0.85, 1.0 and 0.85 were assumed as loading densities for ANFO, Iregel and Energel products respectively, as explained in the previous section of this report. The values in the "TOTALS" columns can be considered as typical charge weight and powder factors for the given blasthole depth. The charge weights are those used for estimation of explosives requirements in the Blast Cost Model described later in Chapter 9.

Fig. 8-3 shows the variation in SDOB for the "standard" blast design in the upper two decks of blastholes, for the overburden depth range 35 to 66 feet. There are, as explained previously, two SDOB's for the upper deck, the larger being the depth of burial of the charge center below surface. Breaks at 41 and 60 feet correspond to the change from 3 to 4, and from 4 to 5 decks, respectively.

Between 40 and 50 feet the increase in SDOB shows as a series of breaks, rather than a gradual increase. This is because loading distances were rounded up or down to the nearest six inches for practical field implementation. Fig. 8-3 is pertinent to all explosive loading carried out in the field blasting program, as ANFO was always used in at least the top two decks.

The variation of SDOB with depth for the lower decks, when ANFO explosive alone is used, is shown in Fig. 8-4. Here the significant changes for this parameter occur at 39, 41, 46, 50 and 60 feet overburden.

Powder factor variation with depth for the upper two decks is shown in Fig. 8-5. The value of this parameter for deck #1 is approximately half of that for the other decks, for reasons explained previously. The trend for these decks is generally ascending as overburden depth increases. The major exception occurs at 60 feet, due to the addition of a fifth deck.

For the lower decks (see Fig. 8-6) there is a drop-off in powder factor around 50 feet overburden depth, until the charge lengths in these columns are increased from 4, through 4.5 to 5 feet. These changes are illustrated in table 8-6, which includes the use of Iregel slurry explosive in the bottom blasthole deck.

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AHL PROJECT - BLAST LOADING DATA SUMMARY

EXPLOSIVE TYPE	ANFO	IREG	EMER
DENSITY	0.85	1.00	0.85
WT. STRENGTH	1.00	0.96	0.88
HOLE DIAMETER	6	IN	
WIDTH OF OPENING	22	FT	
HOLE SPACING	15	FT	
ROW SEPARATION	0	FT	
BURDEN	11.0	FT	
WT/UNIT LENGTH	12.27		
PPRACR	165.0	SQ. FT	

BOTTOM DECK : IREG

DEPTH FT	WT. ANFO	WT. SLURRY	TOP	BOT	EXPL TYPE	LENGTH FT	WT. LB	SCOB UP	SCOB DOWN	P.F.	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SCOB	P.F.
			0 E C K			N U M B E R				1	0 E C K			N U M B E R			2
35	62.6	49.1	9.5	12.5	ANFO	3.0	31.3	3.80	2.06	0.29	17.5	20.5	ANFO	3.0	31.3	2.06	0.64
36	67.0	49.1	9.5	12.5	ANFO	3.0	31.3	3.60	2.06	0.29	17.5	21.0	ANFO	3.5	36.5	2.34	0.63
37	70.2	49.1	9.5	13.0	ANFO	3.5	36.5	3.40	2.04	0.33	18.0	22.0	ANFO	4.0	41.7	2.31	0.60
38	70.2	49.1	9.5	13.0	ANFO	3.5	36.5	3.40	2.34	0.31	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
39	70.2	49.1	9.5	13.0	ANFO	3.5	36.5	3.40	2.34	0.31	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
40	83.4	49.1	9.5	13.5	ANFO	4.0	41.7	3.32	2.45	0.34	20.0	24.0	ANFO	4.0	41.7	2.31	0.60
41	93.0	36.0	9.5	12.5	ANFO	3.0	31.3	3.50	1.75	0.31	16.5	19.5	ANFO	3.0	31.3	2.22	0.60
42	93.0	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	1.75	0.31	16.5	19.5	ANFO	3.0	31.3	2.22	0.60
43	104.3	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	1.75	0.31	16.5	19.5	ANFO	3.0	31.3	2.22	0.60
44	104.3	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	1.75	0.31	16.5	19.5	ANFO	3.0	31.3	2.22	0.60
45	104.3	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	1.75	0.31	16.5	19.5	ANFO	3.0	31.3	2.54	0.54
46	104.3	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	2.06	0.29	17.5	20.5	ANFO	3.0	31.3	2.54	0.54
47	114.7	49.1	9.5	12.5	ANFO	3.0	31.3	3.50	2.06	0.29	17.5	21.5	ANFO	4.0	41.7	2.45	0.65
48	119.5	49.1	9.5	13.0	ANFO	3.5	36.5	3.40	2.19	0.32	18.3	22.5	ANFO	4.0	41.7	2.45	0.65
49	130.3	49.1	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.5	ANFO	4.5	46.9	2.49	0.70
50	135.1	49.1	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.45	0.65
51	135.1	49.1	9.5	13.5	ANFO	4.0	41.7	3.32	2.45	0.34	20.0	24.0	ANFO	4.0	41.7	2.45	0.65
52	135.1	89.2	9.5	13.5	ANFO	4.0	41.7	3.32	2.45	0.34	20.0	24.0	ANFO	4.0	41.7	2.45	0.65
53	130.3	61.3	9.5	13.5	ANFO	4.0	41.7	3.32	2.49	0.34	20.0	24.0	ANFO	4.0	41.7	2.45	0.68
54	140.8	61.3	9.5	13.5	ANFO	4.0	41.7	3.32	2.49	0.34	20.0	24.0	ANFO	4.5	46.9	2.49	0.70
55	151.2	61.3	9.5	14.0	ANFO	4.5	46.8	3.26	2.49	0.37	20.5	25.5	ANFO	5.0	52.1	2.41	0.74
56	156.4	67.8	9.5	14.5	ANFO	5.0	52.1	3.22	2.41	0.41	21.0	26.0	ANFO	5.0	52.1	2.41	0.74
57	166.0	87.8	9.5	14.5	ANFO	5.0	52.1	3.22	2.41	0.41	21.0	26.5	ANFO	5.5	57.4	2.40	0.78
58	172.1	73.6	9.5	15.0	ANFO	5.5	57.4	3.18	2.40	0.44	21.5	27.0	ANFO	5.5	57.4	2.40	0.78
59	187.7	73.6	9.5	15.5	ANFO	6.0	62.6	3.15	2.27	0.48	21.5	27.5	ANFO	6.0	62.6	2.35	0.82
60	177.3	61.3	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
61	177.3	73.6	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
62	187.7	73.6	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
63	198.1	73.6	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
64	208.6	73.6	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
65	213.4	79.7	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60
66	213.0	92.0	9.5	13.5	ANFO	4.0	41.7	3.32	2.16	0.36	19.0	23.0	ANFO	4.0	41.7	2.31	0.60

TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SCOB	P.F.	TOP	BOT	EXPL TYPE	LGTH FT	WT. LB	SCOB	P.F.	TOP	BOT	EXPL TYPE	LGTH FT	WEIGHT LB	SCOB	P.F.	LGTH FT	WEIGHT LB	P.F.
						3							4							5	T O T A L S		
25.5	29.5	IREG	4.0	49.1	2.05	0.81															10.0	111.6	0.52
27.0	31.0	IREG	4.0	49.1	1.91	0.86															10.5	116.9	0.53
28.0	32.0	IREG	4.0	49.1	1.91	0.86															11.5	127.3	0.56
29.0	33.0	IREG	4.0	49.1	1.91	0.86															11.5	127.3	0.55
29.0	33.0	IREG	4.0	49.1	2.19	0.77															11.5	127.3	0.53
30.0	34.0	IREG	4.0	49.1	2.19	0.77															12.0	132.5	0.54
25.0	28.0	ANFO	3.0	31.3	2.06	0.64	33.0	36.0	IRLO	3.0	36.0	1.99	0.72								12.0	130.7	0.52
25.0	28.0	ANFO	3.0	31.3	2.06	0.64	33.0	37.0	IREG	4.0	49.1	1.91	0.86								13.0	142.9	0.56
25.0	29.0	ANFO	4.0	41.7	2.02	0.76	34.0	38.0	IREG	4.0	49.1	1.91	0.86								14.0	153.3	0.58
25.0	29.0	ANFO	4.0	41.7	2.31	0.68	35.0	39.0	IREG	4.0	49.1	1.91	0.86								14.0	153.3	0.57
26.0	30.0	ANFO	4.0	41.7	2.31	0.68	36.0	40.0	IREG	4.0	49.1	1.91	0.86								14.0	153.3	0.56
27.0	31.0	ANFO	4.0	41.7	2.31	0.68	37.0	41.0	IREG	4.0	49.1	1.91	0.86								14.0	153.3	0.55
28.0	32.0	ANFO	4.0	41.7	2.31	0.68	38.0	42.0	IREG	4.0	49.1	1.91	0.86								15.0	163.8	0.57
29.0	33.0	ANFO	4.0	41.7	2.31	0.68	39.0	43.0	IREG	4.0	49.1	1.91	0.86								15.5	169.0	0.58
30.0	34.0	ANFO	4.0	41.7	2.31	0.68	40.0	44.0	IREG	4.0	49.1	1.91	0.86								16.5	179.4	0.60
29.5	33.5	ANFO	4.0	41.7	2.49	0.68	40.0	44.0	IREG	4.0	49.1	2.19	0.77								16.0	174.2	0.57
30.5	34.5	ANFO	4.0	41.7	2.49	0.68	41.0	45.0	IREG	4.0	49.1	2.19	0.77								16.0	174.2	0.56
30.5	34.5	ANFO	4.0	41.7	2.45	0.68	41.0	45.5	IREG	4.5	55.2	2.30	0.79								16.5	180.3	0.57
30.5	35.0	ANFO	4.0	46.9	2.49	0.70	41.5	46.5	IREG	5.0	61.3	2.29	0.84								17.5	191.7	0.59
31.0	35.0	ANFO	5.0	52.1	2.41	0.74	42.5	47.5	IRFO	5.0	61.3	2.22	0.84								18.5	202.1	0.61
32.0	37.0	ANFO	5.0	52.1	2.41	0.74	43.5	48.5	IRLO	5.0	61.3	2.20	0.84								19.5	212.5	0.63
32.5	37.5	ANFO	5.0	52.1	2.41	0.74	44.0	49.5	IREG	5.5	67.5	2.27	0.88								20.5	223.9	0.65
33.0	38.5	ANFO	5.5	57.4	2.40	0.78	45.0	50.5	IRLO	5.5	67.5	2.27	0.88								21.5	234.3	0.67
33.5	39.0	ANFO	5.5	57.4	2.40	0.78	45.5	51.5	IRFO	6.0	73.6	2.27	0.93								22.5	245.7	0.69
34.0	40.0	ANFO	6.0	62.6	2.39	0.82	46.5	52.5	IRFO	6.0	73.6	2.27	0.93								23.0	251.3	0.72
29.0	33.0	ANFO	4.0	41.7	2.31	0.68	39.0	44.0	ANFO	5.0	52.1	2.20	0.78	50.0	55.0	IREG	5.0	61.3	1.90	0.96	22.0	238.6	0.65
29.0	33.0	ANFO	4.0	41.7	2.31	0.68	39.0	44.0	ANFO	5.0	52.1	2.20	0.78	50.0	56.0	IREG	6.0	73.6	1.91	1.05	23.0	250.9	0.67
29.0	33.0	ANFO	4.0	41.7	2.31	0.68	39.0	45.0	ANFO	6.0	62.6	2.27	0.85	51.0	57.0	IREG	6.0	73.6	1.91	1.05	24.0	261.3	0.69
29.0	34.0	ANFO	5.0	52.1	2.28	0.78	40.0	46.0	ANFO	6.0	62.6	2.27	0.85	52.0	58.0	IREG	6.0	73.6	1.91	1.05	25.0	271.7	0.71
29.0	35.0	ANFO	6.0	62.6	2.27	0.85	41.0	47.0	ANFO	6.0	62.6	2.27	0.85	53.0	59.0	IREG	6.0	73.6	1.91	1.05	26.0	282.2	0.72
29.0	35.0	ANFO	6.0	62.6	2.27	0.85	41.0	47.5	ANFO	6.5	67.8	2.27	0.89	53.5	60.0	IREG	6.5	79.7	1.92	1.09	27.0	293.8	0.74
29.0	35.0	ANFO	6.0	62.6	2.27	0.85	41.0	47.5	ANFO	6.5	67.8	2.27	0.89	53.5	61.0	IREG	7.5	92.0	1.94	1.16	28.0	305.8	0.76

TABLE 8-5: EXAMPLE OF SPREADSHEET USED FOR CALCULATION OF TECHNICAL PARAMETERS FOR STANDARD LOADING DATA USING IREGEL SLURRY EXPLOSIVE IN BOTTOM DECK

FIG. 8-3 BLAST TECHNICAL DATA ANALYSIS

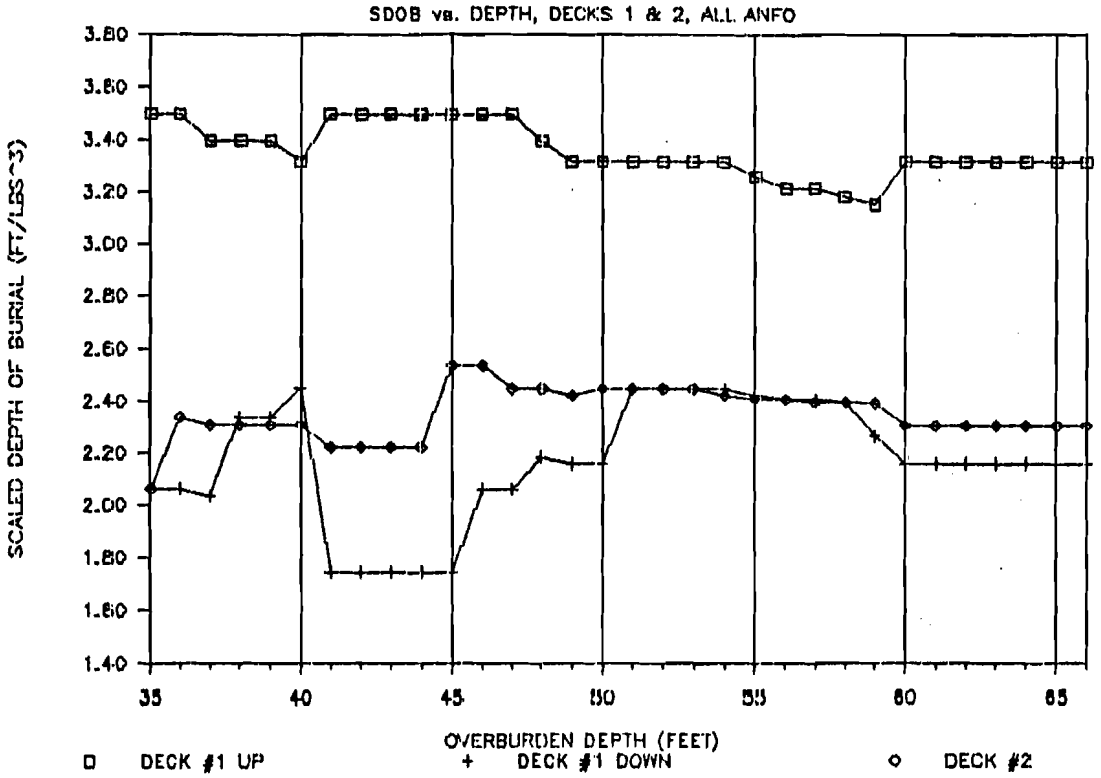


FIG. 8-4 BLAST TECHNICAL DATA ANALYSIS

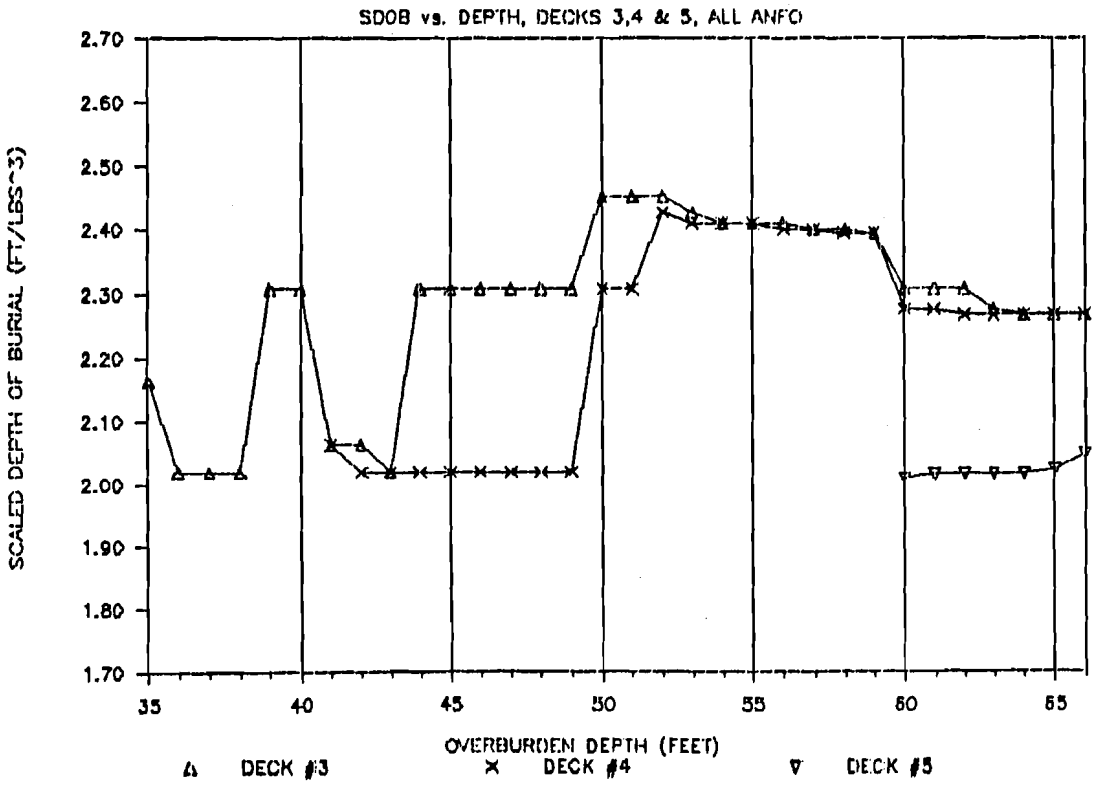


FIG. 8-5 BLAST TECHNICAL DATA ANALYSIS

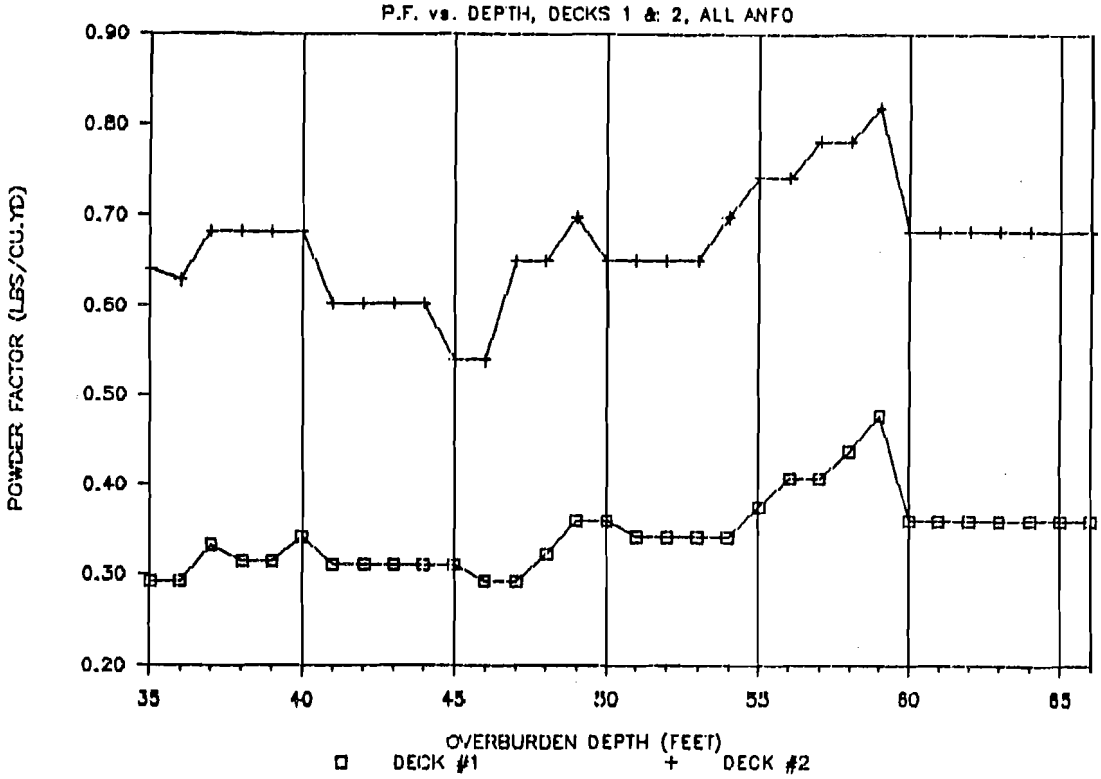


FIG. 8-6 BLAST TECHNICAL DATA ANALYSIS

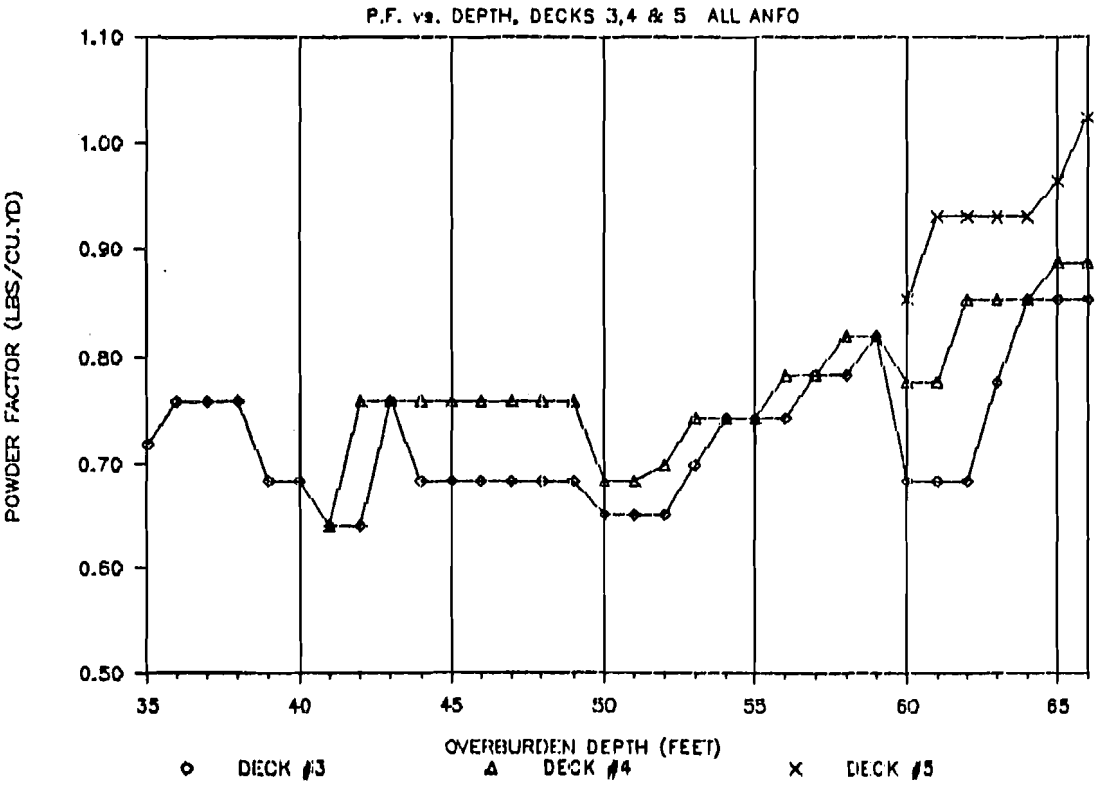


TABLE 8-6 : TECHNICAL PARAMETERS ASSOCIATED WITH BLASTHOLES OF DEPTHS 49 TO 54 FEET.

Hole depth (ft)	DECK 3				DECK 4			
	Col.Ht (ft)	Col.Wt (lbs)	SDOB	P.F.	Col.Ht (ft.)	Col.Wt. (lbs)	SDOB	P.F.
49	4	41.7	2.31	0.68	4	49.1	1.91	0.86
50	4	41.7	2.45	0.65	4	49.1	2.19	0.77
51	4	41.7	2.45	0.65	4	41.7	2.19	0.77
52	4	41.7	2.45	0.65	4.5	55.2	2.30	0.79
53	4.5	46.9	2.43	0.70	5	61.3	2.28	0.84
54	5	52.1	2.41	0.74	5	61.3	2.28	0.84

Other variations in SDOB and powder factor for decks 3, 4 and 5 are shown in Figs. 8-7 to 8-10. In these cases Iregel or Energel slurry explosive are used in the bottom blasthole deck. In these graphs it may be observed that there are many cases where an increase of scaled depth of burial with overburden depth coincides with a decrease in powder factor.

This is again a function of the field practice of approximating charge lengths to the nearest six inches. As overburden depth increases, there will, for a given charge length, be an increasing distance of stemming to the next deck. The same charge weight acts over a larger volume, so powder factors decrease. These trends occur until an overburden depth is encountered where the next six-inch increment of explosive column length is employed.

8.3. SUBSIDENCE AND POST-BLAST PROFILES

One of the major concerns with abandoned mine land is mining subsidence. Not only does this render the land unusable, but it represents a potential safety hazard. From a safety point of view, potential subsidence is much more critical than the actual presence of sinkholes. One of the major aims of blasting as an AML reclamation tool is the elimination of potential subsidence.

In order that any reclamation method is to be successful, it is very important to have some basic understanding as to how "natural" subsidence occurs above mined-out underground

FIG. 8-7 BLAST TECHNICAL DATA ANALYSIS

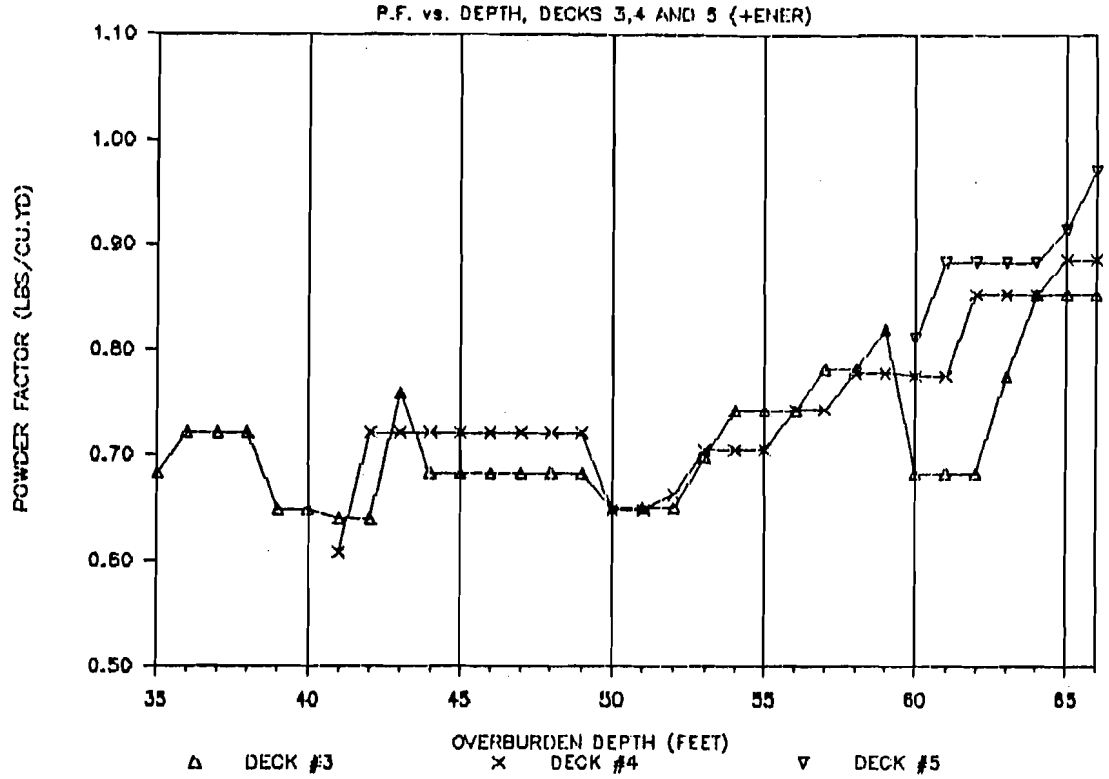


FIG. 8-8 BLAST TECHNICAL DATA ANALYSIS

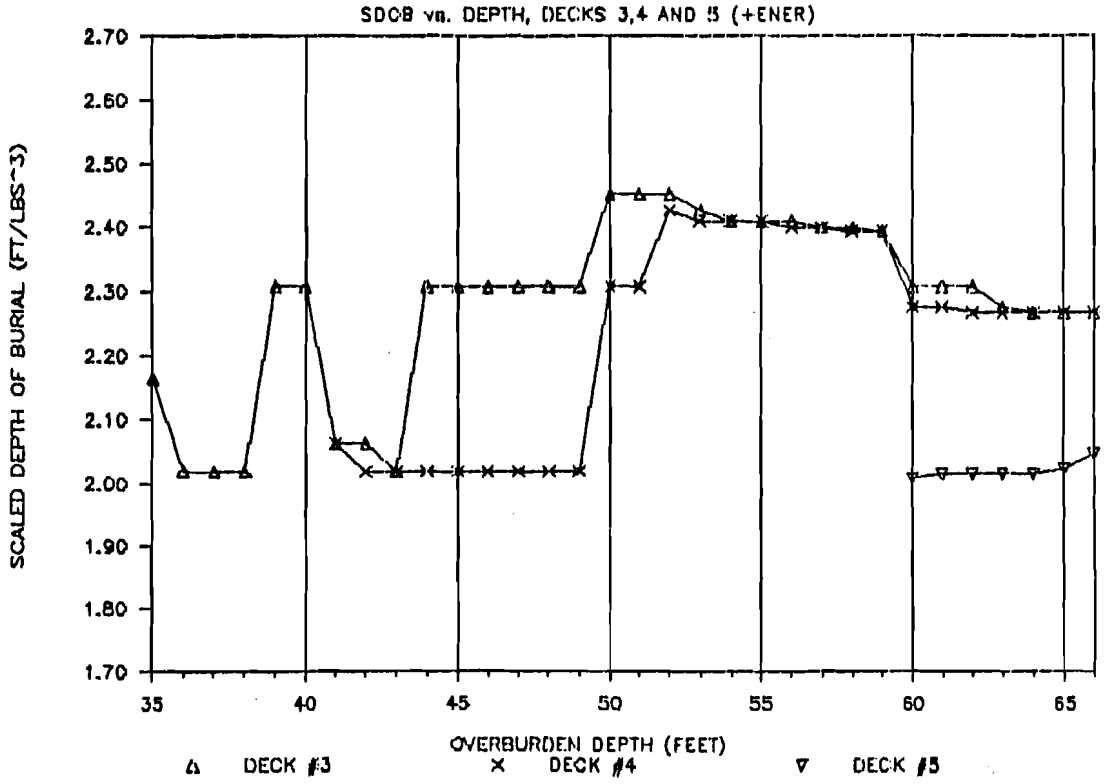


FIG. 8-9 BLAST TECHNICAL DATA ANALYSIS

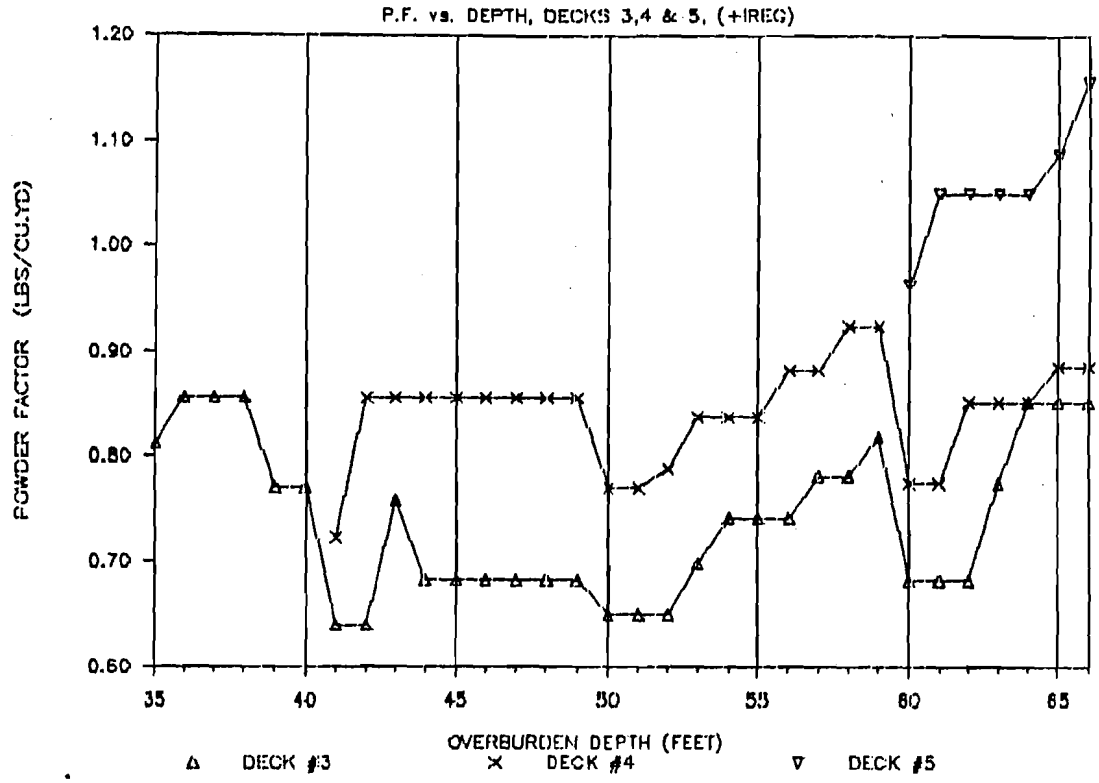
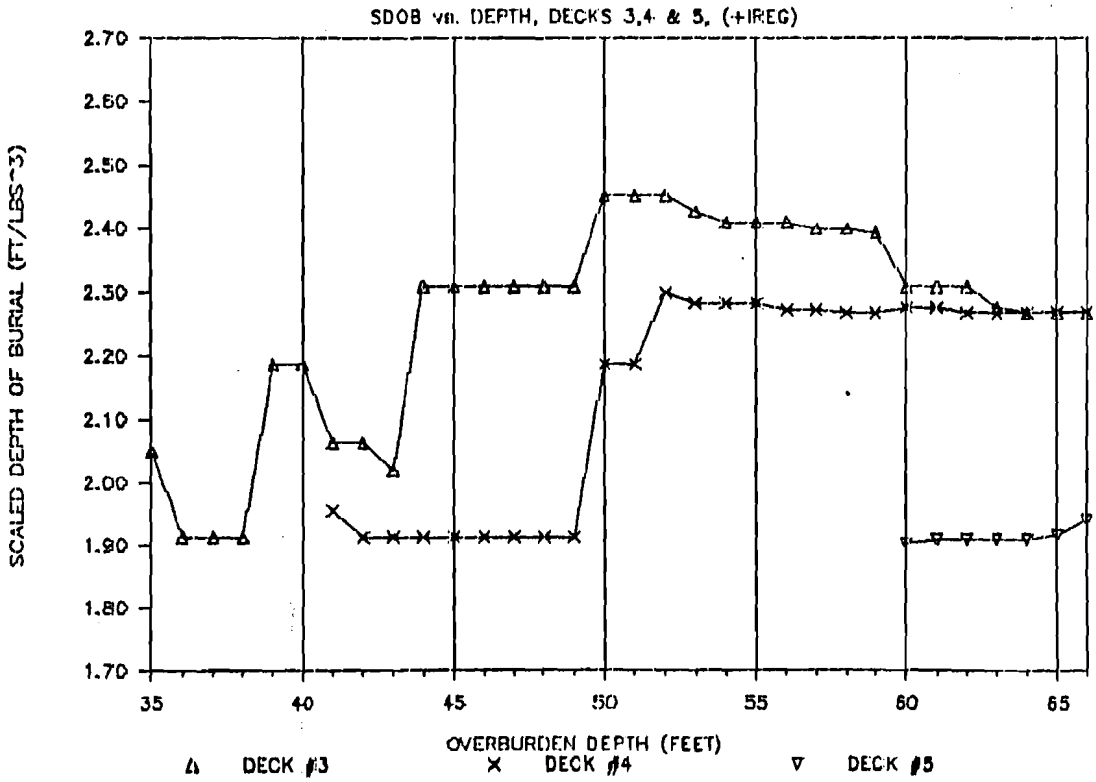


FIG. 8-10 BLAST TECHNICAL DATA ANALYSIS



openings. This section of the report deals briefly with this, and discusses in more detail the effect of blast-induced subsidence on post-blast land profiles. Results of some field measurements from the test blasting program are presented, and a model for blast-induced subsidence is described. The purpose of this model is to reflect actual results, and to provide a rough guideline for future work of this type.

8.3.1. Previous Work

A literature search revealed that there is a surprisingly small amount of previous work on AML subsidence. The reason that this is surprising is that one author, Singh[15], estimates that there are more than 2 million acres of land in the U.S. which have been affected by mining subsidence. This paper presents a largely historic view of the development of different theories regarding mining subsidence.

The various postulated mechanisms of subsidence are concerned primarily with the formation of trough-like depressions over mining development which caves in at some depth. The concern in this project, as with many AML sites, is with near-surface development which actually caves up to surface.

A second paper, by Ghaboussi et al[16], considers the simulation of subsidence over soft-ground tunnels using the finite element method. Again this paper does not address the practical aspects of near-surface development in soft materials, or a satisfactory subsidence model.

A study by Arcamone et al[17] considers the influence of the overburden material on mining subsidence, and considers the effects of stratigraphic factors (strength and thickness of bedded rocks) and structural factors (presence of faults and other major geologic structures). The paper is concerned essentially with the influence of competent overburden on subsidence, and as such is not applicable to the case being considered for AML sites such as that at Beulah with about 50 feet of clay/sand overburden.

It is apparent, therefore, that little has been reported about AML subsidence that is pertinent to this study. There exists the need, therefore, to briefly address the problem as it affects AML sites such as that selected for the test blasting program.

8.3.2. Mechanisms of AML Subsidence

A possible mechanism for "natural" AML subsidence is illustrated in Fig. 8-11. The first stage is the breakage of the roof coal layer which was often left in the development for natural support. Sloughing of the coal is most likely due to the size of the unsupported span, and weakening of the coal over

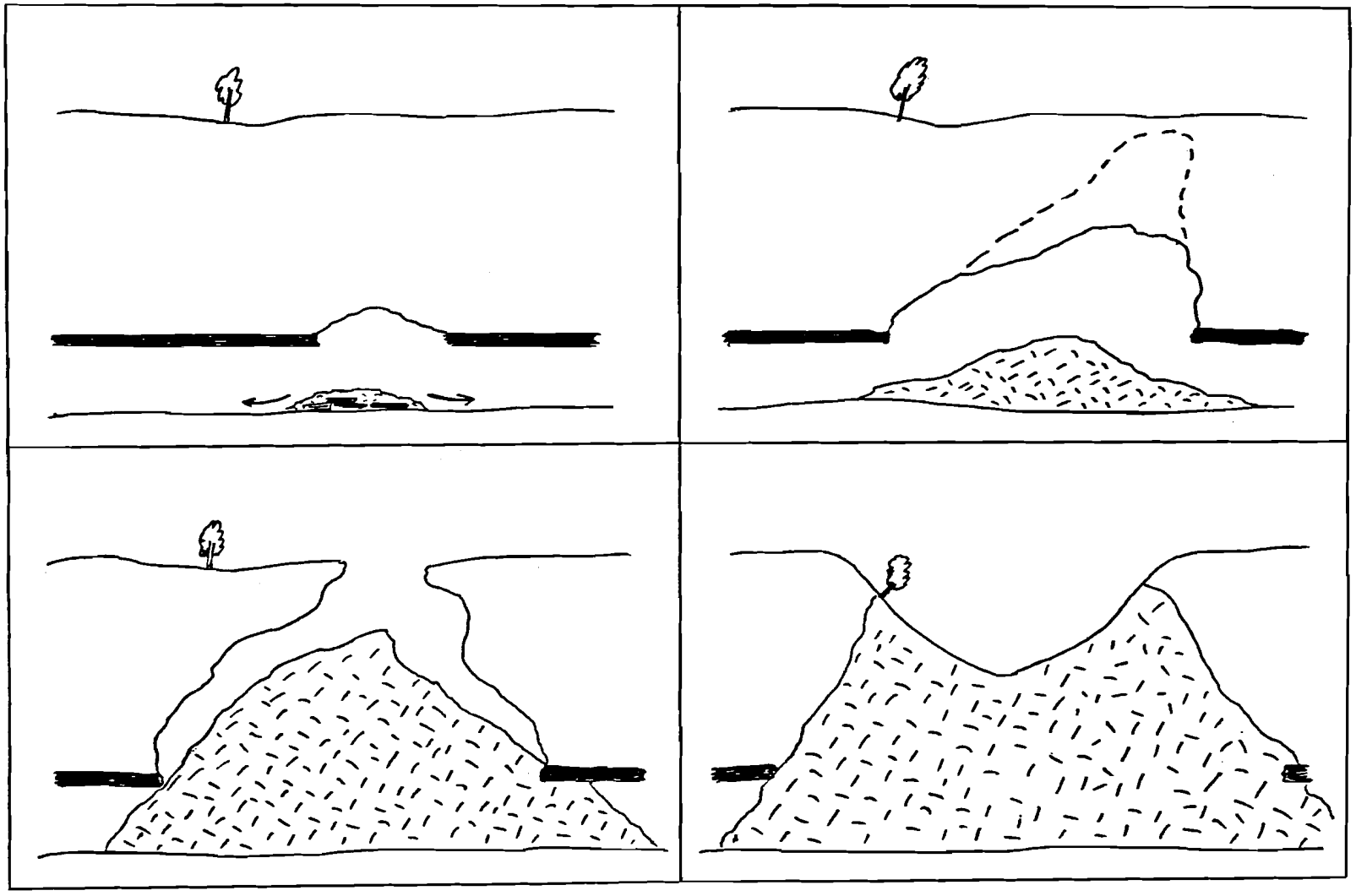


FIG. 8-11: CONCEPTUAL ILLUSTRATION OF THE STAGES OF THE FORMATION OF A "NATURAL" SINKHOLE ABOVE ABANDONED UNDERGROUND DEVELOPMENT

time due to oxidation by air and water. The thickness of coal would, based on experience at the test blast site, be greatest in the more permanent types of access development, such as the panel entries. Intersection points for development, such as cross-cut with panel entry, and room with panel entry, would also tend to have higher roof coal thicknesses to support the large openings at these points.

In overburden conditions such as those at the Beulah test site it is likely that the over consolidated clays and sandy clays would start to fail into the opening quite soon once there was no roof-coal barrier to prevent this. If the development contained water, the fallen material would tend to erode, and gradually flow away from the source area as seasonal and other fluctuations in the level of mine water occur.

As the area over which overburden material is eroding away gets larger, a build-up is likely in the room itself, which will fill up behind the fallen material. At this point there is likelihood of larger slumps of material, especially when undercutting action has developed (see Fig. 8-11).

Eventually a point is reached where there may only be a few feet of unsupported material forming the "roof" to the feature. This obviously represents the state of maximum potential danger, where the weight of a vehicle, or even a person, may cause the structure to cave in. This hole may only be a few feet across at the top, but widens out considerably below.

The surface expression of this process is shown in photographs from actual examples from an AML site, in Figs. 8-12 to 8-15. The first of these shows the development of a circular series of cracks (Fig. 8-12); the second photo shows an initial settling of the ground surface prior to slumping (Fig. 8-13).

In the third photo (Fig. 8-14) the sinkhole has daylighted, and a very small surface feature in the middle distance is a second sinkhole, only about two feet across at surface. The final photo (Fig. 8-15) is a close-up of a newly-developed sinkhole. It shows that the surface expression of the feature is considerably smaller than its extent underground. In time the undercut portion will also cave along the underlying room, and the shape of the feature will be rectangular, rather than circular.

Once the sinkhole has "daylighted", the action of surface erosion, mainly rain water and snow-melt, is added to the gravity-controlled erosion going on below, and in time the sinkhole takes on the rounded profiles typical of "mature" AML areas.

8.3.3. Roof Coal and Void Depth Measurement

The depth of roof coal has been mentioned elsewhere in this

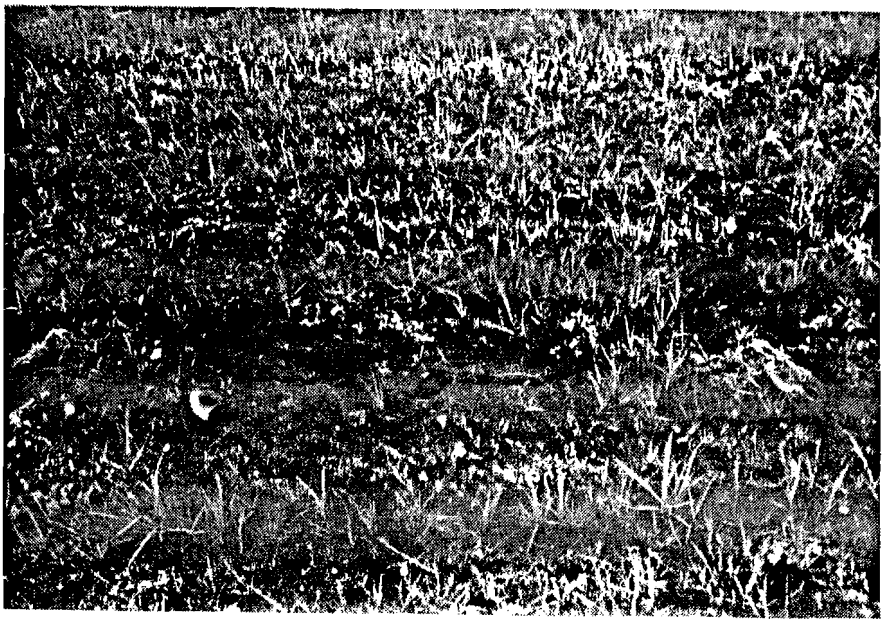


FIG. 8-12: FIRST STAGE IN SURFACE EXPRESSION OF SINKHOLE DEVELOPMENT - CIRCULAR SURFACE CRACKS

FIG. 8-13: SECOND STAGE IN SURFACE EXPRESSION OF SINKHOLE DEVELOPMENT - CIRCULAR AREA BEGINS TO SLUMP

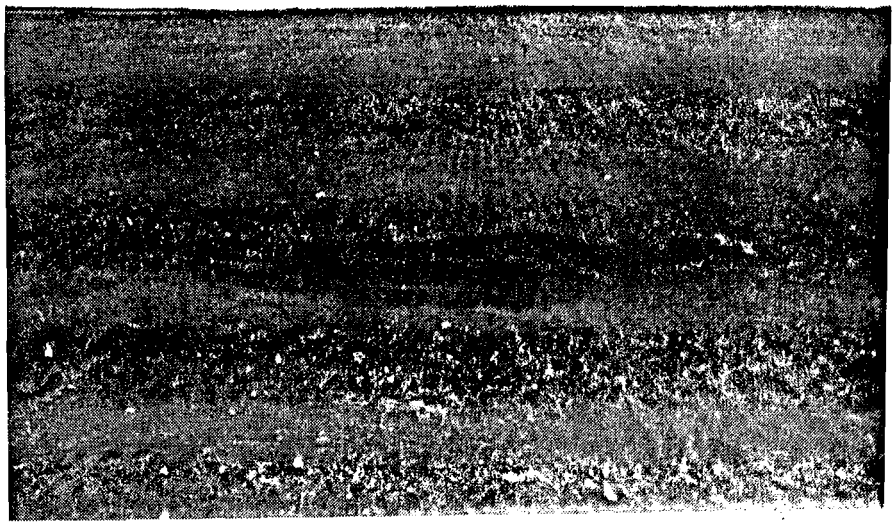


FIG. 8-14: THIRD STAGE IN SURFACE EXPRESSION OF SINKHOLE DEVELOPMENT - SINKHOLE BREAKS THROUGH TO SURFACE

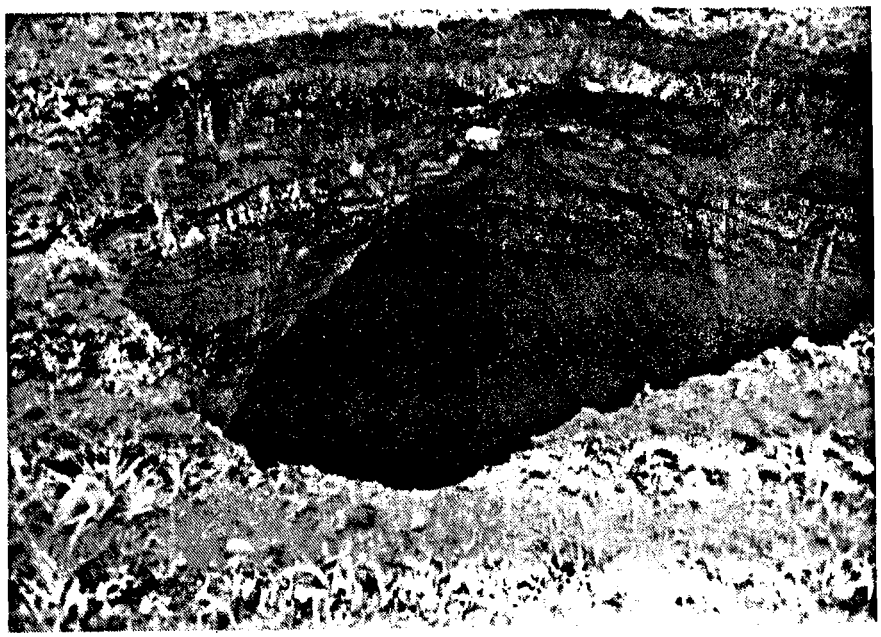


FIG. 8-15: CLOSE-UP OF NEWLY DEVELOPED SINKHOLE, SHOWING UNDERCUTTING OF SURFACE HOLE

report as an important consideration for blast design. It was discussed above in relation to its supporting effect on abandoned underground development.

The measurement in the field of roof coal thickness was thus carried out during the test blasting program (see Chapter 5). This was found to be typically 3-4 feet in rooms, though thickening at their intersection with an entry-way to 5-6 feet. The Panel Entries and cross-cuts, which required greater permanence and therefore a greater degree of natural support, had roof coal thicknesses in the range of 6-7 feet in general.

The coal seam in the test site area was found to be about 18-20 feet thick. Since a sticky clay underlay the coal it is unlikely that mining was carried out to the bottom of the seam. It may be assumed that in rooms about one foot of coal was left in the floor. To avoid water and drainage problems, the floor level for access development was probably maintained a little higher. After mining there was inevitably some filling of this void by material dumped from suspension in mine waters, and from spalling from roof and walls.

The depth of the void remaining prior to blasting was obtained by subtracting the distance to the void from the distance to the floor of the opening, as explained in Chapter 5 of this report. Typical roof coal and void depth figures for the test site area are shown in Table 8-7.

TABLE 8-7 : TYPICAL ROOF COAL AND VOID DEPTH VALUES FOR DIFFERENT DEVELOPMENT TYPES AT THE TEST SITE.

Development Type	Roof Coal	Mined void depth (ft)	Measured void depth (ft)
Rooms	3-4 ft	14-15	10-14
Rooms adj. to entries	5-6 ft	11-12	7-10
Access development	6-7 ft	10-11	6-9

It will shortly be seen that the measurement of the pre-blast void space is very important for the application of the subsidence model proposed for this aspect of AML blasting work.

8.3.4. Measurement of Post-blast Subsidence Profiles

Numerous photographs included in Chapter 4 of this report indicate that in the majority of cases the post-blast profile consisted of a V-shaped depression running along the center of

the blasted room. In some areas the ground surface was disturbed, but remained at the same general elevation as it had been prior to blasting. In other cases, there was surface heave along the length of the blast.

During field blasting testwork it was found that deep holes developed at the ends of some blasts. This occurred where they were terminated against unblasted voids, and the extra depth was certainly due to lateral migration of material into these voids. Many of the deeper holes filled up over the course of a day or two, due to spalling of loose material from the initially near-vertical walls. The post-blast profiles in the main part of the blasts themselves did not change significantly, however, even by the Spring of 1987 after snow-melt.

Some measurements were taken in the field of the depth and width of post-blast features. However, the early onset of winter, plus the desire to allow for some natural infilling to take place, meant that the majority of measurement work was carried out some 5-6 months after blasting.

Measurement was carried out on 11 of the test blasts, and was not carried out to any great degree of accuracy. A tape was stretched across the sinkhole at various positions along its length, and the width measured. Where there was obvious backbreak, this was recorded, but the measurement of main interest was that of the limits of actual slumping movement into the sinkhole. The depth of the central, deepest part of the sinkhole was recorded at the same position.

In some cases it was possible to locate these measurements fairly accurately by means of the identification numbers on wooden stakes marking former blasthole positions. In other cases, these locations were estimated by pacing out the distance along the length of the blast. Where surface heave resulted, the approximate height and width of this was also measured.

Results from this measurement are tabulated in the columns headed "SINKHOLE WIDTH" and "SINKHOLE DEPTH" in Table 8-8. Negative values for "depth" actually refer to the approximate height of material swell above the general ground surface elevation. Measurements taken where there was definite evidence to suggest lateral movement of material into adjacent unblasted voids are indicated in this table. Each blast measured is identified in this table by its number and location in the field testing program. The other parameters in this table are discussed in the next section of this report.

8.3.5. Model for Blast-induced Subsidence

The model employed to represent blast-induced subsidence for the AML test site is a simple geometric model. It can be resolved in two dimensions only, and considers two volumes of

TABLE 8-8 : CALCULATION OF SWELL FACTOR FOR BLASTED MATERIAL BASED ON FIELD SUBSIDENCE MEASUREMENTS (DISTANCES IN FEET)

BLAST #	LOCATION	# ROWS	ROOM WIDTH	O'BURDEN DEPTH	ROOF COAL	VOID DEPTH	SINKHOLE WIDTH	SINKHOLE DEPTH	SWELL FACTOR
4	N-1,N-2	1,2	12	66.5	7.0	8.5	-	-4.0	1.168
			22	66.5	7.0	7.5	-	-6.0	1.178
			22	63.0	4.0	6.0	-	-4.0	1.147
			12	65.0	8.0	7.0	-	-4.5	1.151
			22	64.0	6.5	4.0	-	-1.0	1.077 *
			12	64.5	6.5	7.0	12.0	15.0	0.992 *
5	N-13	2	22	48.0	9.5	9.5	24.0	10.0	1.081 *
			22	49.0	4.0	11.0	18.0	5.0	1.201
			22	48.5	4.0	11.5	18.0	6.0	1.205
			22	48.0	1.5	12.0	30.0	8.0	1.115
			22	49.5	3.0	12.0	24.0	10.0	1.126
			22	50.0	3.0	12.5	32.0	15.0	1.026 *
10	S-1(N)	2	22	64.0	5.5	6.0	-	-5.0	1.154
			22	59.0	3.0	7.0	-	-4.0	1.174
			22	59.0	3.5	7.5	-	-3.0	1.171
11	S-12	2	22	40.0	3.0	8.0	22.0	10.0	1.075
			22	41.0	3.5	11.0	26.0	14.0	1.061
			22	41.0	4.0	12.0	30.0	9.0	1.121
			22	41.0	3.5	11.5	28.0	4.0	1.192
			22	35.0	0.0	15.0	25.0	6.0	1.310
			22	41.0	3.0	11.0	16.0	5.0	1.259
			22	43.0	5.0	8.0	22.0	0.0	1.186
12	N-8(N)	2	22	54.0	0.0	1.0	-	-3.5	1.073
			22	58.0	3.0	7.0	17.0	3.0	1.114
			22	58.0	4.5	8.5	22.0	6.0	1.095
			22	56.0	1.5	8.0	28.0	8.0	1.046
			22	56.0	2.0	8.0	22.0	10.0	1.054
13	N-10	2	22	54.0	4.0	7.5	-	-1.5	1.165
			22	54.0	3.0	6.5	25.0	3.0	1.083
			22	58.0	4.5	6.0	30.0	6.0	1.028
			22	58.5	5.0	8.5	26.0	13.0	1.013
			22	58.0	5.0	9.0	27.0	8.0	1.063
			22	59.0	5.0	8.0	25.0	7.0	1.064
15	S-1(S)	2	22	55.0	4.5	1.5	-	-3.0	1.074
			22	57.0	5.5	7.5	-	-4.0	1.189
			22	55.0	5.0	9.0	22.0	4.0	1.127 *
17	S-10	2	22	43.0	3.0	12.5	31.0	12.0	1.078 *
			22	42.5	3.0	13.5	30.0	10.0	1.133
			22	42.0	3.0	13.5	30.0	9.0	1.148
			22	43.0	3.0	12.0	24.0	4.0	1.218
			22	42.0	1.0	5.5	22.0	0.0	1.131
18	N-6	1	20	61.0	7.0	2.5	22.0	-2.0	1.070
			20	57.5	1.0	1.5	22.0	-1.0	1.043
			20	55.0	0.5	4.0	22.0	0.0	1.069
			20	57.0	4.0	1.5	22.0	-4.0	1.082
			20	55.0	3.0	5.0	18.0	6.0	1.044 *
19	S-9(N)	2	22	43.0	1.5	13.0	25.0	15.0	1.097 *
			22	47.0	3.0	12.0	35.0	13.0	1.027
			22	46.0	2.0	10.5	28.0	5.0	1.140
			22	50.0	6.0	12.0	25.0	19.0	1.023 *
21	S-9(S)	1	22	44.5	3.0	11.0	23.0	14.0	1.081
			22	44.5	3.0	11.5	20.0	13.0	1.132
			22	45.5	5.5	9.5	23.0	8.0	1.114
			22	45.5	3.5	8.5	20.0	10.0	1.091
			22	45.0	3.0	5.0	22.0	0.0	1.111
			22	37.0	0.0	4.0	22.0	8.0	1.000 *

* Value suspect due to adjacent unblasted void into which material could move laterally.

material. The first is that volume initially located above the underground void. This is disturbed by blasting and then settles into a second volume which includes the actual void space available.

Two sub-models are actually employed, one which models the formation of a surface depression, and another which attempts to simulate surface heave.

Based on actual field measurements, these sub-models are used to estimate the average swell induced in the material as a result of blasting. Once this has been obtained, an attempt is made to use typical site parameters for the test blast site to predict likely post-blast profiles in sites of a similar nature.

In each case the following site and field measurement parameters are required:

- width of development (RW)
- depth of overburden (D)
- depth of pre-blast void (VD)
- depth of post-blast sinkhole or height of surface heave (H)
- width of post-blast sinkhole, if present (SW)

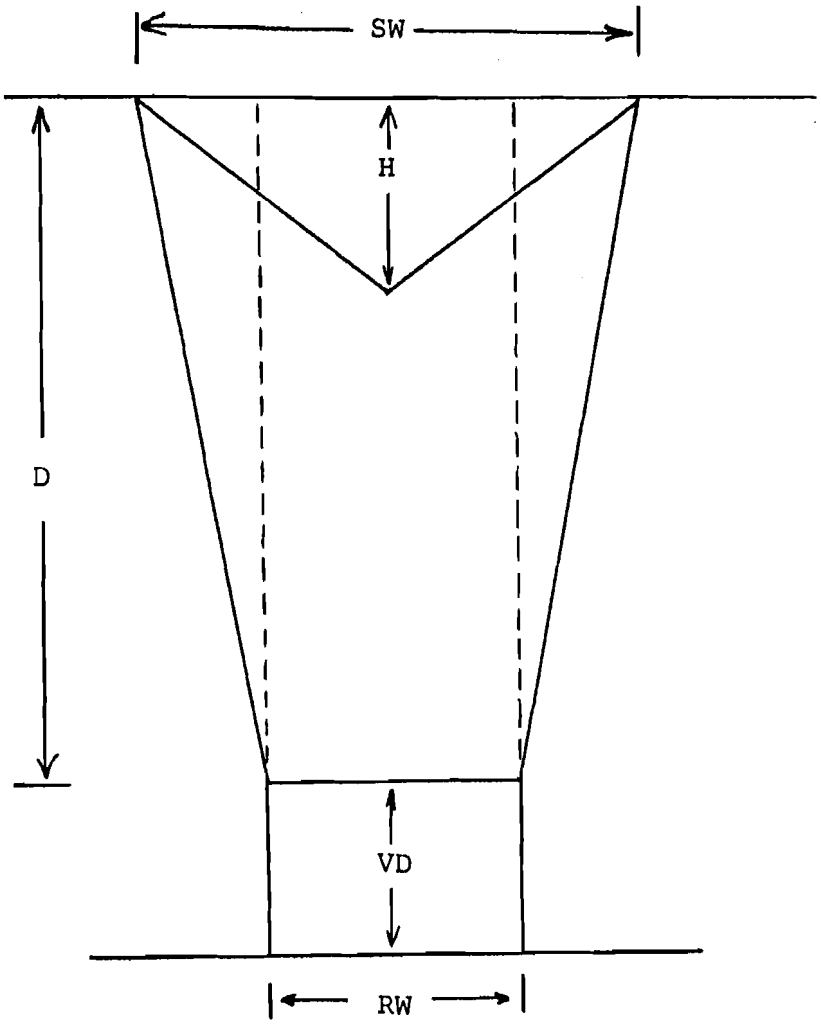
8.3.5.1. Surface Depressions

The geometric model used to calculate volumes for post-blast sinkholes is shown in Fig. 8-16, together with the terminology used and trigonometric equations developed. It can be seen that the model assumes that material vertically above the void is affected. In addittion, two wedge-shaped areas are considered, which are associated with back-break (ie. where SW exceeds RW).

The final area occupied by the blasted material is equal to the initial area, plus the available void area, minus the area of the depression itself.

8.3.5.2. Surface Heave

In the case where the ground heaves above the previous surface elevation a different geometrical model is applied. It has been assumed, to simplify calculations, that the width of the surface heave feature is always equal to the room width. In practice this was found to be generally the case. A second assumption that is made is that the sides of the heave structure slope at 45 degrees. The width across the top of the heaved area, M, is thus a function of the height, H, of the surface heave. The model is illustrated in Fig. 8-17, which also shows the terminology used and the trigonometric relationships developed.



UNSWELLED VOLUME $\quad VOL1 = \frac{SW + RW}{2} \times D$

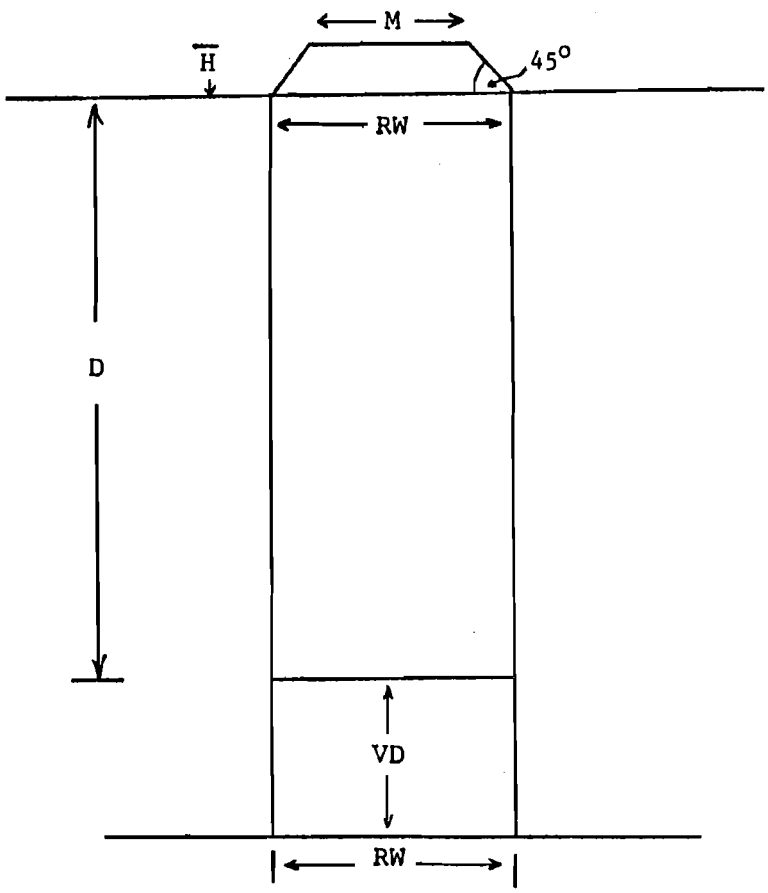
SWELLED VOLUME $\quad VOL2 = VOL1 + (RW \times VD) - (SW/2 \times H)$

SWELL FACTOR $\quad SF = \frac{VOL2}{VOL1} = 1 + \frac{(RW \times VD) - (SW/2 \times H)}{(SW + RW)/2 \times D}$

DEPTH OF BLAST-INDUCED SINKHOLE

$$H = \frac{VOL1 - VOL2 + (RW \times VD)}{SW / 2}$$

FIG. 8-16: GEOMETRIC MODEL FOR CALCULATION OF DEPTH OF BLAST-INDUCED SINKHOLE



<u>UNSWELLED VOLUME</u> $VOL1 = RW \times D$

<u>SWELLED VOLUME</u> $VOL2 = VOL1 + (RW \times VD) + \frac{(RW + M)}{2} \times H$

If sides of heave are at 45 degrees, then $M = RW - 2H$

and $VOL2 = VOL1 + (RW \times VD) + (RW-H) \times H$

<u>SWELL FACTOR</u> $SF = \frac{VOL2}{VOL1} = 1 + \frac{VD}{D} + \frac{(RW - H) \times H}{RW \times D}$

<u>HEIGHT OF BLAST-INDUCED SURFACE HEAVE</u>

$$H^2 - (RW \times H) + (VOL2 - VOL1 - (RW \times VD) = 0$$

solve quadratic for H

FIG. 8-17: GEOMETRIC MODEL FOR CALCULATION OF HEIGHT OF BLAST-INDUCED SURFACE HEAVE

The initial area is simply that which lies vertically above the room. The final area is equal to the initial area, plus the area of the void, plus the area heaved above surface.

8.3.5.3. Calculation of Material Swell

The material swell due to blasting is simply the ratio of final volume to initial volume for each of the geometric models described above. Trigonometric expressions which may be used to calculate this for each of the geometric models are included in Figs. 8-16 and 8-17.

Table 8-8 is actually a spreadsheet, the last column of which calculates the overburden swell factor. The spreadsheet determines which is the correct trigonometric equation to apply by checking whether the "Sinkhole depth" value is positive or negative. In the latter case, where there was surface heave, the "Sinkhole width" column is not used.

A statistical analysis was carried out, using the spreadsheet, on the calculated swell factors. For reasons explained earlier, those values which were associated with measurements at the ends of blasts, adjacent to unblasted voids, were not considered for this analysis.

An average swell factor of about 12% was obtained. This is not unreasonable, given the nature of the overburden type. In strip mining applications for similar materials an initial swell of around 20% is assumed, which is reduced to the range of 10-15% in the reclaimed spoil condition. It is quite likely that the blasting action, and the "packing" of material into the confined underground void volume, could result in a swell of around 12%. Where swell factors in excess of 25% were calculated, this is evidence that some localized bridging may have occurred in the blast.

Results from the statistical analysis are summarized in Table 8-9.

TABLE 8-9 : STATISTICAL ANALYSIS OF CALCULATIONS OF MATERIAL SWELL DURING TEST BLASTS

# observations	46
mean	1.118
standard deviation	0.057
minimum value	1.013
maximum value	1.310

8.3.5.4. Prediction of Post-blast Profiles

The same geometric models employed to calculate swell factors for actual field measurements from the test blasting program can be used to predict the depth of blast-induced subsidence, or the height of surface heave.

Such information has two uses as far as this research project is concerned. Firstly, it provides a basis to explain some of the results obtained in the test blast program. Secondly, it should provide a rough guide for future work of this kind, by providing an indication of the type of profiles to be expected for a given set of site conditions.

The post-blast profile model is again spreadsheet based. Reorganization of the equations shown in Figs. 8-16 and 8-17 give the following expressions for the post-blast profile:

For a post-blast depression:

$$\text{Depth} = \frac{(\text{Vol1} + \text{RW.VD} - \text{Vol2})}{\text{SW}/2}$$

For surface heave:

Height = positive root of this quadratic equation:

$$(\text{Height})^2 - (\text{RW. Height}) + (\text{Vol2} - \text{Vol1} - \text{RW.VD}) = 0$$

Where
Vol1 = initial volume
Vol2 = final volume
RW = room width
VD = void depth
SW = sinkhole width

Three main types of analysis were carried out using the post-blast profile model. These are variation of post-blast profile with :

- void depth, for a given swell factor and overburden depth
- overburden depth, for a given swell factor and void depth
- swell factor, for a given void and overburden depth

Table 8-10 shows the spreadsheet used to calculate these results. It can be seen that void depths in the range of

TABLE 8-10 : EXAMPLE OF SPREADSHEET-GENERATED RESULTS FROM BLASTING-INDUCED SUBSIDENCE MODEL

SF:	1.12	SW:	25.3	VOL1:	1182.5
RW:	22	VD:	10	VOL2:	1324.4
OD:	50				

VD	H (INT)	H	OD	VOL1	H (INT)	H	SF	H (INT)	H
0.0	-11.2	ERR	35	827.8	9.5	9.5	1.00	17.4	17.4
0.5	-10.3	ERR	36	851.4	9.3	9.3	1.01	16.5	16.5
1.0	-9.5	-10	37	875.1	9.1	9.1	1.02	15.5	15.5
1.5	-8.6	-7.5	38	898.7	8.9	8.9	1.03	14.6	14.6
2.0	-7.7	-6.2	39	922.4	8.6	8.6	1.04	13.7	13.7
2.5	-6.9	-5.2	40	946.0	8.4	8.4	1.05	12.7	12.7
3.0	-6.0	-4.3	41	969.7	8.2	8.2	1.06	11.8	11.8
3.5	-5.1	-3.5	42	993.3	8.0	8.0	1.07	10.8	10.8
4.0	-4.3	-2.8	43	1017.0	7.7	7.7	1.08	9.9	9.9
4.5	-3.4	-2.2	44	1040.6	7.5	7.5	1.09	9.0	9.0
5.0	-2.5	-1.6	45	1064.3	7.3	7.3	1.10	8.0	8.0
5.5	-1.7	-1.0	46	1087.9	7.1	7.1	1.11	7.1	7.1
6.0	-0.8	-0.5	47	1111.6	6.8	6.8	1.12	6.2	6.2
6.5	0.1	0.1	48	1135.2	6.6	6.6	1.13	5.2	5.2
7.0	1.0	1.0	49	1158.9	6.4	6.4	1.14	4.3	4.3
7.5	1.8	1.8	50	1182.5	6.2	6.2	1.15	3.4	3.4
8.0	2.7	2.7	51	1206.2	5.9	5.9	1.16	2.4	2.4
8.5	3.6	3.6	52	1229.8	5.7	5.7	1.17	1.5	1.5
9.0	4.4	4.4	53	1253.5	5.5	5.5	1.18	0.6	0.6
9.5	5.3	5.3	54	1277.1	5.3	5.3	1.19	-0.4	-0.2
10.0	6.2	6.2	55	1300.8	5.1	5.1	1.20	-1.3	-0.8
10.5	7.0	7.0	56	1324.4	4.8	4.8	1.21	-2.2	-1.4
11.0	7.9	7.9	57	1348.1	4.6	4.6	1.22	-3.2	-2.0
11.5	8.8	8.8	58	1371.7	4.4	4.4	1.23	-4.1	-2.7
12.0	9.7	9.7	59	1395.4	4.2	4.2	1.24	-5.0	-3.4
12.5	10.5	10.5	60	1419.0	3.9	3.9	1.25	-6.0	-4.3
13.0	11.4	11.4	61	1442.7	3.7	3.7	1.26	-6.9	-5.2
13.5	12.3	12.3	62	1466.3	3.5	3.5	1.27	-7.8	-6.3
14.0	13.1	13.1	63	1490.0	3.3	3.3	1.28	-8.8	-7.9
14.5	14.0	14.0	64	1513.6	3.0	3.0	1.29	-9.7	ERR
15.0	14.9	14.9	65	1537.3	2.8	2.8	1.30	-10.7	ERR
			66	1560.9	2.6	2.6	1.31	-11.6	ERR
							1.32	-12.5	ERR
							1.33	-13.5	ERR
							1.34	-14.4	ERR
							1.35	-15.3	ERR
EFFECT OF VARIABLE VOID DEPTH (FIG. 8-18)			EFFECT OF VARIABLE OVERBURDEN DEPTH (FIG. 8-19)				EFFECT OF VARIABLE SWELL FACTOR (FIG. 8-20)		

SF: SWELL FACTOR
RW: ROOM WIDTH
OD: OVERBURDEN DEPTH
SW: SINKHOLE WIDTH (SET AT 1.15 x RW)
VD: VOID DEPTH

VOL1: UNSWELLED VOLUME
VOL2: SWELLED VOLUME
H(INT): INTERMEDIATE CALCULATED VALUE
H: DEPTH/HEIGHT OF POST-BLAST PROFILE (NEGATIVE VALUE = SURFACE HEAVE)
ERR: OUTSIDE RANGE COVERED BY MODEL

zero (void already filled) to 15 feet are considered. Overburden depths in the range 35-66 feet are employed, which were those used for other types of analysis carried out in this report. Swell factors ranging from 1.00 (no material swell) to 1.35 (35% swell) were used in the third analysis type.

In Table 8-10 a swell factor (SF) of 1.12, that encountered from field results at the test site, was used for the first two analyses. A void depth (VD) of 10 feet was used to generate the second and third sets of data. An overburden depth (OD) of 50 feet was employed for the study of the depth/height (H) of the post-blast profile in the first and third analyses. In each case a 22 foot wide room was considered, and it was assumed that the width of the post-blast depression (SW) was 15% higher than the room width (RW).

In each case an intermediate value (H(INT)) is calculated. If this is negative, implying material formed surface heave, this value is calculated by the quadratic solution described above. Any values marked "ERR" are those which cannot be calculated using the geometric model developed for the surface heave situation.

It can be seen from Table 8-10 that there are certain critical values of the varied parameter at which the creation of a post-blast depression is no longer possible, and surface swell results. For example, it can be seen that no depression will be created for swell of 12%, in 50 feet overburden conditions, if the available void depth is less than 6.5 feet.

For all overburden depths in the range 35-66 feet, however, it should be possible to form a surface depression if the available void is 10 feet deep and material swell is 12%. The critical swell factor for void depth of 10 feet and 50 feet overburden is 1.18 - if material swell exceeds this, there is more material than can be accommodated in the final volume including the void, and surface heave results.

This data is presented graphically in Figs. 8-18, 8-19 and 8-20. Such graphs could be useful for predicting results from future work of this kind. Critical void depths at lower (35 feet) and higher (65 feet) overburden depths are analyzed graphically in Figs. 8-21 and 8-22 respectively. Overburden depth is studied using a lower void depth (7.5 feet) in Fig. 8-23, and the critical swell factor is found to be 1.14 if this same void depth exists at 50 feet overburden (Fig. 8-24).

There are obviously many more combinations of swell factor, void depth and overburden depth that can be analyzed. Tables 8-11, 8-12 and 8-13 include some potentially useful results from this part of the technical analysis work.

FIG. 8-18 : POST-BLAST PROFILES

VS. VOID DEPTH (SF=1.12,OD=50',RW=22')

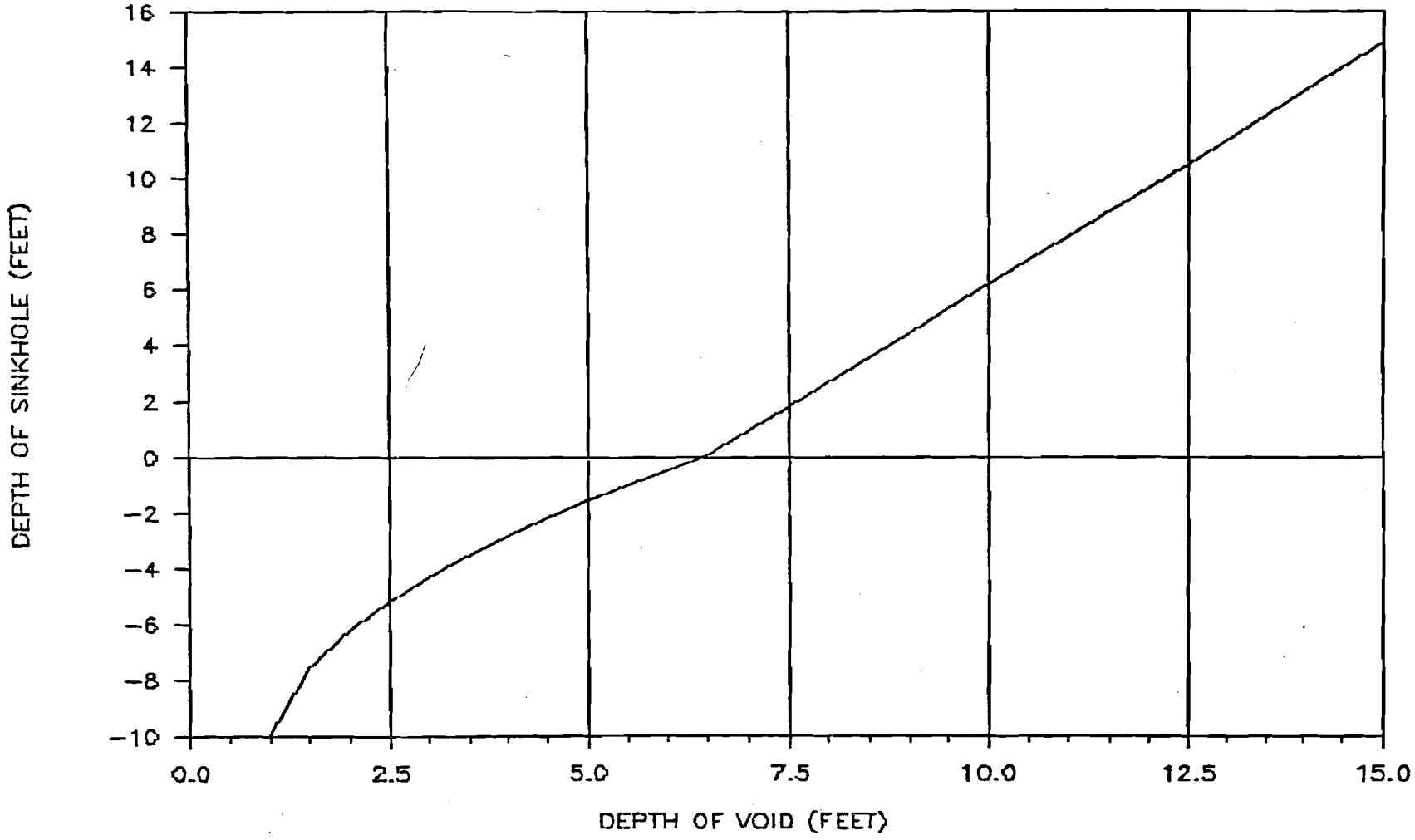


FIG. 8-19 : POST-BLAST PROFILES

VS. O'BURDEN DEPTH (SF=1.12,VD=10,RW=22

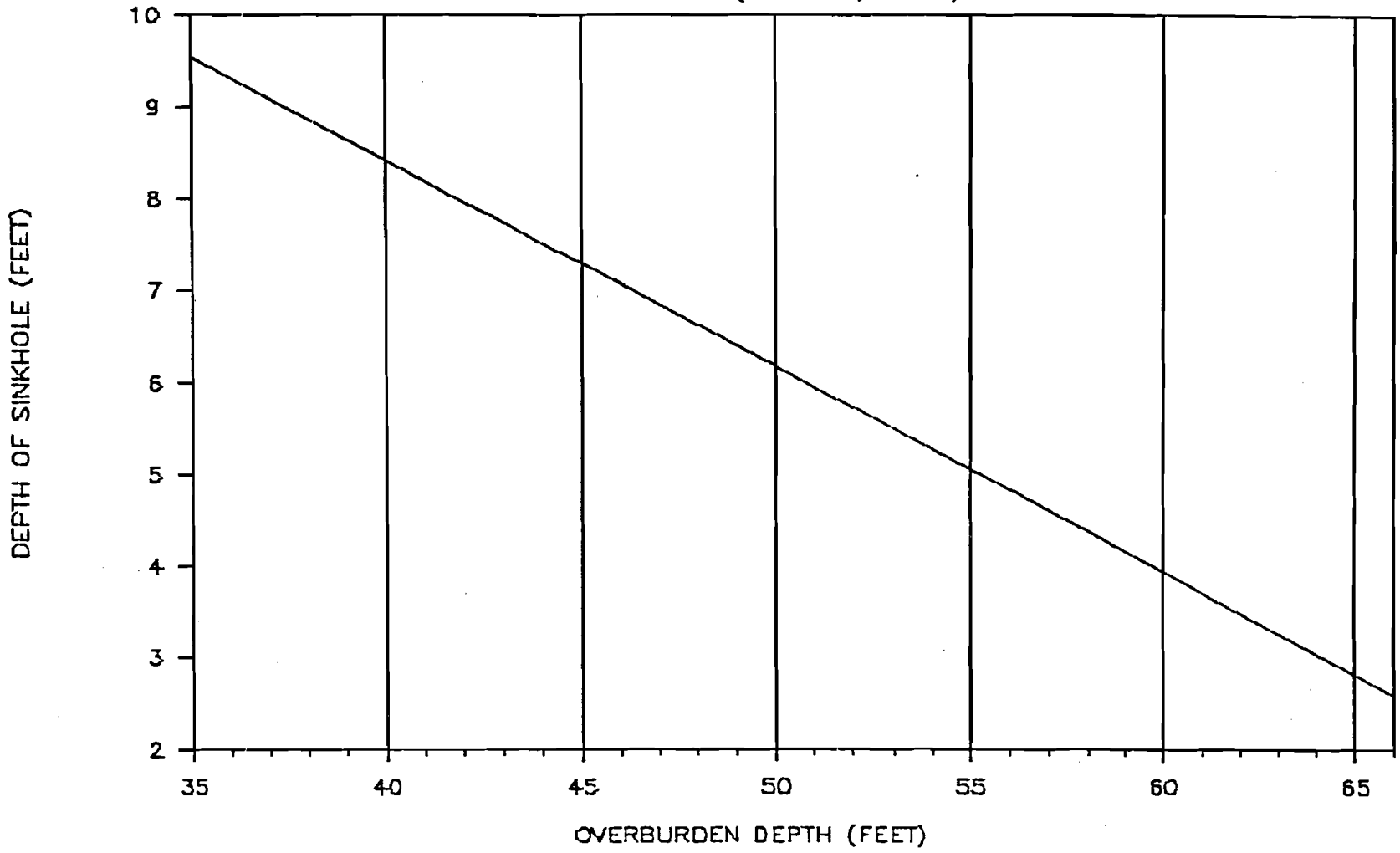


FIG. 8-20 : POST-BLAST PROFILES

VS. SWELL FACTOR (OD=50',VD=10',RW=22')

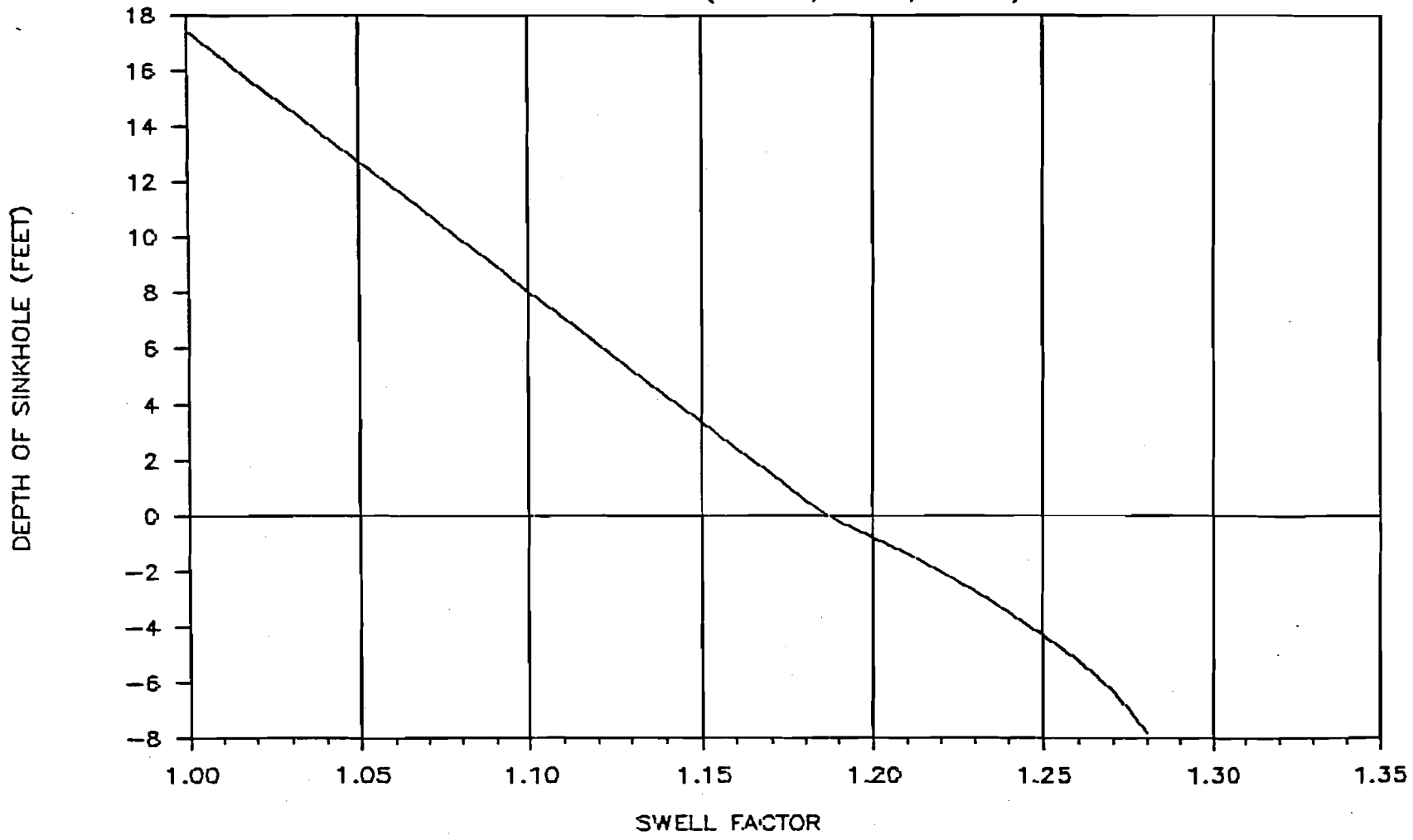


FIG. 8-21 : POST-BLAST PROFILES

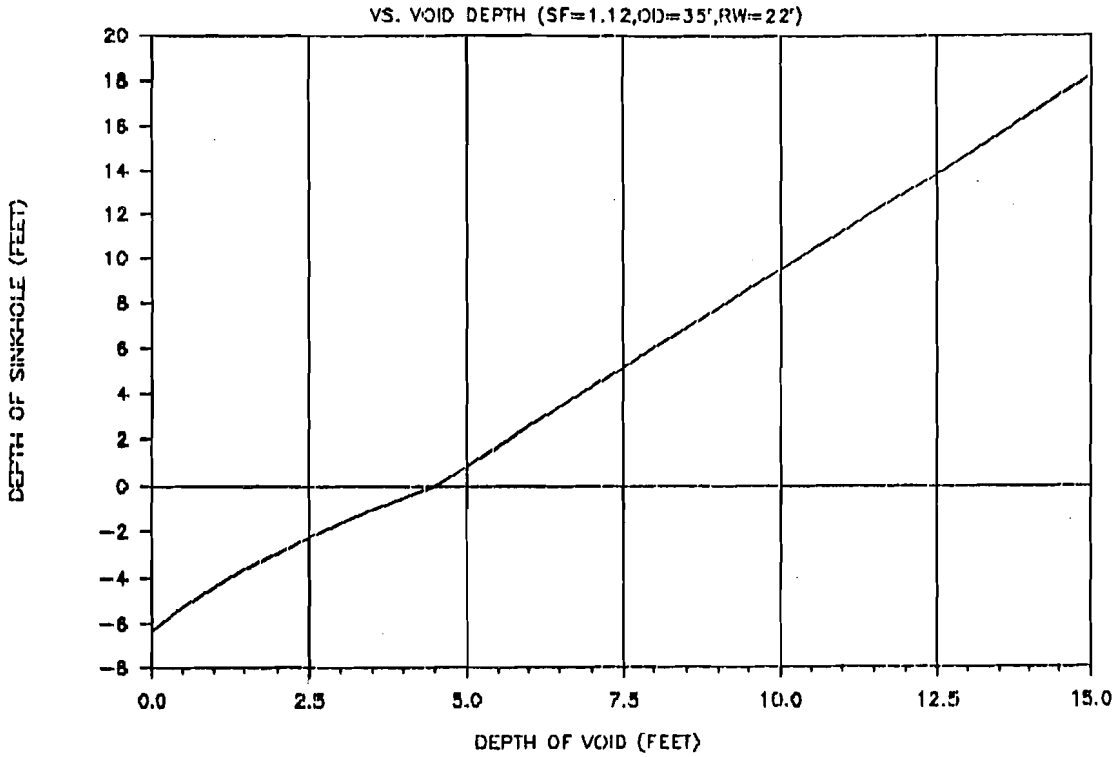


FIG. 8-22 : POST-BLAST PROFILES

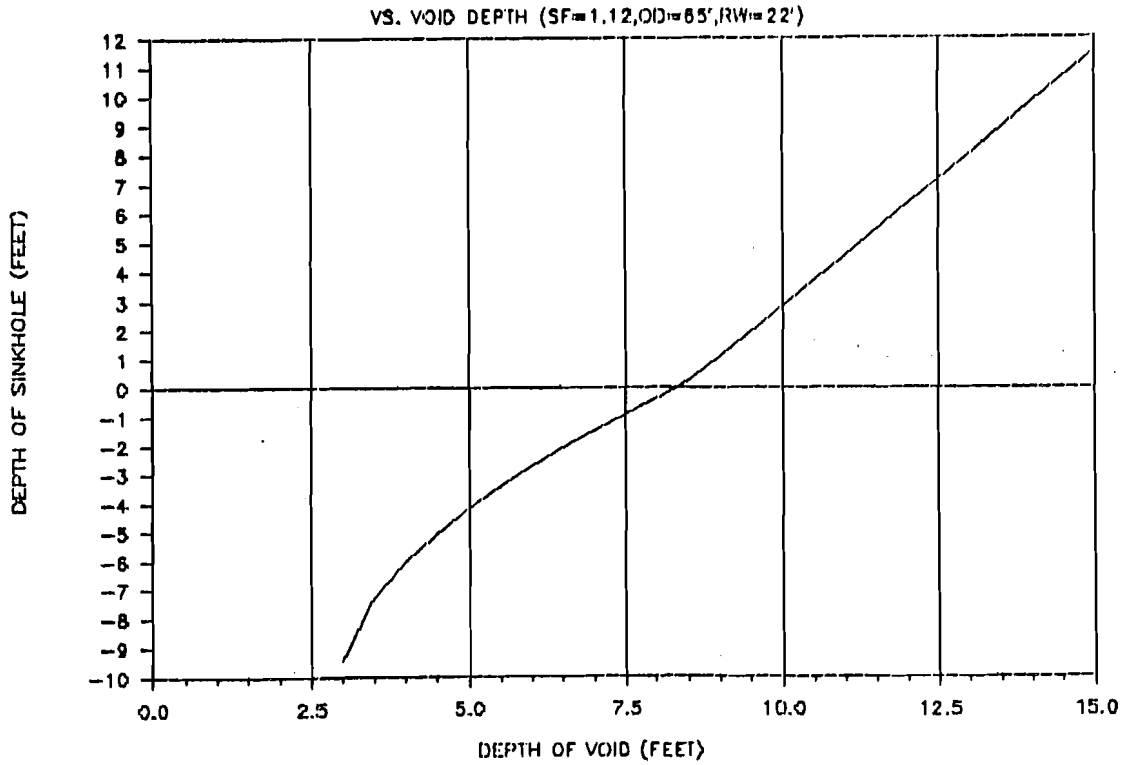


FIG. 8-23 : POST-BLAST PROFILES

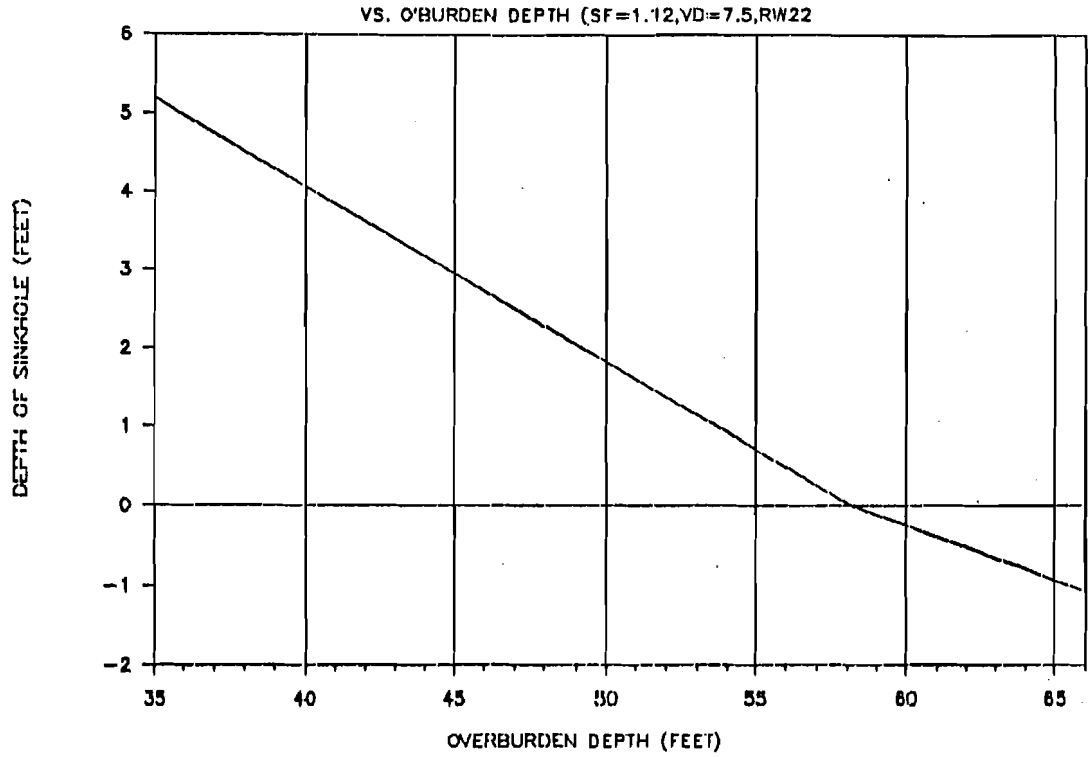


FIG. 8-24 : POST-BLAST PROFILES

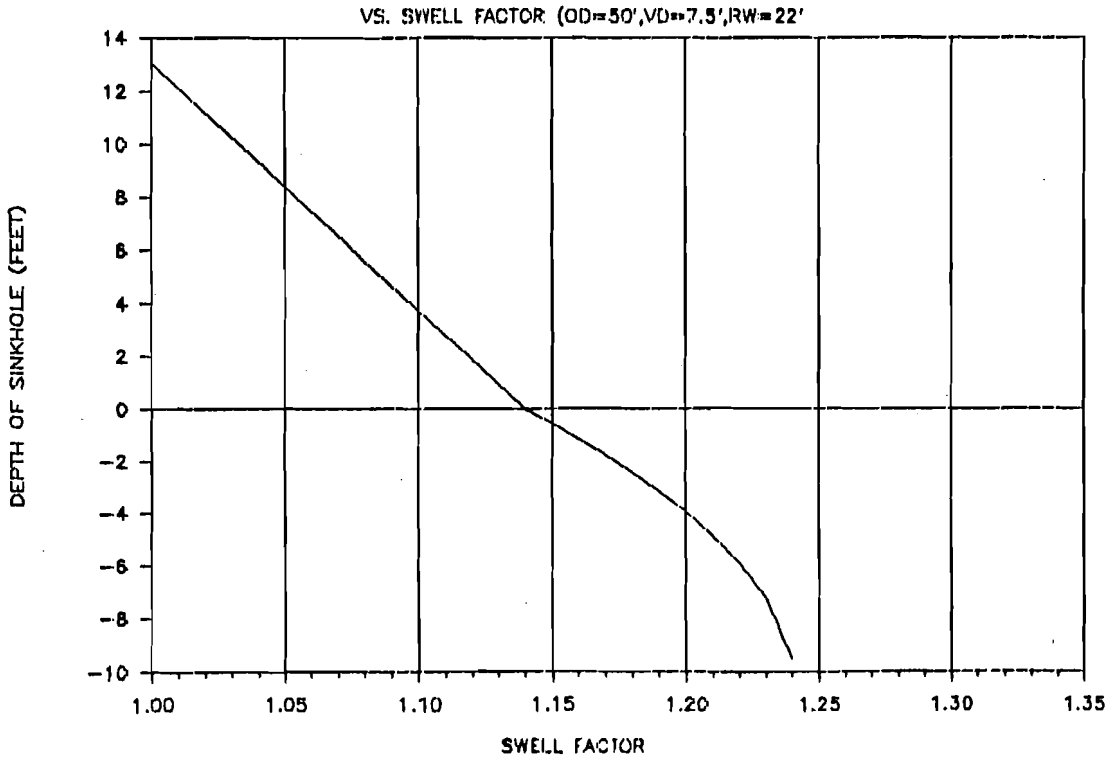


TABLE 8-11 : CRITICAL OVERBURDEN DEPTHS AT WHICH SURFACE SWELL MAY OCCUR AFTER AML BLASTING.

Variable : OVERBURDEN DEPTH							
SF 1.10 RW 22		SF 1.12 RW 22		SF 1.15 RW 22		SF 1.20 RW 22	
VD	OD_c	VD	OD_c	VD	OD_c	VD	OD_c
5	46	5	38	5	< 35	5	< 35
7.5	> 66	7.5	58	7.5	46	7.5	35
10	-	10	> 66	10	62	10	46
12.5	-	12.5	-	12.5	> 66	12.5	58

TABLE 8-12 : CRITICAL VOID DEPTHS AT WHICH SURFACE SWELL MAY OCCUR AFTER AML BLASTING.

Variable : VOID DEPTH							
SF 1.10 RW 22		SF 1.12 RW 22		SF 1.15 RW 22		SF 1.20 RW 22	
OD	VD_c	OD	VD_c	OD	VD_c	OD	VD_c
35	3.5	35	4.5	35	5.5	35	7.5
45	4.5	45	5.5	45	7.0	45	9.5
55	5.5	55	7.0	55	8.5	55	11.5
65	6.5	65	8.0	65	10.0	65	13.5

SF = Swell Factor RW = Room width
OD = Overburden depth VD = Void depth

OD_c = Depth of overburden above which surface heave occurs
VD_c = Depth of void below which surface heave occurs
SF_c = Swell factor above which surface heave occurs

TABLE 8-13 : CRITICAL SWELL FACTORS AT WHICH SURFACE SWELL MAY OCCUR AFTER AML BLASTING.

Variable : SWELL FACTOR							
OD 35 RW 22		OD 45 RW 22		OD 55 RW 22		OD 65 RW 22	
VD	SF_c	VD	SF_c	VD	SF_c	VD	SF_c
5	1.13	5	1.10	5	1.08	5	1.07
7.5	1.20	7.5	1.15	7.5	1.12	10	1.10
10	1.26	10	1.20	10	1.16	7.5	1.14
12.5	1.33	12.5	1.25	12.5	1.21	12.5	1.17

SF = Swell Factor RW = Room width
OD = Overburden depth VD = Void depth

OD_c = Depth of overburden above which surface heave occurs
VD_c = Depth of void below which surface heave occurs
SF_c = Swell factor above which surface heave occurs

8.4. CONCLUSIONS FROM TEST BLASTING PROGRAM

The test blast program carried out in relation to this research project was very successful. It proved that blasting could definitely be used as a means of collapsing underground openings. In addition, it allowed very useful data to be collected which will be of direct relevance to future work of this kind. This includes the establishment of working practices, blast design information, and an important new insight into AML work.

This section of the report will be concerned with the effectiveness of individual blasts, and how field experimentation led to the optimization of blast design. The relevant technical characteristics of successful blasts are used to make recommendations about future blast design. General comments are also made about working practices, and the suitability of blasting as an AML reclamation method.

8.4.1. Effectiveness of Blasts and Field Experimentation

An attempt is made in Table 8-14 to summarize the overall

TABLE 8-14 : CLASSIFICATION AND MAJOR CHARACTERISTICS OF FIELD TEST BLASTS ACCORDING TO FORMATION OF SURFACE DEPRESSIONS.

Blast #	Location	Hole Φ	# Holes	Length	# Rows	Hole Spacing	Aver. Depth	Void depths	Expl. Types	Delay #'s
Surface depression formed over entire blast length										
3	N-11	6 "	20	135'	2	13 x 8	55	8-11'	AN,IR	4567
5	N-13	6 "	30	200'	2	13 x 8	49	10-12'	AN,IR	4567
14	S-11(N)	6 "	23	160'	2	15 x 8	44	9-12'	AN	4689
17	S10	6 "	26	150'	2	15 x 8	43	10-14'	AN,EN	4689
19	S-9(N)	6 "	20	155'	2	15 x 8	46.5	11-13'	AN,EN	4689
21	S-9(S)	6 "	11	130'	1	12'	46	9-11'	AN,EN	4689
Surface depression formed over most of blast length										
1	N-14	6 "	10	165'	1	17'	46	?	AN	4567
8	N-12	6 "	21	145'	2	15 x 8	52	5-11'	AN,IR	4567
11	S-12	6 "	31	190'	2	15 x 8	41.5	6-12'	AN	5678
12	N-8(N)	6 "	17	130'	2	15 x 8	57	3-9'	AN	5678
13	N-10	6 "	23	155'	2	15 x 8	56	4-10'	AN	4567
16	S-11(S)	6 "	15	115'	2	15 x 8	41	8-13'	AN,EN	4689
Surface depression formed over part of blast length										
9	NC-7	6 "	12	120'	1	12'	45	9-12'	AN,IR	4567
No surface depression formed (except at ends of blast)										
2	N-9	6 "	10	70'	2	13 x 8	58	0-9'	AN	45678
4	N-1,N-2	8 "	22	60'	2	16 x10	64	3-7'	AN,IR	4567
15	S-1(S)	6 "	16	105'	2	13 x 8	55	4-8'	AN,EN	4689
18	N-6	8 "	12	145'	1	15'	57	1-4'	AN,EN	4689
No surface depression formed at all										
6	N-8(S)	6 "	10	55'	2	13 x 8	57	4-8'	AN,IR	4567
7	N-7	6 "	19	70'	2	13 x 8	56	3-8'	AN,IR	4567
10	S-1(N)	6 "	26	135'	2	13 x 8	62	4-8'	AN,IR	45678
Blast used to cave an existing sinkhole										
20	N-4 *	6 "	10	n/a	*	10 x 5	20	N/A	AN,EN	45

performance of each test blast. The 21 blasts have been grouped into categories which attempt to classify the surface expression of blast-induced caving, which is the formation of depressions. There is not necessarily a direct correlation between the formation of a post-blast depression and the success of a blast in collapsing the underground void. This will be discussed in more detail later.

From Table 8-14 it can be seen that 12 blasts resulted in the creation of a surface depression which extended over all or most of the blast length. In the cases where not quite all of the blast exhibited a post-blast depression, the absence of a depression in 4 of these was over a distance which corresponded to only one or two blastholes.

The creation of a depression after blasting is definite evidence that the blast was successful in collapsing the underground void. There are a number of characteristics which these 12 blasts have in common:

- all took place where the overburden cover was less than 60 feet in depth

- in all of the totally successful blasts there was not less than 10 feet of void space below the roof coal

- the blasts which were mostly successful had on average at least 10 feet of available void space over the portions where surface depressions were created. The portions of the blasts where a depression was not created coincided with blastholes where only 3 to 6 feet of void space was detected. This reduced void space was generally located in the panel entries and adjacent intersection areas with rooms where more roof coal had been left during mining to increase the stability of the openings.

- all of the blasts were at least 130 feet in length

- all of the blasts employed 6 inch diameter blastholes

- these blasts showed no obvious correlation with the types and combinations of explosives used

- eight of the 12 blasts employed two rows of blastholes spaced 15 feet apart, on rows 8 feet apart

There were 7 blasts where surface depressions were not created. In one case (N-6) the ground surface was disturbed, but remained at a similar elevation to that prior to blasting. In 6

blasts surface heave took place; in half of these cases a depression was created at one or both ends of the blast. There are a number of characteristics which these blasts have in common:

- all of the blasts were in overburden cover in excess of 55 feet deep

- four of the blasts were less than 100 feet long, and none exceeded 150 feet in length

- the maximum available void space below the roof coal for any of these blasts was 9 feet, and all contained some blastholes in which less than 5 feet of void space was measured

- all of the blasts which resulted in surface heave used 25 ms delay periods between blasthole decks

The first blast was taken using a single row of holes 17 feet apart. Although this was mostly successful, there was a region corresponding to about 5 blastholes where surface craters were formed. The collar heights here were 8.5 feet, which for a 3 foot high upper charge deck corresponded with a scaled depth of burial for the charge of 3.2 ft/(lb)$^{1/3}$. This was changed as a result of the observed cratering, and the collars were increased to 9.5 feet for all subsequent blasts (giving a SDOB of 3.5 ft/(lb)$^{1/3}$ for a 3 foot and 3.3 ft/(lb)$^{1/3}$ for a 4 foot upper deck). It was felt that a spacing of 17 feet was probably too great to obtain adequate overlap of craters in a single row application.

In most cases where a 22 foot wide room was blasted, two rows of blastholes were employed. Having achieved success with 13 foot hole spacings, this was increased to 15 feet, which also proved to be successful. On average a four foot column of charge was used, separated from the next deck by 6 or 6.5 feet of stemming. In general, therefore, the average depth of burial for each spherical charge was about 8 feet. A 17 foot hole spacing is thus more than twice the depth of burial, whereas 13 foot spacing is about one and a half times DOB. The 15 foot spacing which was subsequently found to be optimum for room blasting in this type of material is about 1.85 times the depth of burial.

For 12 foot wide access development a single line of holes using 12 foot centers was initially employed. This did not present firm evidence, in the form of a surface depression, that the blast had successfully broken the greater thicknesses of roof coal associated with the panel entries. In latter blast the hole spacing on panel entries was pulled in to 9 feet, and this proved to be more successful.

Where possible blasts were designed such that they terminated against pre-existing sinkholes, or other blasts. This obviously became easier as the program progressed. Where the ends of a blast were open, it was found that deep sinkholes developed. This is because blasted material moved laterally into the available void space, instead of filling up to the same overall level as the rest of the created depression.

In cases where it was necessary to leave one or both ends of a blast open, use was made of "closure" holes. In blast #4 (S-1,S-2) these were drilled in the pillars adjacent to the panel entries, to a depth close to that of the overburden cover. Decked charges were loaded in the usual manner, though in this case the desired action was to be a sideways movement of material into the blast-created sinkhole. The attempt to avoid deep holes at the ends of this blast was not successful.

This was next tried in blast #9 (NC-7), but in this case shorter holes were drilled, to 25 feet depth, again located just inside the pillars on either side of the panel entry. Two decks of charge were used. It is not possible to assess the success of this attempt as this part of the blast appeared to "bridge" up.

Closure holes were used successfully, however, at the ends of the panel entries associated with rooms S-1 (blast #10) and S-12 (blast #11). These were 30 and 25 feet in depth, respectively, and employed two decks of explosive. The closure holes were placed just in the pillars, about 12 feet from the nearest blasthole located above the panel entry itself. The examples quoted are illustrated in the scale drawings of the blasts in Appendix A.

In blast #19 (S-9(N)) problems created by lack of time and weather conditions meant that closure holes were not drilled at the western end of the panel entry associated with room S-9. The result was that a deep hole was initially created, in excess of 25 feet. This subsequently filled up to about 18 feet by sloughing material from its sides. This would tend to back up the idea that closure holes are necessary in this application, and therefore is an indirect indication that the use of closure holes was generally successful during the test blast program.

The other area in which experimentation was carried out during the field blasting program was the filling of individual sinkholes. This was first attempted in blast #7, where holes were positioned around two small sinkholes to the north and northeast, and one larger sinkhole to the south of room N-7. In each case the holes were located about 3-4 feet from the sinkhole rim and drilled to a depth of 20-25 feet.

The two smaller sinkholes had near vertical sides, and the toe distance was thus not appreciably greater than the burden at surface. These were lightly loaded with a 5 foot cylindrical charges placed near the bottom of 20 foot holes and designed to

break material laterally into the sinkhole. Success was achieved with this design.

In the case of the larger sinkhole to the south, two decked charges were loaded into 25 foot deep holes, which were located 4-5 feet back from the edge of the sinkhole and spaced about 15 feet apart. These were not successful in breaking material into the sinkhole. The toe burden was large, due to the 45-50 degree angle on the sinkhole walls. The 4 foot column of ANFO placed at the hole bottoms did not have sufficient toe-pulling capability to move the large toe burden.

Blast #20 was employed exclusively to fill a vertical -walled sinkhole that formed at the northern end of room N-4. Blastholes spaced about 10 feet apart were drilled about 5 feet back from the rim of this sinkhole. Two decks of explosive were used - a five foot cylindrical charge of slurry at the hole bottom, and a four foot ANFO deck separated by a two foot stemming column. This design proved to be very successful, and indicates that considerable potential exists for the use of blasting to fill sinkholes which, by virtue of their vertical sides, represent a significant danger in abandoned mine land.

8.4.2. Factors Influencing Test Blast Results

In the previous section a number of characteristics which the blasts yielding a certain result had in common were listed. The purpose of this section is to combine the observed results from the test program with that from technical data analysis and make some concluding remarks about those factors which influence the results of AML blasting.

8.4.2.1. Blast Design

Obviously the results of an AML blasting program will only be successful if the correct blast design is applied. Blast design has been described in some detail elsewhere in this report. This section will simply summarize those technical parameters which were found to be successful for the overburden type and field conditions encountered at the Beulah test site. These may be applied to future work of this kind in similar materials.

Below 40 feet of overburden 3 explosive decks were found to be sufficient to collapse the underground openings. The bottom deck was 4 feet in height, with a scaled depth of burial ranging from 2.0-2.3 ft/(lb)$^{1/3}$ for ANFO, and 1.9-2.2 ft/(lb)$^{1/3}$ for a typical slurry product. The length of the upper two decks varied from 3 to 4 feet, with associated scaled depths of burial of 2.0-2.3 ft/(lb)$^{1/3}$ for ANFO.

Between 41 and 59 feet of overburden, 4 decks were used.

For holes of depth 41-51 feet, a 4 foot charge column was used in the bottom of the hole, corresponding to a scaled depth of burial ranging from 2.0-2.3 ft/(lb)$^{1/3}$ for ANFO, and 1.9-2.2 ft/(lb)$^{1/3}$ for slurry. Between 52 and 59 feet the length of the lower deck was increased from 4.5 to 6 feet over this depth range to compensate for greater depth of burial. Scaled depths of burial in this case were in the order of 2.4 ft/(lb)$^{1/3}$ for ANFO, and 2.3 ft/(lb)$^{1/3}$ for a typical slurry product. Typical scaled depths of burial for the other 3 decks in this range of overburden were in the range of 2.2-2.5 ft/(lb)$^{1/3}$ for ANFO.

Above 60 feet of overburden cover 5 decks were required, and the use of a higher density explosive than ANFO is recommended for the bottom deck, which should be of the slurry type if water is present. Between 5 and 7.5 feet of charge was employed in holes of this depth range, with a scaled depth of burial in the order of 2.0 ft/(lb)$^{1/3}$ for the lower density Energel product, reduced to about 1.9 ft/(lb)$^{1/3}$ for Iregel. Again typical scaled depths of burial for the other decks in this overburden depth range were 2.2-2.5 ft/(lb)$^{1/3}$ for ANFO.

Less testwork was carried out using 8 inch blastholes. Results for blast #18 (see Appendix D) show that scaled depths of burial were in fact somewhat lower than for 6 inch holes, at 1.7-2.0 ft/(lb)$^{1/3}$ when using an explosive with the density of ANFO. This was largely in response to the less than ideal results from the first blast, #4, which employed larger diameter holes, where the scaled depths of burial were similar to the 6 inch diameter case. There are, as will be seen shortly, other factors which affected the performance of this particular blast.

More testwork would be required where large diameter blastholes were employed before conclusions and recommendations can be made regarding technical design parameters. A general conclusion relating to the use of 8 inch diameter holes is that they will probably be required for overburden depths in excess of 65 feet, simply to get enough charge weight into the column to provide acceptable scaled depths of burial.

As noted in the previous section, a collar height corresponding to scaled depths of about 3.3-3.5 ft/(lb)$^{1/3}$ yielded acceptable results, which was 9.5 feet in 6 inch and 13 feet in 8 inch diameter blastholes.

With regard to hole spacing and the number of rows to use, it may be concluded that for narrow rooms or panel entries a single row of blastholes is sufficient. A hole spacing of 1.5 to 1.8 times the typical depth of burial for a decked charge yielded good results in rooms 22 feet wide. In narrower development such as panel entries where one expects more restricted space for blast movement a spacing of about 1.1 times depth of burial is recommended.

There was evidence from the test blast program that a

single line of 8 inch holes could be successfully employed in a 20 foot wide room. The experimentation carried out on room S-9 (blasts #19 and 21) suggests that a single line of 6 inch holes spaced at 12 feet (1.5 times typical depths of burial) can be effective. From a cost point of view, as will be seen later, this is a very important design consideration.

It is likely, though, that the craters formed by explosive decks in a single row of holes will not be of sufficient radius to break all the way to the pillars. This may result in a narrower, and deeper depression than if two rows are used. As a rough guide, if the wall-to-wall distance for a room is greater than 1.75 times the typical depth of burial for a decked charge, then two rows of blastholes should be employed. For the test site conditions this would correspond to a room width of around 14-15 feet, and explains why two rows were used in most cases.

The only direct evidence from field testwork results that indicated that blast design had been less than totally successful came from post-blast exploration drilling which showed that portions of two blasts (#4, (S-1 and S-2) and #9 (NC-7)) had "bridged" at the position of the upper two decks. While there was evidence that the presence of rock layers, and possible overlap of number 6 and 7 down-the-hole delays could have been the cause of this, it may be that a modification in the blast design could be beneficial.

As seen earlier, the upper explosive deck has a cratering action up to surface (SDOB of about 3.5 $ft/(lb)^{1/3}$), and another down to the deck below (SDOB of about 2.3 $ft/(lb)^{1/3}$). During the detonation of the top deck it is possible, however, that the explosive action felt a greater relief upwards to the "free face" provided by the ground surface than that available below. This would be especially true if there were rock layers in the region associated with the stemming column between the top two decks.

One possible design modification would be to increase the length of the charge column in the uppermost deck to about 12 charge diameters (6 feet in a 6-inch blasthole) which would be classified as a cylindrical charge. However, in the absence of a free face for lateral movement this could behave as two adjacent spherical charges. The collar should be scaled to a depth of 3.5 $ft/(lb)^{1/3}$, based on a spherical charge center located 4 charge diameters down from the top of the column. The stemming interface should then be calculated using a scaled depth of 2.3 $ft/(lb)^{1/3}$, with charge center located 4 charge diameters up from the bottom of the uppermost charge column.

8.4.2.2. Material Swell, Void Space and Overburden Depth

A detailed technical analysis using a model to explain the

relationship between overburden depth, material swell and available void depth in a room was presented earlier in this chapter. It was shown that for a 22 foot wide room in 50 feet of overburden, with a swell factor of 12% one might not expect the formation of a depression if void depths of less than 6.5 feet were available.

When considering the second set of blasts as categorized in Table 8-14 (those where surface depressions formed over almost all of the blast length) it can be seen that in three of these cases surface heave of material over parts of the blasts can be explained by the above relationship. In the case of blast #11 (S-12), in lower cover, the surface heave occurred at the position of the panel entry where there was restricted material movement. In blast #16 (S-11(S) the lack of subsidence was probably due to the fact that available void space into which material could fall had been filled by material which spilled laterally from the southern end of the previous blast.

A second conclusion from the blast-induced subsidence model was that in overburden depths of 55-65 feet one required 7-8 feet of void space to create a surface depression. It was seen that all 7 blasts where surface heave occurred had overburden depths of 55 feet or greater, and that the maximum void depth encountered was 9 feet. Over much of these blasts void depths were five feet or less. Post-blast exploration drilling adjacent to room S-1 (blasts #10 and #15) indicated that the rooms appeared to have been filled up to the level of the roof coal.

It should be clear, therefore, that the creation of surface heave is by no means an indication that a blast has not been successful in collapsing and filling an underground void. Blast #18 (N-6) resulted in no change of surface elevation, though the very limited void depth indicates that surface heave should in fact have occurred in this case. It is likely, therefore, that the single row of holes employed did not break the roof coal over its entire width between the pillars. Some material was able to migrate laterally into the room - which was subsequently filled and thus presents no potential hazard from future caving.

The only blast where some surface heave occurred which cannot be explained directly by a lack of adequate void space is #9 (NC-7). Here, as previously described, there was other evidence why this had occurred. Another interesting point which ties in the idea of available void space was the very large hole created at the intersection of the cross-cut and panel entry in this blast. Probably as a result of a pillar failure at this location, a large void volume was created, which resulted in a much wider and deeper sinkhole on surface than would be expected for 12 foot wide access development.

8.4.2.3. Length of Blast

It was observed from Table 8-14 that 4 out of the 7 blasts where surface heave resulted were shorter than 100 feet. It was thought earlier on in the study that this was a direct contributing factor to the success of a blast, and that the collapsing action was restricted in shorter blasts due to there being less overall distance over which the mechanism could develop. This mechanism was likened to the "domino effect" - if there was not enough room for the first domino to fall, there would be no collapse of the structure.

However, blast #16 (S-11(S)) was fairly short, but was definitely successful. In all of the shorter blasts there was limited void space which itself explains the surface heave. Indeed, a blast was often short because it was terminated against a pre-existing sinkhole. Creation of such a sinkhole was almost certainly the reason for the limited void space, due to the lateral migration of material into the un-collapsed part of the room.

There is no apparent reason, therefore, based on results from the test blasting program, why short blasts should be any less successful than longer ones. There will be more "wastage" in that void space available to one blast will be partly reduced by lateral filling from the previous and adjacent blast. There is a distinct cost advantage of maximizing blast length, as will be seen in the next chapter, during any AML reclamation project using blasting.

8.4.2.4. Millisecond Delays

Millisecond delays were employed on surface between boreholes, and between decks in the hole. There were two reasons for the use of millisecond delays. Firstly, it is desirable from the point of view of minimizing ground vibration levels that low charge weights are detonated for a given delay interval. It is obvious from results presented in Chapter 7 that this was achieved, and that the use of surface and down-the-hole delays was justified.

In addition, use of down-the-hole delays was required in order that each blasthole functioned as a series of cratering charges, blasting to the free face created by the charge below. The evidence presented in previous sections of this Chapter for the successful collapse and filling of underground development indicates that correct use of down-the-hole millisecond delays was made.

Initially a 25 millisecond delay period was used between successive explosive decks. From blast #14 (S-11(N)) onwards this period was increased to 50 milliseconds, with the arrival of the 250 ms period number 9 delay. The technical rationale

behind this decision was explained in Chapter 4 of this report. The 50 ms period between decks gave more time for material movement, and increased the chances that there was a free face for material to blast into.

It was noticed from high-speed films that the direction of the caving action immediately after the blast changed also with the use of 50 milliseconds between decks. When using 25 ms delay intervals the caving action tended to start at the end of the blast, and work back towards the start. When this was changed to 50 ms the caving action appeared to follow behind the blast from its start to its finish. This is a strong indication that the greater delay interval was beneficial, as it brought on the caving/collapsing action more quickly, thus reducing the possibility of "hang-ups" in the blast.

8.4.3. Technical and Design Implications from Test Blast Results

8.4.3.1. Effectiveness of Working Practices

The test blast program at the Beulah AML site indicated convincingly that blasting could be safely and successfully used as an AML reclamation method. A mobile rotary drill using compressed air was found to be practical for drilling work. Mobility within a site where there were pre-existing sinkholes was found to be possible, and the rig could be moved quickly and easily around the site. Drilling set-up time was minimal. The contracting of a drilling crew which has had previous experience in AML sites is a very distinct advantage.

Use of pickups for access and explosives transport was found to be feasible, even under difficult and deteriorating weather conditions. Safe and efficient working practices were developed for all of the "unit operations" associated with AML blasting work, which were described in Chapter 5.

The use of explosives was found to be effective - slurry type explosives, with their higher density and waterproof properties, were effective in the lowermost deck. Use of the Nonel system for surface trunkline and down-the-hole delays was found to be advantageous, both from the point of view of noise levels (which were minimal) and for ease of use.

A 300 foot long blasting cable was sufficient for safe initiation of blasts - collar heights were selected correctly and fly-rock was minimized. The site was in a remote location, but sufficient personnel were available to man road-blocks and ensure that safe procedures were adopted at blast time.

A seismograph for airblast and ground vibration measurement should be considered as essential in work of this type. Results

from seismograph measurements taken indicated that work of this type, carried out under the conditions described, should not present vibration levels of any concern at distances of greater than 400 feet from an AML blast. The use of a high-speed camera is not essential, but good practice if such an instrument is available.

It was found that blasting activity did not cause the collapse of adjacent or nearby sinkholes, even where it was subsequently determined that these had caved by natural processes to within 10 feet of the surface. It was found that a room could be blasted in two portions without significant loss of efficiency. However, blastholes located in or adjacent to the room but not blasted could be lost, and it is not therefore recommended that holes be drilled too soon before they are required.

8.4.3.2. Recommended Design Philosophy

A blast design philosophy whereby cratering charges are blasted successively into the underground opening from the bottom of the hole upwards is recommended. To minimize ground vibration levels and to ensure that there is sufficient time for collapse to occur, holes should be separated by surface delays.

In loosely consolidated materials such as glacial till, clays and soft sandstones collar heights should be calculated according to a scaling factor of 3.3-3.5 $ft/(lb)^{1/3}$. Scaled depths of burial of between 2.0 and 2.5 $ft/(lb)^{1/3}$ should be employed in such materials to design the charge and stemming heights and placement. Use of a charge column of length 12 charge diameters is recommended for the uppermost deck, which should act as two adjacent spherical charges.

A blasthole spacing on surface of 1.5 to 1.85 times the typical depth of burial for a charge should be employed in development 15 feet or wider. Below this, due to confinement, it may be appropriate to reduce this spacing to around 1.1-1.2 times the typical depths of burial.

When the wall-to-wall distance for the development exceeds 1.75 times the typical depth of burial it is recommended that two rows of blastholes be employed. These rows should be separated such that there is approximately the same distance between rows as there is distance between a row and the pillar wall. It is necessary to stagger the hole placement on rows to optimize the distribution of explosives and minimize the drilled footage.

Standardization of loading instructions for holes of a given diameter and depth is essential for efficient field operations. Modifications should be made where necessary to take into account factors such as thickness of roof coal and the

presence of hard rock bands. If water is detected in blastholes it is essential to employ a slurry type explosive.

The use of column charges to cause the collapse of underground openings from surface is not recommended, nor was it attempted during the project fieldwork. Reasons for the use of spherical crater charges, analogous to those used in the VCR method of underground mining, were presented in Chapter 2. In conventional pit bench blasting one or more vertical free faces are present to the side of a blasthole. Therefore, it is often possible to widen blasthole spacings by making more explosive energy available in a hole. This can be achieved by use of larger diameter blastholes, or of a higher energy explosive, or by fully loading the charge column, as opposed to deck loading.

In AML blasting, though, the only available free-face for blasting is downward, and below the hole. The orientation of the free face is not suitable for bench blasting techniques. It is very unlikely, therefore, that the extra explosive quantity in a column charge would do any useful work. An increase in the blasthole spacing would probably result in incomplete caving of the room. The extra explosive consumed in a column charge would be wasted, and would certainly increase the cost of AML blasting.

9. COST ANALYSIS

9.1. INTRODUCTION

The analysis of the costs associated with the use of blasting as an AML reclamation tool is a very important aspect of the overall feasibility of the method. Even though it has been proven that the method is technically feasible, it is necessary to assess its economic viability.

This overall aims of this section of the report are as follows:

- to report the costs associated with the fieldwork for this project
- to analyze these costs and their significance
- to identify key cost centers
- the use of actual costs as a basis for the establishment of a "Cost Model" for AML blasting work
- use of the Cost Model to project costs for future work of this kind
- to compare the cost of blasting with other AML reclamation methods

Care was taken during the analysis to separate those costs which had an essentially "research component" from those which would be incurred in the actual use of the method, once proven and refined, for AML reclamation work.

The cost of initial site selection and evaluation is not included in this analysis, nor is the exploration drilling which was carried out to define the overall site characteristics. Mention of the project costs for this, and other potential AML sites, is made in Chapter 3 of this report. The cost of exploratory "line-up" drilling is incorporated into this analysis, however.

Extensive use for this analysis has been made of a "spreadsheet" program, using a microcomputer. The actual software employed was the Lotus 1-2-3 package. Use of the graphics capability of this program has also been employed to produce many of the figures associated with the Cost Model.

9.2. ANALYSIS OF ACTUAL FIELDWORK COSTS

9.2.1. Unit Costs Employed

Table 9-1, which actually forms part of the spreadsheet used for this analysis, shows the breakdown of unit costs into cost centers, and the actual values that were incurred during the 1986 testwork.

Drilling was carried out at 6 and 8 inch diameters, and the costs incurred for both of these are contained in the spreadsheet. Explosives were ordered in two separate batches; the first was larger, and therefore lower unit costs for ANFO and slurry were incurred. The cost of explosives accessories, and labor charges, remained constant throughout the project.

The general costs of travel, subsistence, rentals and miscellaneous field items were totalled for the project, and then assigned a "per blast" cost simply by dividing the total item cost by 21, the number of blasts taken. Although this is something of an oversimplification, this approach is felt justified as the general cost associated with a particular blast is small when compared with the drilling, blasting and labor costs.

As the actual costs incurred are to be used as the basis of a Cost Model to predict cost of future work of this kind (see Section 9.3. of this report) it was necessary to separate those costs which had a distinctly "research oriented" component, namely those associated with targeting explosives with Nonel trunkline, and high-speed camera work.

9.2.2. Calculation of Blasting Cost Components

The second page of the blast costing spreadsheet calculates the actual cost of the major components of each blast, and the total cost associated with it. These are contained in Appendix F for the 21 blasts carried out in the fieldwork. Table 9-2, which contains the analysis of the cost of blast number 13, will be used to illustrate the following description of how costs were actually calculated.

Each blast is identified by its number and location, and the number and diameter of blastholes is recorded, together with the number of rows of blastholes employed. The length of the blast is multiplied by the affected width (in this case the width of the room plus half of the distance in the pillar to the next room) to obtain an "area of influence", which is used to

AML PROJECT - BLAST COST ANALYSIS.

		COSTS USED	
		8 INCH	6 INCH
DRILLING COST	/FT	$0.900	$0.750
EXPLOSIVES COSTS		BATCH 2	BATCH 1
ANFO	/LB	$0.212	$0.161
SLURRY	/LB	$0.451	$0.386
BOOSTERS	EACH		$3.120
DTH DELAYS	EACH		$1.680
SURF. DELAYS	EACH		$2.200
HOLE PLUGS	EACH		$1.820
CAPS	EACH		$1.976
PRIMALINE	/FT		$0.079
NONEL T'LINE	/FT		$0.057
BALER TWINE	/FT		$0.003
LABOR COSTS			
BLAST. ENGR.	/HR		$27.00
FIELD ENGR.	/HR		$18.50
LABORER	/HR		$10.00
GENERAL COSTS			
TRAVEL (PICKUP RENTAL, MILEAGE, GAS)			
TOTAL FOR PROJECT			$2,566.00
PER BLAST			$122.19
SUBSISTENCE			
TOTAL FOR PROJECT			$831.00
PER BLAST			$39.57
RENTALS			
SEISMOGRAPH			$1,120.00
MAGAZINES			$665.00
TRAILER			$621.00
TOTAL FOR PROJECT			$2,406.00
PER BLAST			$114.57
MISCELLANEOUS			
MISC. FIELD EQUIPMENT			$419.00
GRAVEL (STEMMING)			$263.00
SNOW REMOVAL, BOBCAT HIRE			$340.00
WOODEN STAKES			$275.00
TOTAL FOR PROJECT			$1,297.00
PER BLAST			$61.76
STAKES, TARGETS ETC. (RESEARCH COMPONENT)			$380.00
PER BLAST			$27.14

TABLE 9-1: COST PARAMETERS AND VALUES USED FOR CALCULATION OF INDIVIDUAL BLASTING COSTS

AML PROJECT - BLAST COST ANALYSIS. BLAST NO. 13

LOCATION :	N-10	# HOLES	23	DIAM.	6
LENGTH	155 FT	WIDTH AFFECTED :	40 FT		
NO.OF ROWS	2	AREA OF INFLUENCE	6200 SQ. FT		

						COST
DRILLING						
	ROOM LINE-UP	180 FT @	$ 0.750 /FT	=		$135.00
	PRODUCTION	1347 FT @	$ 0.750 /FT	=		$1,010.25
	TOTAL					$1,145.25
BLASTING						
	ANFO	5200 LBS @	$ 0.175 /LB	=		$910.00
	SLURRY	0 LBS @	$ 0.386 /LB	=		$0.00
	BOOSTERS	92 @	3.120 EACH	=		$287.04
	DTH DELAYS	92 @	1.680 EACH	=		$154.56
	42 MS DELAYS	23 @	2.200 EACH	=		$50.60
	HOLE PLUGS	25 @	1.820 EACH	=		$45.50
	CAPS	1 @	1.976 EACH	=		$1.98
	PRIMALINE	1420 FT @	$ 0.079 /FT	=		$111.47
	BALER TWINE	2840 FT @	$ 0.003 /FT	=		$7.10
	TOTAL					$1,568.25
	NONEL T'LINE (RESEARCH COMPONENT)	1400 FT @	$ 0.057 /FT	=		$79.80
LABOR						
	LINEUP	1.75 HRS @	$ 18.50 /HR	=		$32.38
	PREPARATION	3.50 HRS @	$ 28.50 /HR	=		$99.75
	BLASTING	10.00 HRS @	$ 55.50 /HR	=		$555.00
	TOTAL					$687.13
	OVERHEADS	50 % DIR LAB		=		$343.56
	TOTAL					$1,030.69
	RESEARCH COMPONENT	2.00 HRS @	$ 55.50 /HR	=		$111.00
	TOTAL (+ OVERHEADS)					$166.50
GENERAL						
	TRAVEL	/BLAST		=		$122.19
	SUBSISTENCE	/BLAST		=		$39.57
	RENTALS	/BLAST		=		$114.57
	MISC.	/BLAST		=		$61.76
	TOTAL	/BLAST				$338.10
	RESEARCH COMPONENT	/BLAST		=		$27.14
TOTAL COST (EXCL. RESEARCH)				=		$4,082.28
TOTAL COST (INCL. RESEARCH)				=		$4,355.72
COST/ACRE (BASED ON THIS BLAST, EXCL. RESEARCH)						$28,681.30

TABLE 9-2: EXAMPLE OF SPREADSHEET USED TO CALCULATE COST OF INDIVIDUAL BLASTS

pro-rate the cost of the blast to a cost per acre (explained later).

The room line-up drilling footage was estimated for each blast; this does not include the exploration drilling at the site to locate rooms themselves, but is the drilling carried out to locate the lateral limits of the underground openings in order that blastholes could be properly positioned. The production drilling footage is the actual amount of drilling carried out during field operations for that particular blast.

For blasting cost calculation, all consumption of explosives and explosives accessories are those recorded from magazine inventory sheets for that day. Primaline and baler twine footages are estimated, based on the known drilling footage for a blast. Nonel trunkline, where used for targets in high-speed camera work, is recorded as a separate cost as this represented a purely "research component" of the fieldwork.

A uniform approach was taken in assigning labor charges. It was assumed that on average each blast involved about 1.75 hours work by the field engineer during the room line-up stage. In addition, about 3.5 hours work was involved, by the field engineer and one helper, in preparation for the blast. This includes the location and staking out of the blasthole pattern, and the measurement and plugging of the blastholes prior to actual blasting operations.

It is estimated that a blasting operation took, on average, about 10 hours with the crew size that was available during the 1986 testwork. This included all blasting "unit operations", including loading powder from the magazine, making up the boosters with the appropriate down-the-hole delays, and actual charging, measuring and stemming operations. It also included the location and setup of the engineering seismograph, the tie-in of the blast using surface delays, the wiring-up, and the blast itself, followed by a visual check for misfires and a brief assessment of its success.

Again, the "research element" of labor cost was kept separate - it was estimated that for a typical blast where the high-speed camera was used, a further two hours work were involved over and above normal blasting operations. This includes the time spent in attaching Nonel trunkline to the appropriate delay, the fixing of the coiled Nonel to the targets, and the set-up, film loading, testing and actual operation of the high-speed camera.

The general costs associated with the blasting fieldwork constituted only about 10% of the total expenditure. They included travel, subsistence, rentals and miscellaneous field equipment as shown in Table 9-2. This cost was divided equally between the 21 blasts for the purpose of this analysis. Had this site been more remote from the area of operations of the

participating companies, these costs would have been more significant. Then, it would be necessary to pro-rate them on some other basis, such as the blast area of influence.

9.2.3. Calculation of Total Blasting Cost

The costs of each blast component were totalled in the spreadsheet to give a total cost for the blast, inclusive and exclusive of the research component (see Table 9-2). Data for each individual blast is contained in Appendix F. This is also summarized, together with other pertinent data, in Table 9-3. It can be seen from this table that the total cost of a blast taken to collapse an underground opening during the 1986 testwork varied from $2300 to $5575. Blast number 20 was employed only to fill a sinkhole, with a correspondingly lower cost.

Total and component costs for the 21 blasts are illustrated graphically in Fig. 9-1, where it can be seen that the labor and general costs associated with each blast are assumed to be constant for all cases except blast # 20. These are exclusive of the "research" cost element. In almost every case, explosives represented the largest cost element.

9.2.4. Calculation of "Pro-rated" Cost per Acre

Once the actual cost of a blast is calculated, it is possible to "pro-rate" this to a cost per acre, assuming exactly the same site conditions over that acre, according to the following expression:

$$\text{Cost per acre} = \frac{\text{Cost of blast} \times 43560}{\text{Area of influence (sq.ft.)}}$$

Thus a blast is pro-rated according to the number of times its area of influence fits into an acre. Since the area of influence includes the pillars, this pro-rated cost will apply only for an acre of land in which the exact same mining configuration occurred.

The pro-rated cost per acre for each of the 21 blasts forms the bottom line of each cost analysis spreadsheet (see Table 9-2) and is exclusive of any research component. These numbers are also summarized in Table 9-3, and shown graphically in Fig. 9-2.

It can be seen that, based on the actual costs incurred, that a cost per acre of between $15,000 and $53,000 was experienced. Further discussion of these results will be made in the next section.

AML PROJECT - BLAST COST ANALYSIS - COST DATA SUMMARY

BLAST #	LOCATION	# HOLES	AVER. DEPTH	# ROWS	AREA OF INFL.	NORMAL BLASTING OPERATIONS DRILLING	EXPLOSIVE	LABOR	GENERAL	TOTAL	COST/ACRE
1	N-14	10	46	1	6800	$502.50	$495.86	$1,030.69	$338.10	$2,367.15	$15,163.68
2	N-9	10	58	2	2800	$598.50	$613.67	$1,030.69	$338.10	$2,580.96	$40,152.36
3	N-11	20	55	2	5400	$1,271.25	$1,433.00	$1,030.69	$338.10	$4,073.04	$32,855.86
4	N-1,N-2	22	64	2	6500	$1,602.00	$2,604.25	$1,030.69	$338.10	$5,575.04	$37,361.34
5	N-13	30	49	2	8000	$1,275.75	$2,016.03	$1,030.69	$338.10	$4,660.57	$25,376.80
6	N-8(S)	10	57	2	2200	$565.50	$778.23	$1,030.69	$338.10	$2,712.52	$53,707.90
7	N-7	19	56	2	4800	$622.50	$787.99	$1,030.69	$338.10	$2,779.28	$25,221.97
8	N-12	21	52	2	6000	$990.75	$1,446.50	$1,030.69	$338.10	$3,806.04	$27,631.85
9	NC-7	12	45	1	5600	$507.00	$732.28	$1,030.69	$338.10	$2,608.07	$20,287.06
10	S-1(N)	26	62	2	5400	$1,290.00	$2,090.70	$1,030.69	$338.10	$4,749.49	$38,312.55
11	S-12	31	41.5	2	7800	$1,046.25	$1,283.84	$1,030.69	$338.10	$3,698.88	$20,656.82
12	N-8(N)	17	57	2	5200	$933.75	$997.66	$1,030.69	$338.10	$3,300.20	$27,645.52
13	N-10	23	56	2	6200	$1,145.25	$1,568.25	$1,030.69	$338.10	$4,082.29	$28,681.38
14	S-11(N)	23	44	2	6400	$966.75	$1,348.41	$1,030.69	$338.10	$3,683.95	$25,073.88
15	S-1(S)	16	55	2	4800	$918.00	$1,583.10	$1,030.69	$338.10	$3,869.89	$35,119.25
16	S-11(S)	15	41	2	4600	$606.75	$1,001.60	$1,030.69	$338.10	$2,977.14	$28,192.22
17	S-10,SC-7	26	43	2,1	10800	$911.25	$1,646.40	$1,030.69	$338.10	$3,926.44	$15,836.64
18	N-6	12	57	1	6000	$1,300.50	$1,565.07	$1,030.69	$338.10	$4,234.36	$30,741.45
19	S-9(N)	20	46.5	2	6200	$840.75	$1,569.34	$1,030.69	$338.10	$3,778.88	$26,549.68
20	N-4	10	20	1*	1963.5	$150.00	$419.62	$342.00	$338.10	$1,249.72	$27,724.88
21	S-9(S)	11	46	1	5200	$489.75	$864.99	$1,030.69	$338.10	$2,723.53	$22,814.80
21	-	384			118664	$18,535	$26,847	$20,956	$7,100	$73,437	$26,958.04

TABLE 9-3: SUMMARY OF COSTS OF INDIVIDUAL BLASTS CARRIED OUT DURING TEST BLAST PROGRAM

FIG. 9-1 : BLASTING COST SUMMARY

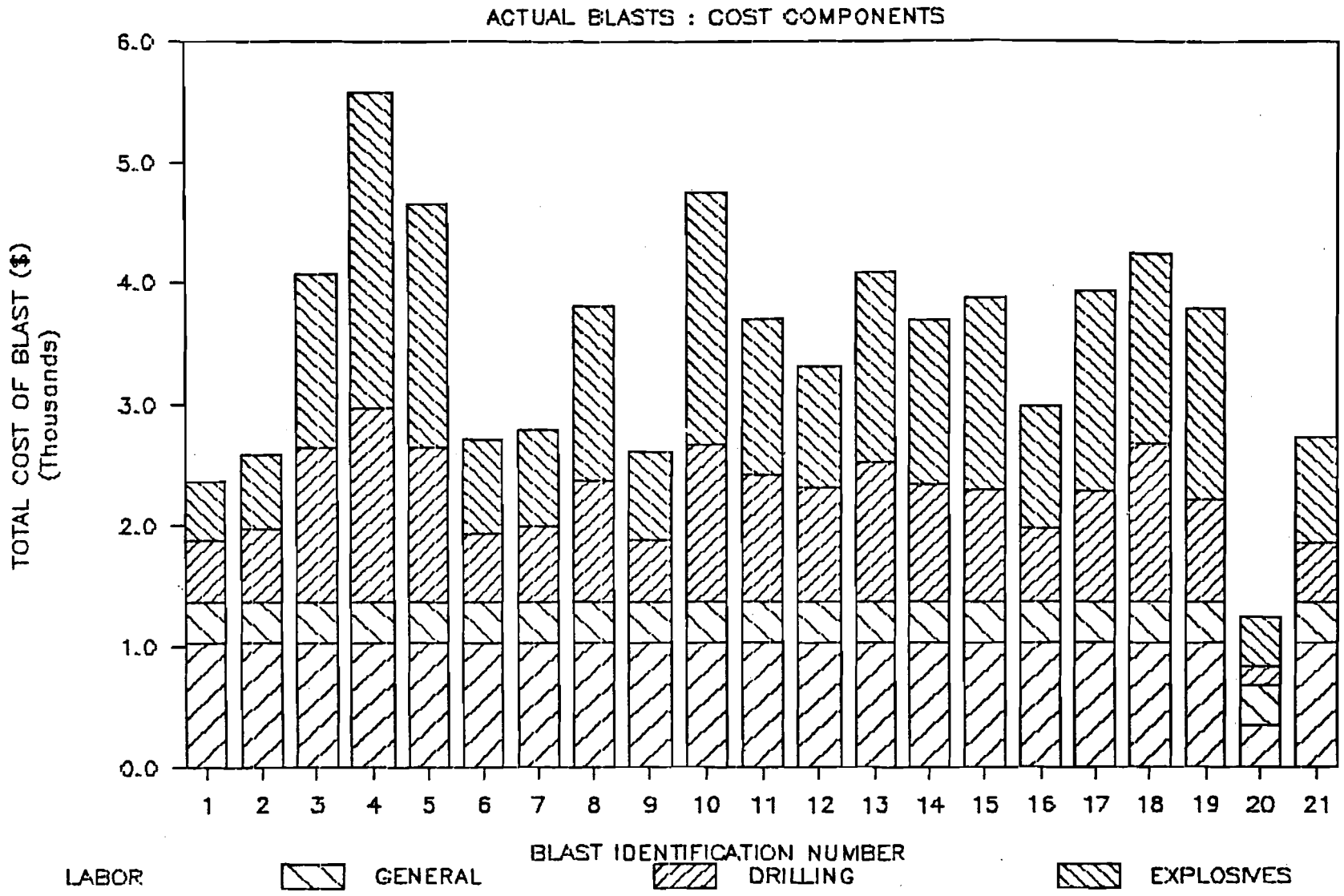
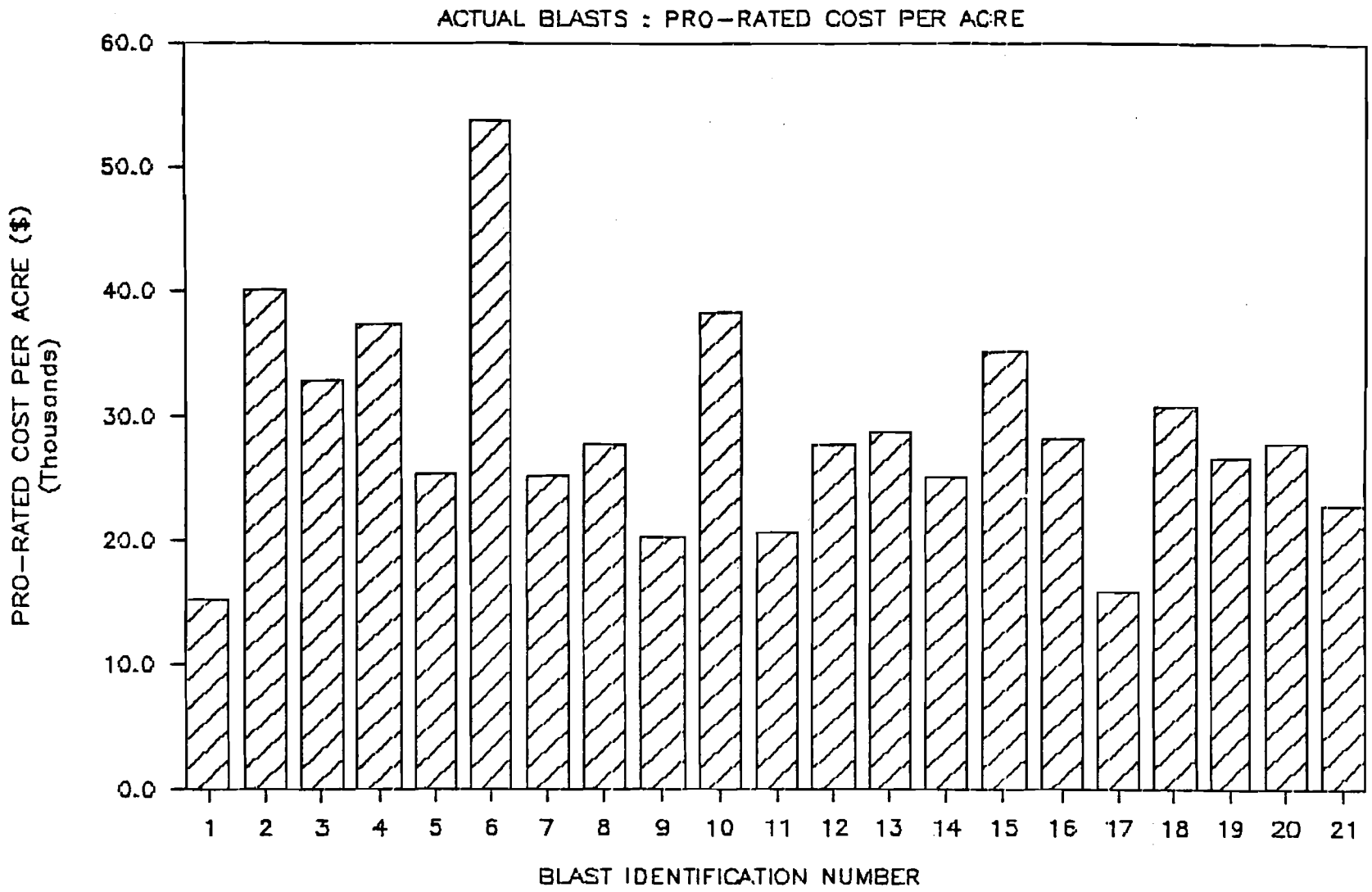


FIG. 9-2 : BLASTING COST SUMMARY



9.2.5. Cost Analysis of Actual Blasts

Variation in actual blasting cost is obviously very dependent on the size of the blast taken, with longer blasts requiring more blastholes, and therefore more drilling footage, and using a greater total weight of explosives. It is thus of little relevance to compare the actual blast cost data from Table 9-3 or Fig. 9-1; discussion will therefore be focussed on the pro-rated cost per acre of each blast.

The particular value of this part of the analysis is in the assessment of those unit costs to which total cost per acre is most sensitive, so that these parameters may be studied using the Cost Model (see Section 9.4. of this report).

It is apparent that blasting cost per acre is dependent on some of the following factors:

9.2.5.1. - Overburden Depth

Deeper overburden involves more drilling and higher explosives consumption. Fig. 9-3 illustrates the average depth to void for the 21 blasts carried out. To a large extent, the high pro-rated costs per acre in blasts 10 and 15, for room S-1, are due to the deep overburden cover in this area. The same may be said for blast number 18 on room N-6.

9.2.5.2. - Blasthole Diameter

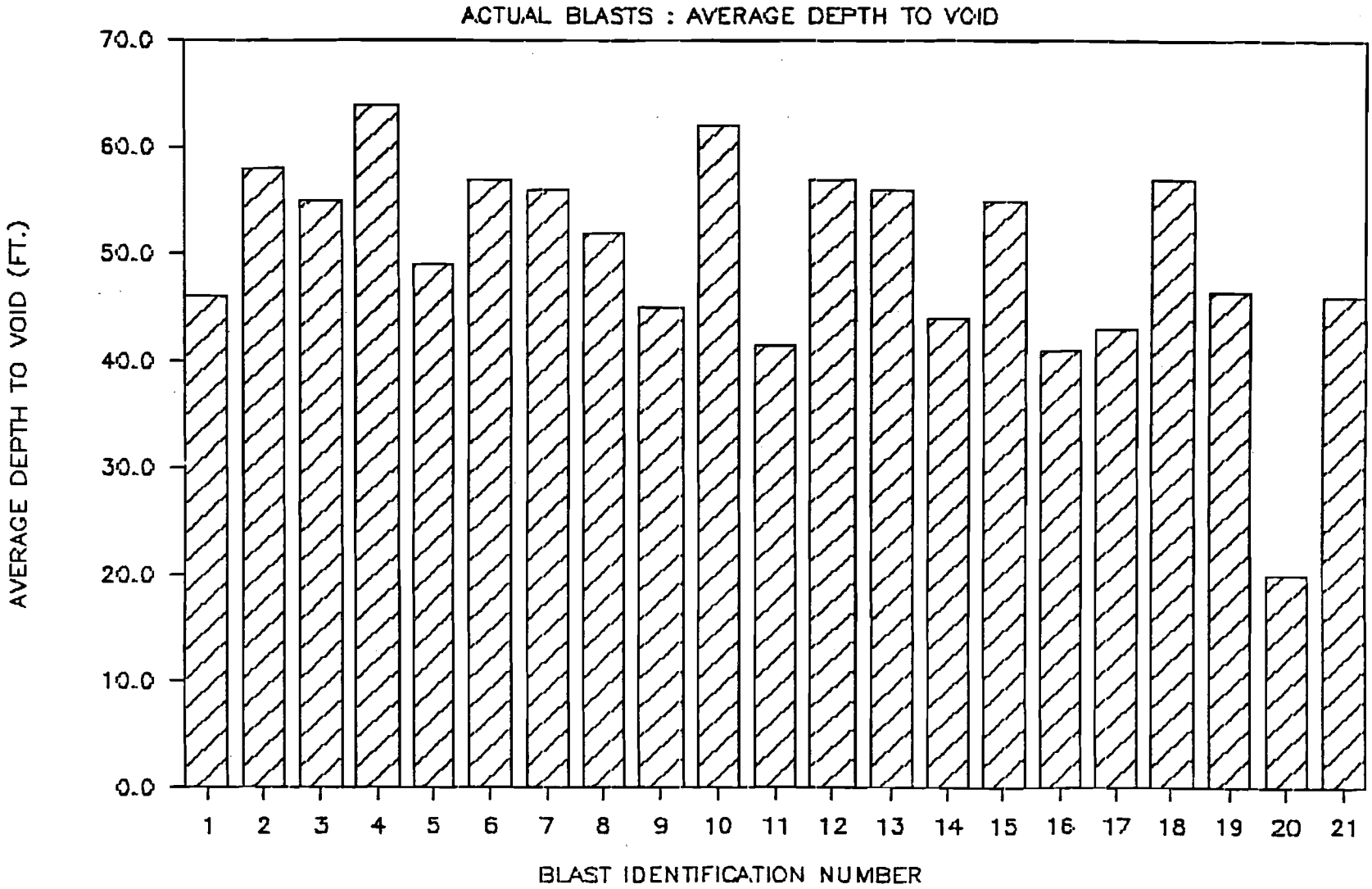
Higher explosives costs may result from the use of 8 inch, as opposed to 6 inch blastholes. An 8 inch blasthole contains 78% more explosive volume per foot than a six inch. This is offset to some extent by the fact that it is often possible to space the 8 inch blastholes further apart, and use a single line of blastholes, thus reducing the number of holes and the total drilling footage.

This was the case for blast number 18 (N-6), where the single line of blastholes used exactly halved the drilling footage that would have been incurred using a double line of 6 inch holes. The fact that the pro-rated cost per acre is higher than average for this blast is due more to high overburden depth than the fact larger holes were used

9.2.5.3. - Number of Rows of Blastholes

It is obviously considerably less costly to blast a room of a given width with a single line of blastholes than with a double row. Though the hole spacing may have to be reduced in a single row case for the same blasthole diameter, the footage of blasthole to be drilled and charged will be almost halved. However, it must be assured that a single row will successfully collapse a wider room.

FIG. 9-3 : BLASTING COST SUMMARY



The first blast taken during the testwork, number 1 on room N-14, had one of the lowest pro-rated costs per acre because a single row of holes was employed in a 22 foot wide room. Blast number 9 also used a single row, and the hole diameter and average overburden depths were essentially the same. The pro-rated cost per acre was higher, though, for reasons which will shortly be discussed.

Blasts 19 and 21 were taken in adjacent parts of room S-9, using double and single rows of blastholes respectively. The site conditions were, therefore, virtually identical. Blast 19 was a little longer, and had, in fact, a 20% greater area of influence; this had no effect on the pro-rated cost per acre, however. Blasthole spacing for the single row blast was reduced by 3 feet relative to the double row blast.

When the correction is applied for the different areas of influence, it is found that the drilling and blasting costs for the double row case were 44% and 52% higher, respectively, than for single row blasting. The cost per acre associated with the former was, at $26,549, about 16% higher than for the single row case. This reduction does not seem very high, in view of the major savings in explosives for the single row case. However, the explanation lies in the fact that the same labor charges were applied to both blasts.

In a non-research application, however, this would not be the case. Single row blasting involves less time for a given blast size, and it would therefore be possible to schedule longer blasts than for a double-row case. Since labor costs are also pro-rated by the blast area of influence, they would be lower in a longer, single row blast and therefore the total pro-rated cost per acre would be reduced more than is the case in this example. This is a significant factor when considering the development of a Cost Model, and will be discussed further later on.

9.2.5.4. - Explosive Type

If an area can be blasted using ANFO alone, then the explosives cost will be appreciably lower than if the higher cost slurry type is required. This must be weighed against the increased bulk strength of the higher density slurry explosives, and their superior performance in wet conditions.

The effect of explosive type on overall cost can perhaps be best illustrated by comparing blast numbers 12 and 15 (N-8(N)and S-1(S) respectively), which were both in about 55 feet of overburden cover. The total cost of blast number 15, where two decks of slurry explosive were employed due to wet hole conditions, was about 27% higher than that for blast 12, where all ANFO was used. This is almost entirely due to the difference in cost between the two explosive types - both blasts used about

3200 pounds of explosives. The explosives cost for the blast using slurry was 78% higher than the case for all ANFO, when correction is made for the different blast areas of influence.

9.2.5.5. - Area of Influence

It has already become apparent from previous discussion that this is a very important factor in determining the cost per acre for a blast. The pro-rated costs for blasts 2 and 6 from the testwork are very high because their associated areas of influence were low. The blasts were short, and involved the same labor cost as larger blasts. Such costs would not normally be encountered in AML blasting of a non-research nature, where efforts would be made to maximize the length of room blasted in a shift.

The area of influence that a blast has in an AML property is very closely linked with the relative widths of rooms and pillars, and the type of development, at the site. The concept of "percent extraction" will be introduced here. In many coal deposits where room and pillar mining was employed, the three dimensional concept of mining extraction (the volume of coal mined from the deposit or part of the deposit relative to its minable reserves) can be resolved essentially into two dimensions. If one assumes that for the area of interest that a seam of constant thickness was mined with rooms of consistent height, then the percent extraction by area and by volume (mining extraction) are the same.

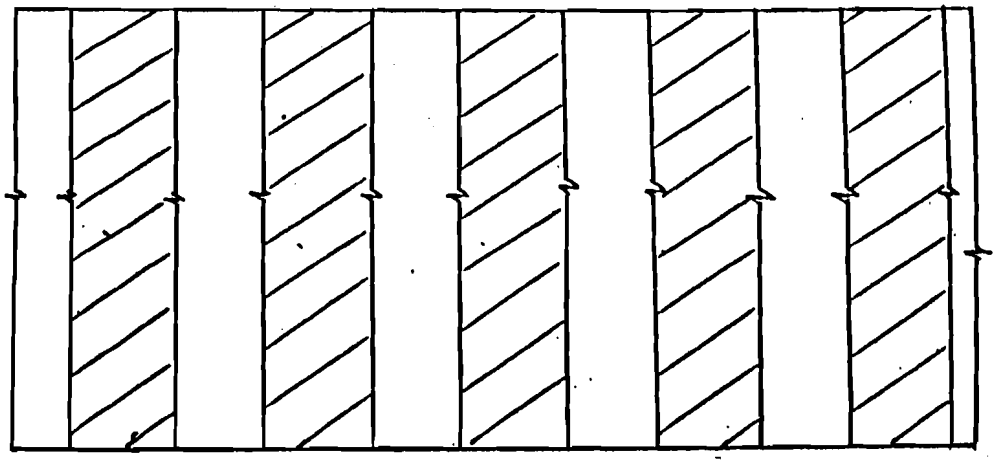
This being the case, then the percentage extraction from an AML site can be obtained from the following expression:

$$\text{Percent extraction} = \frac{\text{Room width}}{\text{Room width} + \text{Pillar width}}$$

At the 1986 test site in Beulah rooms were on average 22 feet wide and separated by 18 foot pillars; this is a "mining extraction" of 55%. Fig. 9-4 illustrates how, for a given percent extraction, the amount of blasting work required at an AML site will vary if rooms are narrower than 22 feet. It is assumed at this time that the typical double row blast that can be taken in one day is 220 feet long, whereas a single row blast of 350 feet is possible. The actual cost for each of these blasts is assumed to be $5000, including drilling, explosives, labor and general costs.

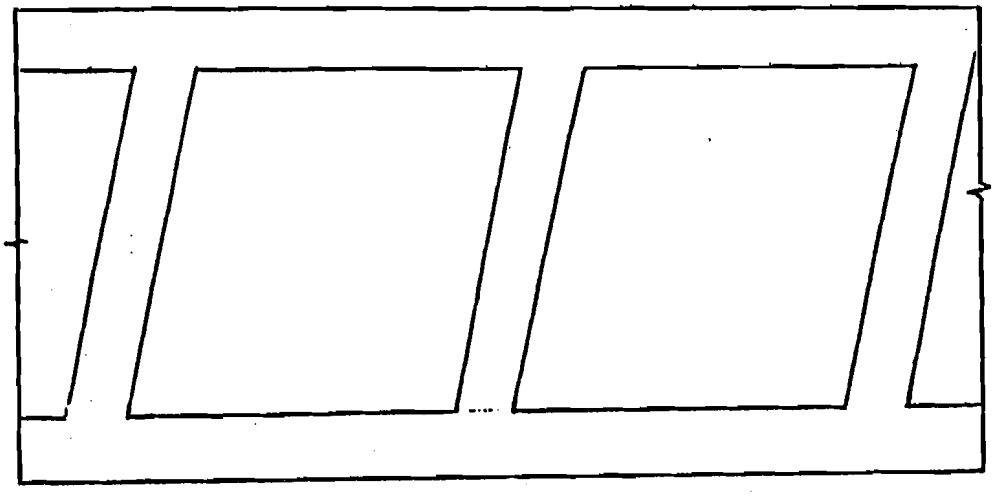
The number of blasts per acre for a given room and pillar configuration is thus expressed by:

$$\text{No. blasts} = \frac{43{,}560}{\text{Blast length} \times (\text{Room width} + \text{Pillar width})}$$



22' wide rooms, 18' pillars

55% extraction : 0.55 x 43560 = 23,958 sq. ft./acre
DOUBLE ROW : 23958 ÷ 220 ÷ 22 = 4.95 blasts/acre = $25,000/acre
SINGLE ROW : 23958 ÷ 350 ÷ 22 = 3.11 blasts/acre = $15,500/acre



12' wide access development, 80' spacing

15% extraction : 0.15 x 43560 = 6,543 sq. ft./acre
SINGLE ROW : 6543 ÷ 350 ÷ 12 = 1.56 blasts/acre = $7,800/acre

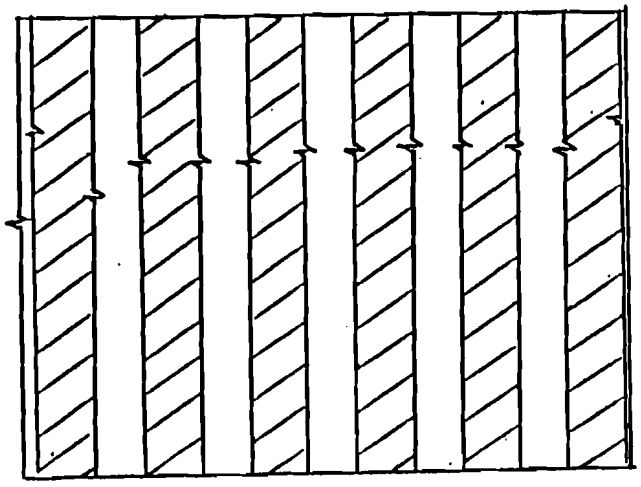


FIG. 9-4: ILLUSTRATING RELATIONSHIP BETWEEN ROOM WIDTH, PERCENT EXTRACTION AND BLASTING COST PER ACRE FOR 55% EXTRACTION

12' wide rooms, 9.8' pillars
55% extraction : 0.55 x 43560 = 23,958 sq. ft./acre
SINGLE ROW : 23958 ÷ 350 ÷ 12 = 5.7 blasts/acre = $28,500/acre

For the typical case at the test site, this gives:

$$\text{No. blasts} = \frac{43,560}{220 \times (22 + 18)} = 5 \text{ blasts/acre}$$
[Double-row blasting]

and

$$\text{No. blasts} = \frac{43,560}{350 \times (22 + 18)} = 3.1 \text{ blasts/acre}$$
[Single-row blasting]

It is therefore possible to estimate that for 55% extraction the cost of single row blasting is, at around $15,000/acre, about 60% of that when a double row is used for rooms of the same size. If, however, the room width which can be successfully blasted with a single row is less, then there will be no cost saving. This is illustrated in the lower part of Fig. 9-4, where it can be seen that a 12 foot wide room, at 55% extraction, is associated with a 9.8 foot wide pillar. There are 5.7 such blasts in an acre, and the cost per acre increases to $25,000.

However, when smaller opening size is combined with greater pillar widths, the cost of blasting an acre may be reduced very appreciably. Part of the test site consisted of panel entries and cross-cuts which were 12 feet wide and separated by 80 foot pillars. This is illustrated in Fig. 9-4, and represents only a 15% extraction. As such, there are only 1.5 such blasts in an acre, and the per acre blasting cost is only $7,500.

It is for this reason that the pro-rated cost for blast number 9, on NC-7, was so much lower than average. Its area of influence was much higher than for room blasting. It also explains why blast number 17 had such a low pro-rated cost, even though the majority of the blast consisted of a 22 foot wide room using two rows of blastholes. Part of this blast also consisted of a 12 foot wide cross-cut, with a much larger area of influence associated with it.

This subject is discussed further in the section describing analysis with the Cost Model for AML blasting.

9.3. AML BLASTING COST MODEL

9.3.1. Introduction

From the previous section of this report it should be obvious that the cost per acre of AML blasting work can be very variable. It is sensitive to variable site parameters, cost of drilling and explosives, and very sensitive to the percent extraction from a site. One of the major aims of the research effort in this project has therefore been the development of a method to model and predict costs for future work of this kind.

To this end a model has been devised, using the Lotus 1-2-3 package, which is based very closely on the spreadsheet designed for analysis of actual costs. However, while much of the information for the actual cost analysis was input from fieldwork cost records, the Cost Model calculates many of these items, including the drilling footage, explosives consumption and actual labor charges that are likely to be associated with a given set of site characteristics.

A schematic for the Cost Model is presented in Fig. 9-5, which also illustrates the formation of the spreadsheet database described previously in this report and presented in Appendix G.

The general aims of the Cost Model developed have been the following:

1. Establish typical costs for each item (drilling, explosives, labor, general field and project expenses) based largely on the 1986 experience, and on data included as Appendix F in this report.

2. Design the model in such a way that the following site parameters may be varied, and accurately incorporated:

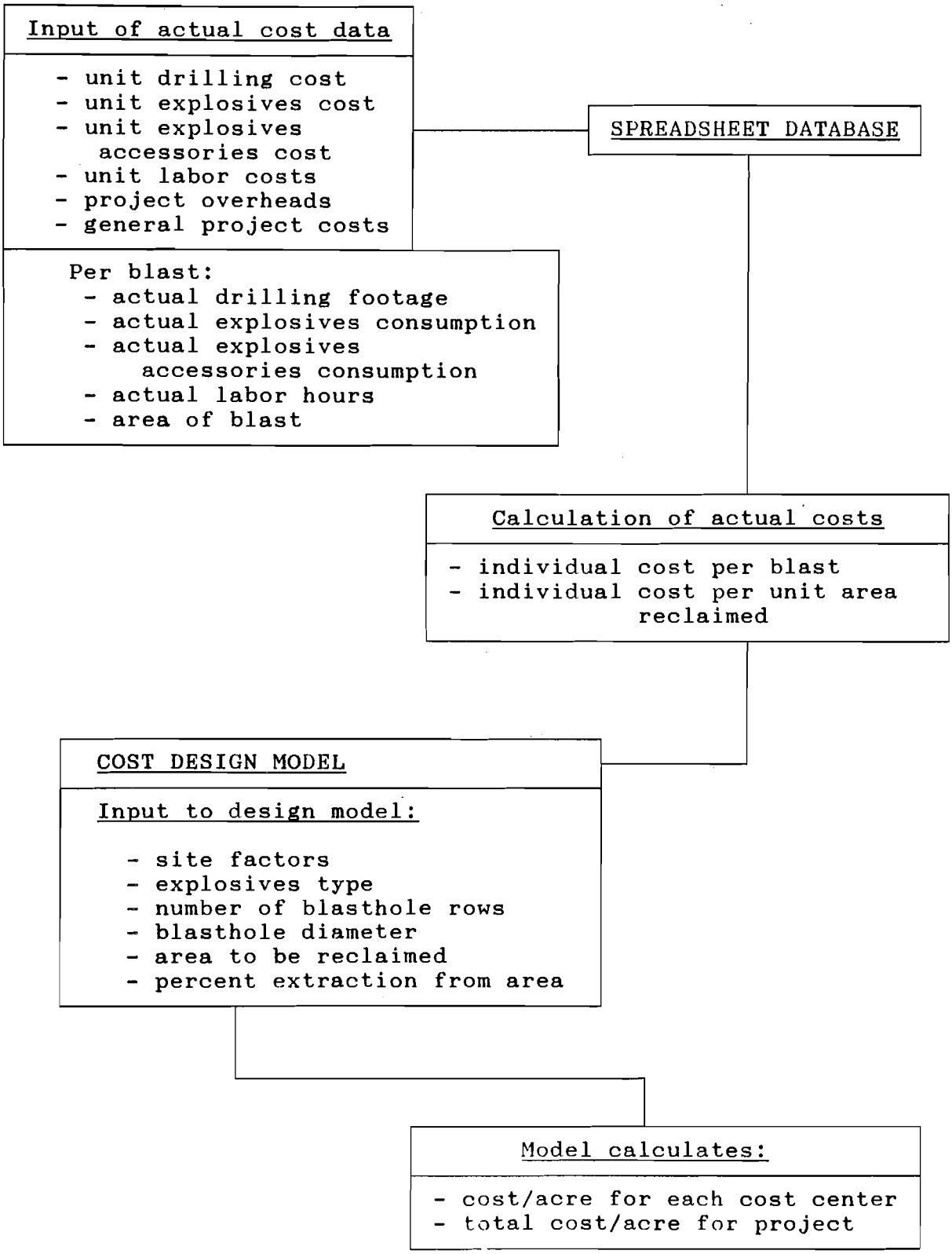
   - choice of single or double row blasting
   - selection of room width, and blast length
   - selection of average depth of overburden
   - selection of overall site area
   - selection of a percentage extraction
   - selection of explosive type(s)

3. Based on the above cost items and selected site parameters, the following must be calculated automatically by the model:

   - footages of room line-up and production drilling
   - theoretical consumption of explosives and explosives accessories
   - labor charges for line-up, preparation and blasting
   - general costs (travel, subsistence, rentals, etc.)

4. Generation of tables and graphs by the model to carry out a cost "parameter analysis". This will calculate the change in blasting cost per acre due to variation in the following important criteria:

FIGURE 9-5 : SCHEMATIC FOR COST DESIGN MODEL



- overburden depth
- drilling cost
- explosives cost
- labor cost
- percent mining extraction from a site
- size of site area

For this model single-row and double-row cases are considered separately. In practice a combination of both configurations will probably be required at a site, with the relative importance of one or the other dependent very largely on the distribution of development widths and types within it.

9.3.2. "Base Case" Cost Model Parameters

In order to model the effects of varying site and cost parameters on the overall blasting cost per acre it was first necessary to establish "base" values for each of these parameters. The Cost Model consisted of two spreadsheets, very similar to those employed for analysis of actual cost data from fieldwork. The first contains unit cost data for drilling and explosives, and the itemized calculation of unit labor and general site costs for a 10 acre site. This data is then used by the second spreadsheet, which calculates the total cost per acre based on the input site characteristics.

Table 9-4 shows the first of these spreadsheets, with all of the "base case" values employed. Values which are marked by "****" in this table are those which are actually calculated by the spreadsheet, and are dependent on site conditions. Their calculation will be described later.

A drilling cost of $1.00/ft is assumed; this is consistent with current prevailing rates in North Dakota and Montana, but may have to be revised to reflect the more favorable economic climate for drilling firms in some other parts of the country.

The explosives costs for ANFO and slurry are a little lower than those we experienced in our 1986 testwork, but are nevertheless considered appropriate for a non-bulk product for a single delivery of a size consistent with a typical 10 acre site. The prices of explosives accessories (delays, etc.) are virtually identical to those encountered in our field work.

Actual calculation methods employed by the spreadsheet will be described in the next section of this report. It was felt that the model should be developed for a ten-acre site, as this was consistent with the size of the 1986 test site. In most cases, since blasting cost is pro-rated on a per acre basis by the Cost Model, one would not expect a very marked change in cost per acre as site size increases. There would be some "economies of scale" for a larger site; this is discussed later.

AML PROJECT - BLAST COSTING MODEL.

		COSTS USED	
		8 INCH	6 INCH
DRILLING COST	/FT	$1.500	$1.000
EXPLOSIVES COSTS			BATCH 1
ANFO	/LB		$0.145
SLURRY	/LB		$0.360
BOOSTERS	EACH		$3.200
DTH DELAYS	EACH		$1.750
SURF. DELAYS	EACH		$2.250
HOLE PLUGS	EACH		$1.850
CAPS	EACH		$2.000
PRIMALINE	/FT		$0.079
NONEL T'LINE	/FT		$0.057
BALER TWINE	/FT		$0.003
LABOR COSTS			
BLAST. ENGR.	/HR		$30.00
FIELD ENGR.	/HR		$20.00
LABORER	/HR		$10.00

GENERAL COSTS

TRAVEL (PICKUP RENTAL, MILEAGE, GAS, MOBILIZATION)			
TOTAL FOR PROJECT			$3,000.00
PER BLAST			****
SUBSISTENCE			
ENGINEERS	@	$50 /DAY =	****
LABORERS	@	$40 /DAY =	****
TOTAL FOR PROJECT			****
PER BLAST			****
RENTALS			
SEISMOGRAPH	@	$500 /MONTH =	****
MAGAZINES	@	$400 /MONTH =	****
TRAILER	@	$350 /MONTH =	****
TOTAL FOR PROJECT			****
PER BLAST			****
MISCELLANEOUS			
MISC. FIELD EQUIPMENT			$750.00
GENERAL SITE SERVICES			$450.00
CONSUMABLE FIELD SUPPLIES			$300.00
TOTAL FOR PROJECT			$1,500.00
PER BLAST			****

TABLE 9-4: ILLUSTRATING COST CENTERS AND VALUES USED FOR COST MODEL SPREADSHEET (**** denotes calculated value)

The duration of the work, for a site of a given size is obviously a function of the size of the drilling and blasting crew. This in turn affects the total cost of travel, subsistence and rental charges for a project.

A total travel cost of $3000 has been used for work at a ten acre site. This is intended to include rental of at least one pickup, the mileage charge on this and at least one other vehicle, gas, and equipment mobilization. It has been assumed that the site would be located at such a distance from the offices of the companies involved with the blasting work that it would be possible to drive to the site at the beginning, and back to home location at the end of the project. If the distances involved were significantly greater, then the $3000 estimate would have to be revised accordingly.

A daily subsistence of $50 has been assumed for each of the two engineers for room and meals. As a minimum crew size is being assumed, it is felt that at least one of the laborers should be experienced with this type of work. As such, therefore, it may well be necessary to bring this person in from outside of the site area. A daily subsistence for one laborer at $40 per day has also been included in the Cost Model. The second laborer, required for deeper overburden sites, could probably be hired locally.

It has been assumed that, as was the case for 1986 testwork, rental of a powder magazine (semi-trailer type), plus smaller magazines for boosters and primers, would be required. In addition, some kind of small mobile trailer would be needed for use as a field office. In most cases it would be advisable, even if not mandatory, to use a blasting seismograph at the site in case of possible complaint and/or litigation by local residents. This may not be necessary at very remote sites.

Miscellaneous general site expenses would include equipment such as tapes, blasting cable, blasting machine and other sundry tools and items. These would in many cases be re-usable items, and for this reason their cost should not be extended in a strictly linear manner to larger blast sites than 10 acres. General site services may include items such as limited dozer work to level magazine sites or minor work on existing roads. It would not include a major access development. Consumable field supplies would typically include marker stakes, tape and sundry other items.

9.3.3. Variable "Base Case" Parameters

The second spreadsheet used by the Cost Model is presented as Table 9-5. Though this is concerned largely with calculation (calculated values are shown as "****" here), there are some items that are employed which may be considered as variable "base case" data. These are parameters that may well be fixed

ANL PROJECT - BLAST COST MODEL. ANALISYS NO. nn

COMMENTS: VARIABLE nnnnn
10 ACRE SITE. BASE CASE. nnnnnn ROW.

DESIGN PARAMETERS :

HOLE DIAM.	6 INCH	NO. ROWS :	1 or 2
LENGTH	350 or 200 FT	ROOM WIDTH	12 or 20 FT
ROW SEPARAT.	8 FT	HOLE SPACING	12 or 15 FT
# HOLES	****	AV. DEPTH TO VOID	55 FT
PILLAR WIDTH	**** FT	AREA OF INFLUENCE	**** SQ. FT
% EXTRACTION	40.0%	VOID/ACRE	**** SQ. FT
SITE AREA	10 ACRES	NO. BLASTS	****

LOADING PARAMETERS : CHOICE ------> n

1. ALL ANFO
2. BOTTOM DECK ENERGEL
3. BOTTOM DECK IREGEL
4. BOTTOM 2 DECKS ENERGEL
5. BOTTOM 2 DECKS IREGEL

						COST
DRILLING						
	ROOM LINE-UP	**** FT @	$ 1.000 /FT	=		****
	PRODUCTION	**** FT @	$ 1.000 /FT	=		****
	TOTAL					****
BLASTING						
	ANFO	**** LBS @	$ 0.145 /LB	=		****
	SLURRY	**** LBS @	$ 0.360 /LB	=		****
	BOOSTERS	**** @	3.200 EACH	=		****
	DTH DELAYS	**** @	1.750 EACH	=		****
	SURF. DELAYS	**** @	2.250 EACH	=		****
	HOLE PLUGS	**** @	1.850 EACH	=		****
	CAPS	**** @	2.000 EACH	=		****
	PRIMALINE	**** FT @	$ 0.079 /FT	=		****
	BALER TWINE	**** FT @	$ 0.003 /FT	=		****
	TOTAL					****
LABOR						
	LINEUP	1.75 HRS @	$ 20.00 /HR	=		$35.00
	PREPARATION	3.50 HRS @	$ 30.00 /HR	=		$105.00
	BLASTING	10.00 HRS @	$ **** /HR	=		****
	TOTAL					****
	OVERHEADS	50 % DIR LAB		=		****
	TOTAL					****
GENERAL						
	TRAVEL	/BLAST		=		****
	SUBSISTENCE	/BLAST		=		****
	RENTALS	/BLAST		=		****
	MISC.	/BLAST		=		****
	TOTAL	/BLAST				****
TOTAL COST				=		****
COST/ACRE	(BASED ON THIS BLAST)			=		****

TABLE 9-5: ILLUSTRATING SPREADSHEET USED FOR COST MODEL CALCULATIONS (**** DENOTES CALCULATED VALUE)

for a given site, but will vary from one site to another.

9.3.3.1. Blasthole Diameter

During the 1986 testwork it was found in most cases that 6 inch blastholes were successful in collapsing underground openings in the material types and overburden depths encountered. The Cost Model assumes that blastholes are of this diameter. If analysis were to be required at other diameters, such as 8 inch, all that would have to be provided in addition is a table of typical loading weights for different blasthole depths at this diameter. Such a table for 6 inch holes is described and included later in this chapter.

9.3.3.2. Number of Blasthole Rows

As previously discussed, single or double row blasting can be considered by the Cost Model. The choice here will depend very much on overburden strengths, and on the widths and types of underground development.

9.3.3.3. Room Width

This is obviously very site specific, and may also vary within a site. In the latter case, it would be necessary to run the Cost Model for each room width in order that the pro-rated costs per acre accurately reflect the development being blasted.

9.3.3.4. Blasthole Spacing

This will depend very much on material types being blasted. In most analyses presented in this report it has been assumed that single row blasting uses 12 foot spacing, and double row blasts have holes spaced 15 feet apart on rows separated by 8 feet.

9.3.3.5. Blast Length, Crew Size and Project Duration

During 1986 testwork it was found that the minimum practical labor force was three men. However, for larger blasts (>25 holes) or those in deeper cover requiring 5 explosive decks, four men were required, using two loading crews of two men each.

For the "base case" a 3 man team is considered, consisting of a senior blasting engineer, a field engineer, and one additional laborer. This is the absolute minimum that is considered necessary for work of this kind, and would involve

labor by all three people in the actual hole loading process, and considerable manual labor by the field engineer throughout the fieldwork. When the overburden depth exceeds 60 feet, the Cost Model automatically includes a second laborer at $10.00/hr, as this is the depth at which it is assumed that 5 deck blasting would be employed.

The use of 50 working hours per week is consistent with experience during our testwork. The $10.00/hr charge for the laborers has been set a little higher than might be currently expected to reflect overtime rates and to simplify calculations.

It was found during the 1986 testwork that the maximum number of blastholes that could be loaded in a 10 hour day was about 30. The practice of leaving a blast partly loaded overnight would certainly be prohibited by safety regulations unless a permanent guard was mounted. On this basis, the length of typical single and double row blasts may be calculated, which is found to be about 350 feet and 200 feet respectively. This assumes a blasthole spacing of 12 feet for single row and 15 feet for double row, which are consistent with successful practice during the 1986 testwork.

From this it is then possible to calculate the number of blasts that will have to be taken in a 10 acre site, and thus the project duration can be determined. The actual calculations carried out by the Cost Model to do this are described in section 9.3.4.8. of this report.

9.3.3.6. Depth of Overburden

This is such an important site parameter in determining AML blasting costs that it is considered in every analysis using the Cost Model. It is used as a variable parameter, and when other site and cost parameters are being varied, the model is still run at four different overburden depths. The value for overburden depth which might be considered typical for sites such as the one in which testwork was carried out in 1986 is 50-55 feet.

9.3.3.7. Percent Extraction

This parameter is studied in some depth later in section 9.4.2.5. A "base case" value of 40% is considered typical for near surface underground coal deposits mined using the room and pillar method. The value of 55% for the 1986 test site was rather higher than might be expected, given the material types. The 22 foot wide rooms encountered are rather wider than usual for this type of mining operation.

9.3.3.8. Theoretical Explosives Consumption

In order to calculate the cost of the explosives that will be consumed in a blast, it is necessary for the Cost Model to use "base case" data regarding typical explosive weights for a blasthole of a given size and length.

This information is obtained from the technical data analysis described in Chapter 8 of this report. Table 9-6 gives this information for 6 inch blastholes for the five different loading options available. These are:

1. Use of ANFO only.
2. Use of ENERGEL (slurry) explosive in the bottom deck.
3. Use of ENERGEL in the bottom two decks of 4 and 5 deck holes.
4. Use of IREGEL (slurry) explosive in the bottom deck.
5. Use of IREGEL in the bottom two decks of 4 and 5 deck holes.

If blasting is to take place in very different material types, or is to be attempted in overburden depths outside of the range presented here, then it would be necessary to generate additional tables for explosives consumption. This would be more of a concern initially at the technical design stage than for cost analysis. Different types of slurry explosives may generate slightly different explosive weights for a given blasthole depth. However, it is felt that the five loading options presented above cover most of the conditions that will be encountered in this type of work.

9.3.4. Cost Model Calculations

The calculations as they are actually carried out by the Cost Model are described in the order in which they appear on the two pages of the spreadsheet. They refer to the lines marked "****" in Tables 9-4 and 9-5.

9.3.4.1. Travel Costs per Blast

A total travel cost for a ten acre site is assumed to be $3000. This is pro-rated in a linear manner for larger sites - the cost foe a 15 acre site is assumed by the Cost Model to be $4500, for example. The total travel cost is divided equally

TABLE 9-6: EXPLOSIVES WEIGHTS AND BOREHOLE DEPTHS AS EMPLOYED BY AML BLASTING COST MODEL. (ALL WEIGHTS EXPRESSED IN POUNDS)

DIAMETER	6 INCH		6 INCH		6 INCH		6 INCH		6 INCH	
LOADING	ALL ANFO		BOTTOM ENERGEL		BOTTOM IREGEL		BOTTOM 2 ENERGEL		BOTTOM 2 IREGEL	
DEPTH FT	WT. ANFO	WT. SLURRY	WT. ANFO	WT. SLURRY	WT. ANFO	WT. SLURRY	WT. ANFO	WT. SLURRY	WT. ANFO	WT. SLURRY
35	104.3	62.6	62.6	41.7	62.6	49.1	62.6	41.7	62.6	49.1
36	109.5	67.8	67.8	41.7	67.8	49.1	67.8	41.7	67.8	49.1
37	119.9	78.2	78.2	41.7	78.2	49.1	78.2	41.7	78.2	49.1
38	119.9	78.2	78.2	41.7	78.2	49.1	78.2	41.7	78.2	49.1
39	119.9	78.2	78.2	41.7	78.2	49.1	78.2	41.7	78.2	49.1
40	125.1	83.4	83.4	41.7	83.4	49.1	83.4	41.7	83.4	49.1
41	125.1	93.8	93.8	31.3	93.8	36.8	62.6	62.6	62.6	73.6
42	135.6	93.8	93.8	41.7	93.8	49.1	62.6	73.0	62.6	85.9
43	146.0	104.3	104.3	41.7	104.3	49.1	62.6	83.4	62.6	98.1
44	146.0	104.3	104.3	41.7	104.3	49.1	62.6	83.4	62.6	98.1
45	146.0	104.3	104.3	41.7	104.3	49.1	62.6	83.4	62.6	98.1
46	146.0	104.3	104.3	41.7	104.3	49.1	62.6	83.4	62.6	98.1
47	156.4	114.7	114.7	41.7	114.7	49.1	73.0	83.4	73.0	98.1
48	161.6	119.9	119.9	41.7	119.9	49.1	78.2	83.4	78.2	98.1
49	172.1	130.3	130.3	41.7	130.3	49.1	88.6	83.4	88.6	98.1
50	166.8	125.1	125.1	41.7	125.1	49.1	83.4	83.4	83.4	98.1
51	166.8	125.1	125.1	41.7	125.1	49.1	83.4	83.4	83.4	98.1
52	172.1	125.1	125.1	46.9	125.1	55.2	83.4	88.6	83.4	104.3
53	182.5	130.3	130.3	52.1	130.3	61.3	83.4	99.1	83.4	116.5
54	192.9	140.8	140.8	52.1	140.8	61.3	88.6	104.3	88.6	122.7
55	203.3	151.2	151.2	52.1	151.2	61.3	99.1	104.3	99.1	122.7
56	213.8	156.4	156.4	57.4	156.4	67.5	104.3	109.5	104.3	128.8
57	224.2	166.8	166.8	57.4	166.8	67.5	109.5	114.7	109.5	134.9
58	234.6	172.1	172.1	62.6	172.1	73.6	114.7	119.9	114.7	141.1
59	250.3	187.7	187.7	62.6	187.7	73.6	125.1	125.1	125.1	147.2
60	229.4	177.3	177.3	52.1	177.3	61.3	125.1	104.3	125.1	122.7
61	239.8	177.3	177.3	62.6	177.3	73.6	125.1	114.7	125.1	134.9
62	250.3	187.7	187.7	62.6	187.7	73.6	125.1	125.1	125.1	147.2
63	260.7	198.1	198.1	62.6	198.1	73.6	135.6	125.1	135.6	147.2
64	271.1	208.6	208.6	62.6	208.6	73.6	146.0	125.1	146.0	147.2
65	281.5	213.8	213.8	67.8	213.8	79.7	146.0	135.6	146.0	159.5
66	292.0	213.8	213.8	78.2	213.8	92.0	146.0	146.0	146.0	171.8

into whatever number of blasts is calculated for the site (calculation of this is explained later).

9.3.4.2. Subsistence for Project

It is assumed that one blast per day is taken for whatever site size and number of blasthole rows is selected. The total subsistence for a project is thus calculated as follows:

Subsistence = 2 x daily rate x no. blasts in site (for engineers)

Subsistence = No. laborers x daily rate x # blasts in site (for laborers)

The maximum number of laborers requiring subsistence would generally be one, as explained earlier, whatever crew size may be required.

The total subsistence cost is shared equally between blasts as was travel cost.

9.3.4.3. Rentals Cost

The actual rentals cost is arrived at in the following manner:

$$\text{Rental cost} = \text{Item rental/month} \times \frac{\text{\# blasts in site}}{20}$$

It is assumed here that blasting is carried out on weekdays only - there are therefore on average 20 blasts taken per month. The per blast cost of rentals is assigned as described above for travel and subsistence.

9.3.4.4. Miscellaneous Items

As mentioned earlier, some field equipment will be re-usable; this is estimated by the Cost Model as follows:

Field equipment cost = \$750 (for sites less than 10 acres)

$$\text{Field equipment cost} = \$750 \times \frac{\text{Site size}}{10} \times 0.75$$

(for sites > 10 acres)

The costs of general site services and consumable field supplies are calculated as follows:

$$\text{Cost} = \$450 \text{ (or } \$300) \times \frac{\text{Site size}}{10}$$

Miscellaneous costs are equally divided into the number of blasts calculated for the site.

9.3.4.5. Number of Holes in Blast

The number of holes in a blast is calculated by the Cost Model according to the following expression:

$$\#\text{ holes} = \#\text{ rows} \times \left[\text{Integer value of } \left(\frac{\text{blast length}}{\text{hole spacing}}\right) + 1 \right]$$

9.3.4.6. Pillar Width

Pillar width and percent extraction are characterized by the following formula:

$$\%\text{ extraction} = \frac{\text{Room width}}{(\text{Room + pillar width})} \times 100$$

Pillar width is thus given by:

$$\text{Pillar width} = \text{Room width} \times \left[\frac{100}{\%\text{ extr.}} - 1\right]$$

9.3.4.7. Area of Influence and Void/acre

Area of influence in this column of the Cost Model is calculated as the actual blast area:

Area of influence = Blast length x room width

The void per acre is simply the number of square feet in an acre (43,560) multiplied by the percentage extraction as a fraction. For 40% extraction, therefore, the void area per acre = 43560 x 0.4 = 17,424 sq. ft.

9.3.4.8. Number of Blasts

This calculation, which is very important for the calculation of project duration and other site related cost

centers, is calculated as follows:

$$\text{\# blasts} = \text{Integer value of } \left[\frac{\text{Site area} \times 43560 \times \text{\% extr.}}{\text{blast length} \times \text{room width} \times 100} \right] + 1$$

(where site area is in acres and blast dimensions are in feet)

9.3.4.9. Room Line-up Drilling Footage

The Cost Model assumes that a typical blast (whether 200 feet long in two-row or 350 feet long in single row cases) requires about 6 line-up holes to ensure that the drilling pattern is centered on the room. In many cases, especially where a fixed room width and pillar width was not adhered to, this may in fact be a little conservative. However, it is hoped that the initial exploration drilling (not included in the Cost Model) would help here. The drilling line-up footage for each blast is thus calculated as six times the average overburden depth.

9.3.4.10. Production Drilling Footage

This is simply calculated by:

Production footage = # holes x average overburden depth

9.3.4.11. ANFO Explosive Consumption

Depending on what loading parameters option is selected, the Cost Model obtains the weight of ANFO for that hole depth using a spreadsheet "lookup" function for the information contained in Table 9-6. It scans the hole depths until the correct depth is reached, and then scans across to the appropriate column.

The returned ANFO weight is simply multiplied by the number of holes in the blast.

9.3.4.12. Slurry Explosive Consumption

This is obtained in exactly the same way as described above for ANFO, if options 2 to 5 are selected for loading parameters.

9.3.4.13. Consumption of Explosives Accessories

- Boosters and DTH delays;

The total number of boosters and down-the-hole delays is

calculated as follows:

Total = # holes x # decks in a hole

The Cost Model determines the number of decks in a hole according to the selected depth :

< 41 ft	:	3 decks
41-59 ft	:	4 decks
>59 ft	:	5 decks

In the case of boosters an additional 2 boosters per blast is added to the above total to cover wastage.

- Surface delays, hole plugs and caps

The number of surface delays and hole plugs used in a blast is simply set to the number of holes in it. It is assumed that one electric blasting cap per blast is used.

- RX Primaline and baler twine footages

It is assumed by the cost model that the primaline footage used per blasthole is equal to the depth of the hole plus an additional 3 feet used for tie-off at the hole collar. As two strands of baler twine were used to hold the plug in place at the bottom of the hole, the associated footage is assumed to be twice that for primaline. Primaline footage is calculated as follows:

Primaline footage = Production drilling footage + [# holes x 3]

9.3.4.14. Labor Costs

The calculation of labor costs by the Cost Model is based on the assumptions about charges and hours described earlier. The model automatically "adds on" the additional laborer when blasthole depths go over 60 feet and five explosive decks need to be loaded.

Labor overheads are calculated and included in the labor charges by the Cost Model. The rate of 50% has been assumed for the cases studied in this report.

9.3.4.15. General Costs

The general cost items of travel, subsistence, rentals and miscellaneous that were previously calculated on a per-blast basis are totalled for each blast.

9.3.4.16. Cost of Blast

The total cost of a blast is obtained by adding the subtotals of drilling, blasting, labor and general costs.

9.3.4.17. Cost per Acre, Based on this Blast

The pro-rated cost per acre of a blast is obtained from the actual blast cost according to the following expression:

$$\text{Cost/acre} = \frac{\text{Cost of blast} \times 43560 \times \%\ \text{extraction}}{\text{Blast Area of influence}}$$

9.4. COST MODEL ANALYSIS

9.4.1. General Points

As with previous cost analysis work carried out in conjunction with this project, two basic approaches to blasting were considered : single row and double rows of blastholes. The analysis of the effect of varying unit costs was carried out at at four different overburden depths : 35 feet, 45 feet, 55 feet and 65 feet. In general, therefore, there were 8 separate analyses carried out for a single varying cost parameter. The other "base case" conditions which were applied have been previously described.

The loading method assumed as typical consisted of slurry (ENERGEL type) explosive in the bottom deck, and ANFO in the other decks. The model may be used to generate variable cost data using other loading configurations if desired.

The spreadsheet design was essentially identical to that used for analysis of actual cost data from the fieldwork. The difference was the use of a "MACRO" command structure to calculate and tabulate total costs for a given variable cost. Each table produced contains the value of the varied parameter, the total cost per acre resulting, and a breakdown of this into the major cost centers of drilling, blasting, labor and general expenses.

A total of 35 tables of cost data have been produced for this study using the Cost Model. These are identified according to the codes presented in Table 9-6, and are contained in Appendix G of this report. One of these analyses is presented as an example in Tables 9-7, 9-8 and 9-9; the first two of these tables present the Cost Model spreadsheet, including calculated values. Table 9-9 shows the variation of AML blasting reclamation cost centers and total cost for variable overburden depths, using double-row blasting and the "base case" parameters described previously.

TABLE 9-6 : SUMMARY OF COST ANALYSES CARRIED OUT USING COST MODEL

Studied Parameter	No. Rows	O/B Depth	Minimum Value	Maximum Value	Incre-ment	Code #
Overburden Depth	1	-	35 ft.	66 ft.	1 ft	OD1
	2	-	35 ft.	66 ft.	1 ft	OD2
	1	-	35 ft.	66 ft.	1 ft	OD3
Drilling Cost ($/ft)	1	35 ft	$0.50	$2.50	$0.10	DC1
	1	45 ft	$0.50	$2.50	$0.10	DC2
	1	55 ft	$0.50	$2.50	$0.10	DC3
	1	65 ft	$0.50	$2.50	$0.10	DC4
	2	35 ft	$0.50	$2.50	$0.10	DC5
	2	45 ft	$0.50	$2.50	$0.10	DC6
	2	55 ft	$0.50	$2.50	$0.10	DC7
	2	65 ft	$0.50	$2.50	$0.10	DC8
Explosives Cost ($/lb AN) ($/lb sl.)	1	35 ft	$0.10	$0.23	$0.01	EC1
	1	45 ft	$0.35	$0.48	$0.01	EC2
	1	55 ft	$0.10	$0.23	$0.01	EC3
	1	65 ft	$0.35	$0.48	$0.01	EC4
	2	35 ft	$0.10	$0.23	$0.01	EC5
	2	45 ft	$0.35	$0.48	$0.01	EC6
	2	55 ft	$0.10	$0.23	$0.01	EC7
	2	65 ft	$0.35	$0.48	$0.01	EC8
Labor Cost (% base rate)	1	35 ft	85%	115%	5%	LC1
	1	45 ft	85%	115%	5%	LC2
	1	55 ft	85%	115%	5%	LC3
	1	65 ft	85%	115%	5%	LC4
	2	35 ft	85%	115%	5%	LC5
	2	45 ft	85%	115%	5%	LC6
	2	55 ft	85%	115%	5%	LC7
	2	65 ft	85%	115%	5%	LC8
Percent extraction (%)	1	35 ft	20%	60%	2%	PE1
	1	45 ft	20%	60%	2%	PE2
	1	55 ft	20%	60%	2%	PE3
	1	65 ft	20%	60%	2%	PE4
	2	35 ft	20%	60%	2%	PE5
	2	45 ft	20%	60%	2%	PE6
	2	55 ft	20%	60%	2%	PE7
	2	65 ft	20%	60%	2%	PE8

AML PROJECT - BLAST COSTING MODEL.

		COSTS USED	
		8 INCH	6 INCH
DRILLING COST	/FT	$1.500	$1.000
EXPLOSIVES COSTS			BATCH 1
ANFO	/LB		$0.145
SLURRY	/LB		$0.360
BOOSTERS	EACH		$3.200
DTH DELAYS	EACH		$1.750
SURF. DELAYS	EACH		$2.250
HOLE PLUGS	EACH		$1.850
CAPS	EACH		$2.000
PRIMALINE	/FT		$0.079
NONEL T'LINE	/FT		$0.057
BALER TWINE	/FT		$0.003
LABOR COSTS			
BLAST. ENGR.	/HR		$30.00
FIELD ENGR.	/HR		$20.00
LABORER	/HR		$10.00
GENERAL COSTS			
TRAVEL (PICKUP RENTAL, MILEAGE, GAS, MOBILIZATION)			
TOTAL FOR PROJECT			$3,000.00
PER BLAST			$68.18
SUBSISTENCE			
ENGINEERS @	$50 /DAY =		$4,400.00
LABORERS @	$40 /DAY =		$1,760.00
TOTAL FOR PROJECT			$6,160.00
PER BLAST			$140.00
RENTALS			
SEISMOGRAPH @	$500 /MONTH =		$1,100.00
MAGAZINES @	$400 /MONTH =		$880.00
TRAILER @	$350 /MONTH =		$770.00
TOTAL FOR PROJECT			$2,750.00
PER BLAST			$62.50
MISCELLANEOUS			
MISC. FIELD EQUIPMENT			$750.00
GENERAL SITE SERVICES			$450.00
CONSUMABLE FIELD SUPPLIES			$300.00
TOTAL FOR PROJECT			$1,500.00
PER BLAST			$34.09

TABLE 9-7: COST CENTERS AND VALUES USED FOR CALCULATION OF COST MODEL FOR VARIABLE OVERBURDEN DEPTH (ANALYSIS OD2)

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```
AML PROJECT - BLAST COST MODEL.            ANALISYS NO.              OD2
============================
                          COMMENTS:          VARIABLE DEPTH
                                    10 ACRE SITE. BASE CASE. DOUBLE ROW.
DESIGN PARAMETERS :       ---------------------------------------------
-------------------
HOLE DIAM.            6 INCH      NO. ROWS :              2
LENGTH              200 FT        ROOM WIDTH             20 FT
ROW SEPARAT.          8 FT        HOLE SPACING           15 FT
# HOLES              28           AV. DEPTH TO VOID      50 FT
PILLAR WIDTH       30.0 FT        AREA OF INFLUENCE    4000 SQ. FT
% EXTRACTION       40.0%          VOID/ACRE           17424 SQ. FT
SITE AREA            10 ACRES     NO. BLASTS             44

LOADING PARAMETERS :    CHOICE      ------)       2
-------------------
1. ALL ANFO                       4. BOTTOM 2 DECKS ENERGEL
2. BOTTOM DECK ENERGEL            5. BOTTOM 2 DECKS IREGEL
3. BOTTOM DECK IREGEL
-----------------------------------------------------------------------
                                                                 COST
DRILLING                                                         ----
--------     ROOM LINE-UP      150 FT @    $  1.000 /FT =       $150.00
             PRODUCTION       1400 FT @    $  1.000 /FT =     $1,400.00
                                                            ------------
                  TOTAL                                       $1,550.00
BLASTING
--------     ANFO           3503.7 LBS @   $  0.145 /LB    =    $508.04
             SLURRY         1167.9 LBS @   $  0.360 /LB    =    $420.45
             BOOSTERS          114      @     3.200 EACH   =    $364.80
             DTH DELAYS        114      @     1.750 EACH   =    $199.50
             SURF. DELAYS       28      @     2.250 EACH   =     $63.00
             HOLE PLUGS         28      @     1.850 EACH   =     $51.80
             CAPS                1      @     2.000 EACH   =      $2.00
             PRIMALINE       1484 FT    @   $ 0.079 /FT    =    $116.49
             BALER TWINE     2968 FT    @   $ 0.003 /FT    =      $7.42
                                                            ------------
                  TOTAL                                       $1,733.50

LABOR
-----        LINEUP           1.75 HRS @   $  20.00 /HR     =     $35.00
             PREPARATION      3.50 HRS @   $  30.00 /HR     =    $105.00
             BLASTING        10.00 HRS @   $  60.00 /HR     =    $600.00
                                                            ------------
             TOTAL                                              $740.00
             OVERHEADS          50 % DIR LAB                =    $370.00
                                                            ------------
                  TOTAL                                       $1,110.00

GENERAL
-------        TRAVEL              /BLAST                    =     $68.18
               SUBSISTENCE         /BLAST                    =    $140.00
               RENTALS             /BLAST                    =     $62.50
               MISC.               /BLAST                    =     $34.09
                                                            ------------
                  TOTAL            /BLAST                       $304.77
                                                            ------------
-----------------------------------------------------------------------
TOTAL COST                                                   =  $4,698.27

COST/ACRE    (BASED ON THIS BLAST)                           = $20,465.67

=======================================================================
```

TABLE 9-8: SPREADSHEET USED FOR COST MODEL CALCULATIONS FOR VARIABLE OVERBURDEN DEPTH (ANALYSIS OD2)

AML PROJECT COST MODEL - VARIABLE DEPTH, 2 ROWS. (OD2

PARAMETER	C O S T		P E R		A C R E
DEPTH (FT)	DRILL COST	BLAST COST	LABOR COST	GENERAL COST	COST/ACRE
35	$4,726	$5,688	$4,835	$1,328	$16,577
36	$4,861	$5,790	$4,835	$1,328	$16,815
37	$4,996	$5,985	$4,835	$1,328	$17,144
38	$5,131	$5,995	$4,835	$1,328	$17,289
39	$5,266	$6,005	$4,835	$1,328	$17,435
40	$5,401	$6,108	$4,835	$1,328	$17,672
41	$5,536	$6,448	$4,835	$1,328	$18,148
42	$5,672	$6,916	$4,835	$1,328	$18,751
43	$5,807	$7,111	$4,835	$1,328	$19,080
44	$5,942	$7,121	$4,835	$1,328	$19,226
45	$6,077	$7,131	$4,835	$1,328	$19,371
46	$6,212	$7,142	$4,835	$1,328	$19,516
47	$6,347	$7,336	$4,835	$1,328	$19,846
48	$6,482	$7,439	$4,835	$1,328	$20,083
49	$6,617	$7,633	$4,835	$1,328	$20,413
50	$6,752	$7,551	$4,835	$1,328	$20,466
51	$6,887	$7,561	$4,835	$1,328	$20,611
52	$7,022	$7,800	$4,835	$1,328	$20,985
53	$7,157	$8,132	$4,835	$1,328	$21,451
54	$7,292	$8,326	$4,835	$1,328	$21,781
55	$7,427	$8,521	$4,835	$1,328	$22,111
56	$7,562	$8,852	$4,835	$1,328	$22,577
57	$7,697	$9,047	$4,835	$1,328	$22,907
58	$7,832	$9,378	$4,835	$1,328	$23,373
59	$7,967	$9,665	$4,835	$1,328	$23,795
60	$8,102	$9,637	$5,489	$1,502	$24,729
61	$8,237	$10,105	$5,489	$1,502	$25,332
62	$8,372	$10,299	$5,489	$1,502	$25,662
63	$8,507	$10,494	$5,489	$1,502	$25,992
64	$8,642	$10,689	$5,489	$1,502	$26,321
65	$8,777	$11,020	$5,489	$1,502	$26,788
66	$8,912	$11,488	$5,489	$1,502	$27,391

TABLE 9-9: TABLE SHOWING VARIATION OF BLAST-ASSOCIATED COSTS WITH OVERBURDEN DEPTH, GENERATED BY COST MODEL (ANALYSIS OD2)

In addition, 11 graphs were prepared showing the variation of total cost and major cost center subtotals (on a per acre basis) with the variable site and cost parameters selected. These are presented in the following section of this report, which also contains a brief description, and discussion, of the results of the cost analyses carried out

It is intended that this information will provide a useful guide for assessing the cost feasibility of future AML blasting projects of this type. It is felt that the use of a spreadsheet based model, employing a readily available commercial software package, with which many are familiar, is appropriate for this type of work. This analysis can by no means be regarded as exhaustive, however. The number of permutations of site parameters and cost parameters is obviously very large. Nevertheless, it should provide the type of "ball-park" figures which are so often required, but so seldom available, for feasibility studies.

9.4.2. Results of Analysis using Cost Model

9.4.2.1. Variable Overburden Depth

Depths varying from 35 to 66 feet (those which were encountered during 1986 testwork) were considered. Three actual sets of data were produced:

- for single row blasting of a 12 foot wide room (OD1)
- for double row blasting of a 20-foot wide room (OD2)
- for single row blasting of a 20 foot wide room (OD3)

Analysis OD3 was carried out to illustrate the effect of use of a single row of blastholes, as opposed to two rows, for a wider room, with very significant results.

Results are presented in full tabular form in Appendix G-1. They are summarized in graphical form in Figs. 9-6, 9-7 and 9-8. Graphical results indicate clearly two "breaks" in the ascending trend of total cost with drilling depth. These occur at 41 feet and 60 feet overburden depths, which correspond to a change from 3 to 4 decks, and from 4 to 5 explosive decks respectively.

For three deck blasting using a single row (analysis OD1) the total cost per acre increases by about $235 per foot of additional overburden in the range 35 to 40 feet. For four deck blasting this increase is about $315 per foot in the 41-59 feet overburden range. This increases to an extra $431 per foot of overburden when five decks must be used in the range 61-66 feet. The corresponding increases for "base case" double row blasting

FIG. 9-6 : AML BLASTING - COST MODEL

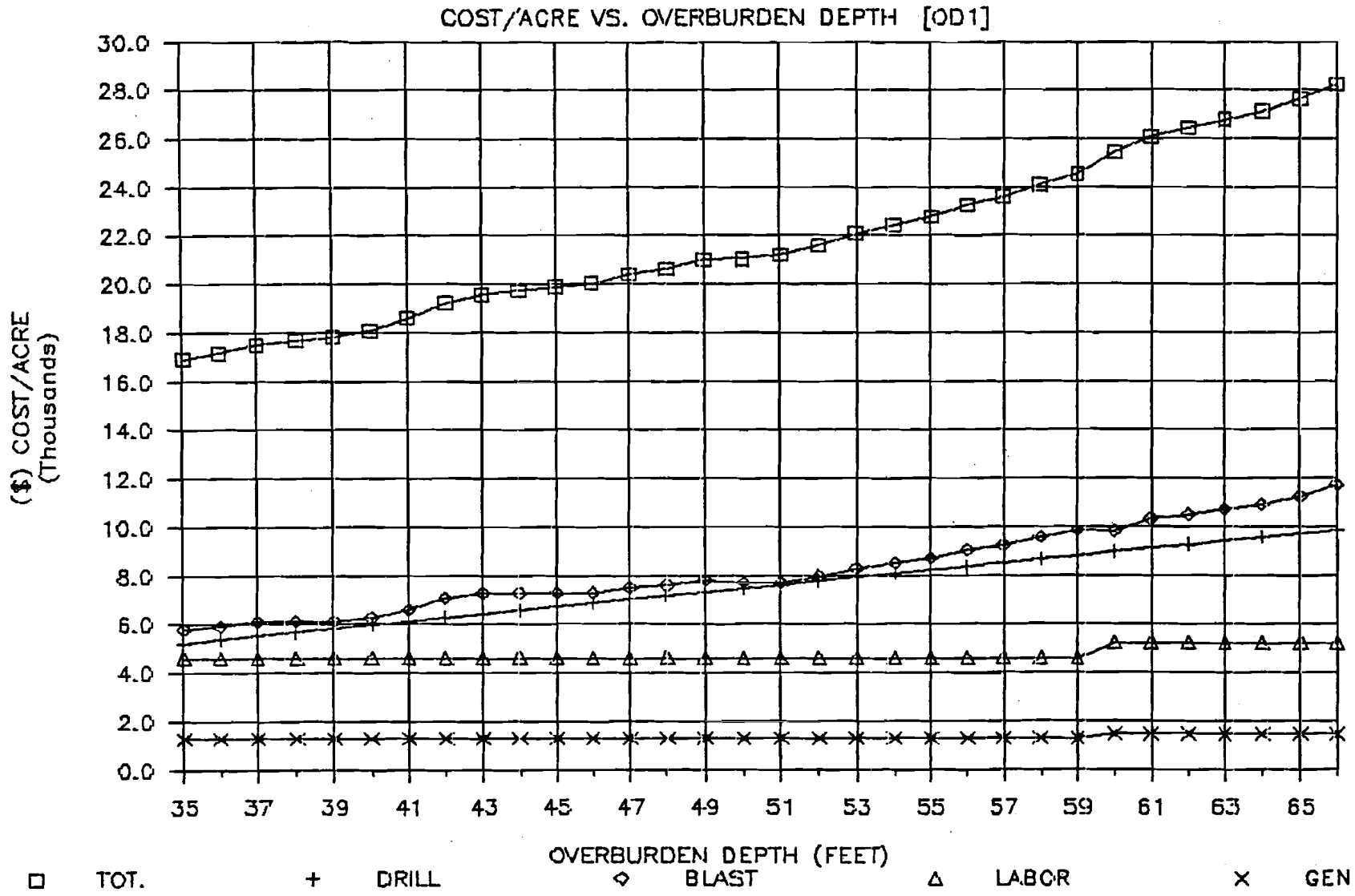


FIG. 9-7 : AML BLASTING - COST MODEL

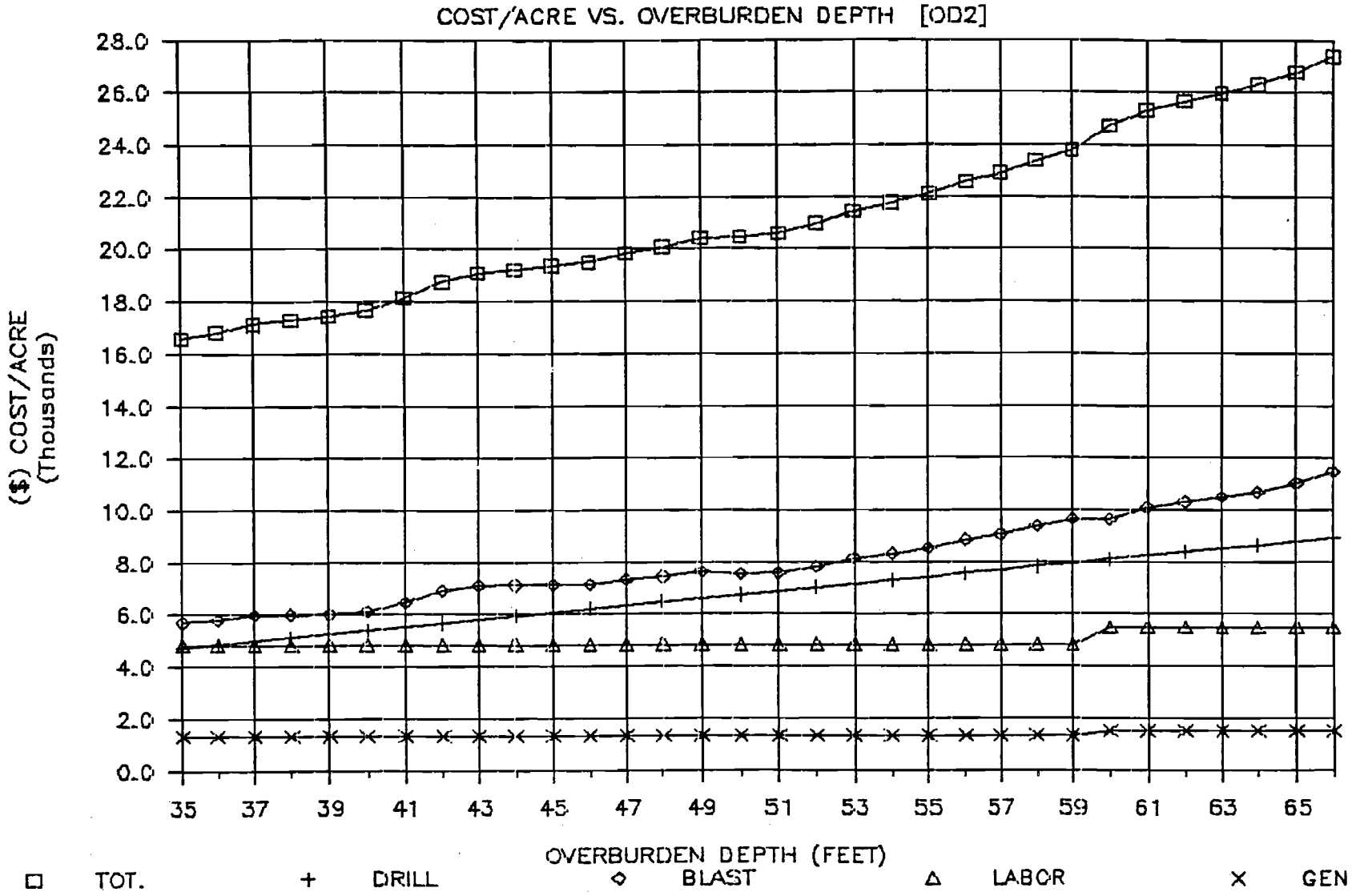
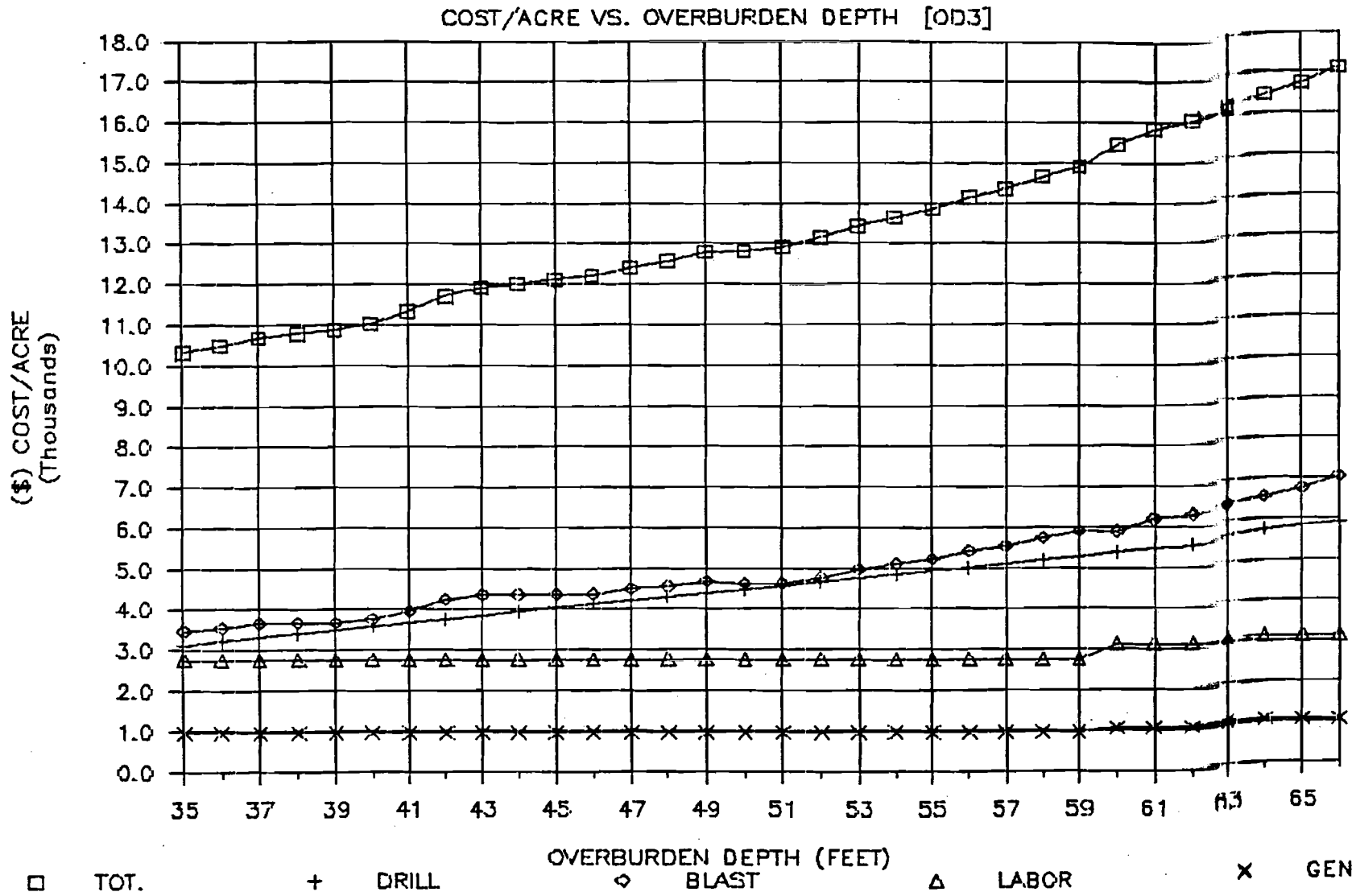


FIG. 9-8 : AML BLASTING - COST MODEL



over the 3, 4 and 5 deck overburden ranges are $219, $297 and $412 per foot, per acre, respectively. These are slightly lower than for the single row case.

Both drilling and explosives costs per acre increase directly with overburden depth. Labor costs remain constant until the requirement for five decks causes the Cost Model to recognize the extra crewman needed for loading operations.

It can be seen that the results from this analysis do not vary significantly between single row blasts in a 12 foot room, and double row blasts in a 20-foot wide room (analyses OD1 and OD2 respectively) when the extraction from a site has been constant at 40%. This is because the areas of influence for an actual blast (room width times blast length) are very similar (4200 sq. ft. and 4000 sq. ft. for single and double row blasts respectively).

However, if single row blasting can be successfully employed in the wider room, the savings are appreciable (compare analyses OD2 and OD3). For example, at 50 feet overburden, savings of about 37% of the total cost per acre can be made if a single row of blastholes can be substituted for a double row in a 20 foot wide room. The reason for this is that the area of influence increases for the single row case, as a longer blast may be taken for the same approximate number of blastholes. The increase in area of influence is about 75% when a single row can be employed; the number of blasts required in a ten acre site is reduced from 44 to 25.

9.4.2.2. Variable Drilling Cost

The results of varying the drilling cost for 6 inch diameter blastholes from $0.50 - $2.50 per foot are presented in Appendix G-2. Results from two of these analyses, for 55 feet of overburden using single and double row blasting respectively, are shown in graphical form in Figs. 9-9 and 9-10.

It can be seen from analyses DC3 and DC7 that the total cost per acre almost doubles over this range of drilling costs. This confirms observations made so far in this study that drilling cost is a very important control on overall AML blasting costs. The increase in total AML blasting cost per acre is $782 for each 10 cent increase in drilling cost for single row blasting and is $707 per 10 cent increase in the double row case.

9.4.2.3. Variable Explosives Cost

For a situation where ANFO and slurry are combined in a typical blast, the cost of these explosives was varied according to the following:

FIG. 9-9 : AML BLASTING - COST MODEL

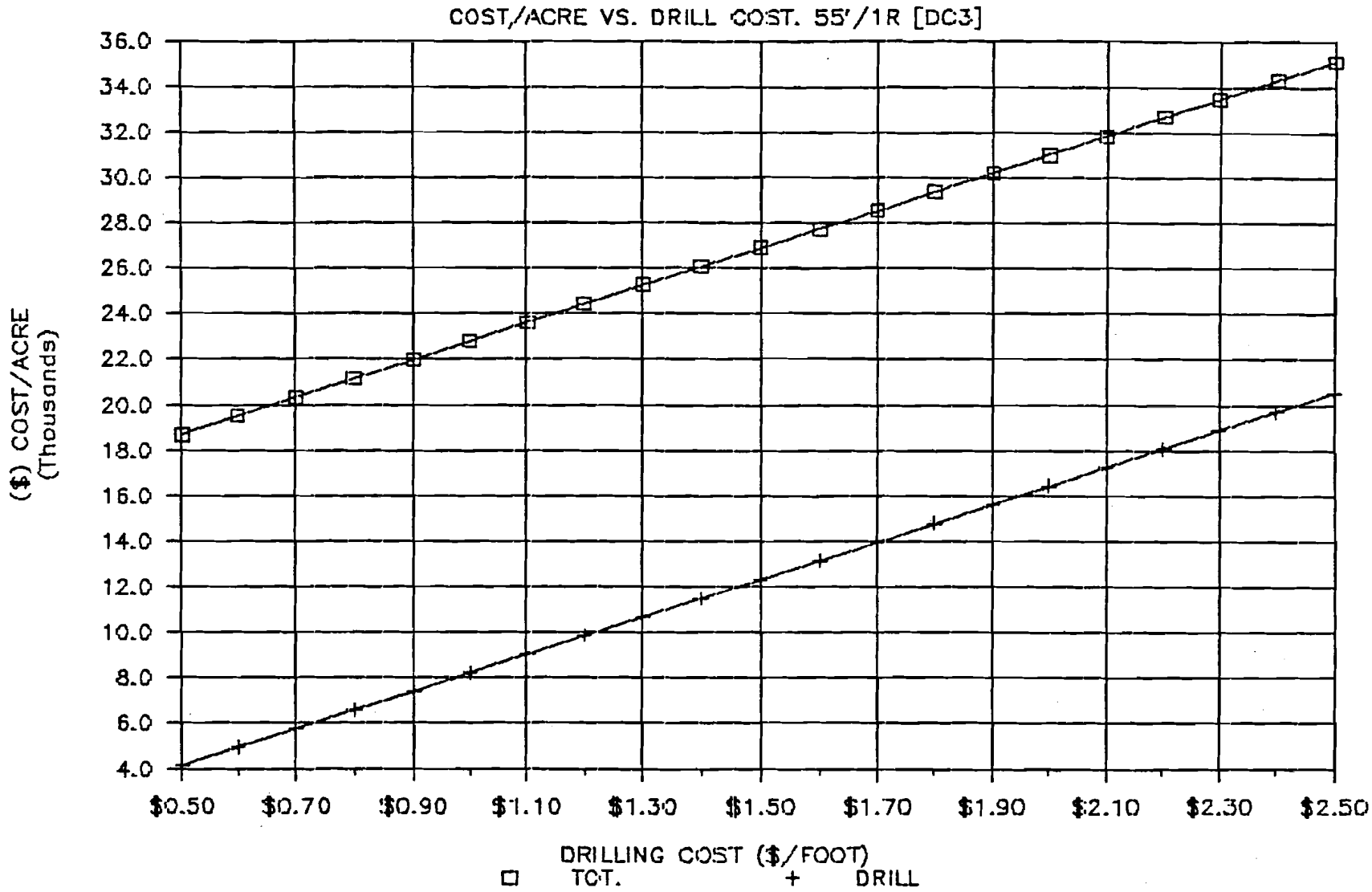
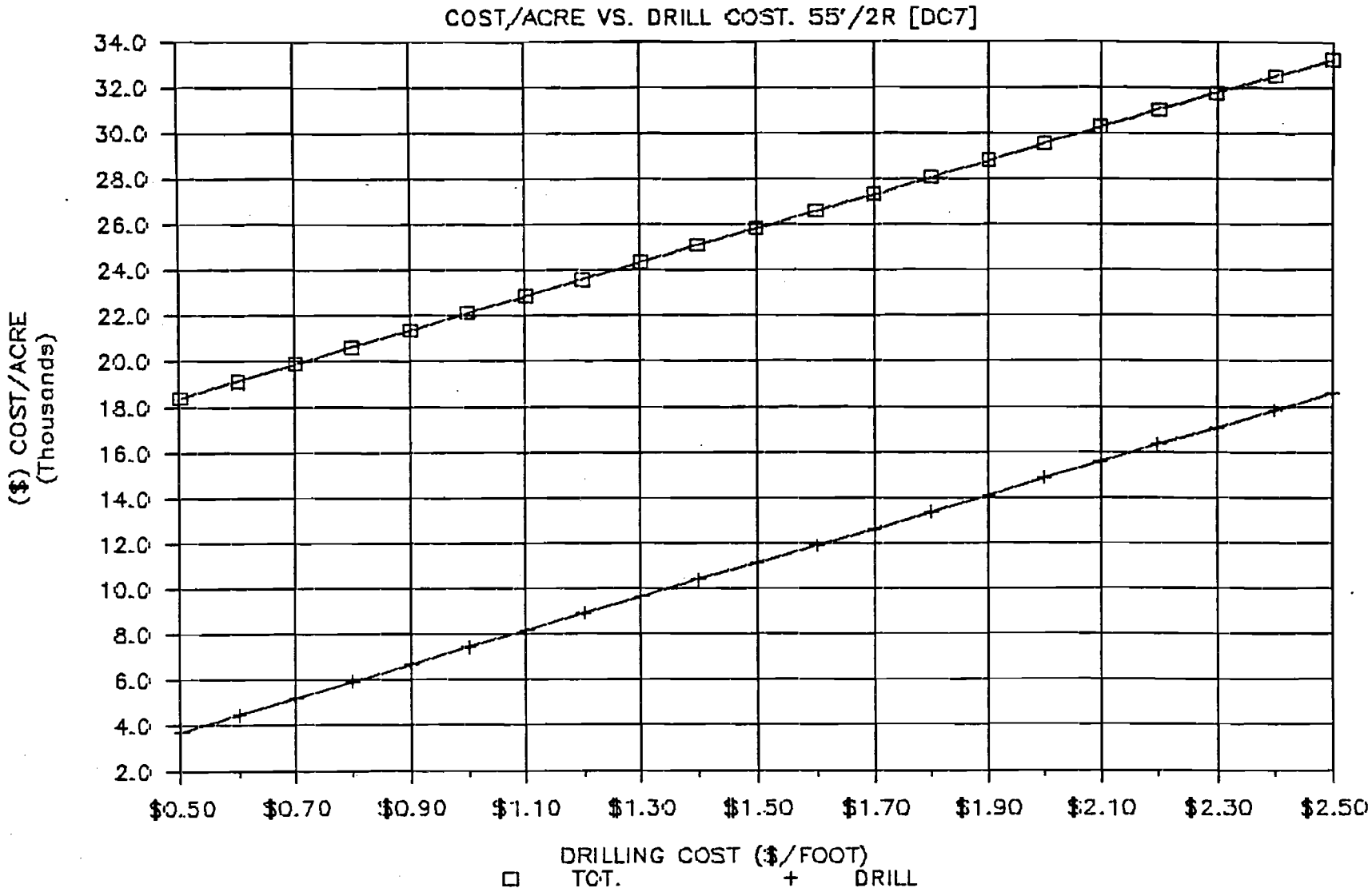


FIG. 9-10 : AML BLASTING - COST MODEL



- ANFO cost from 10 cents/lb to 23 cents/lb
- slurry cost from 35 cents/lb to 48 cents/lb.

It is unlikely that ANFO costs of less than 15 cents per pound will be experienced unless one is dealing with a bulk product. Indeed, the likelihood of obtaining bagged ANFO for smaller sites is decreasing, according to a local explosives supplier, due to the very low demand and handling difficulties experienced with this product.

The range of slurry costs represents, more than anything, the economies of scale that can be made with a bagged product, and the interrelationship of manufactured batch size with transport costs associated with different consignment sizes.

Tables contained in Appendix G-3 illustrate that the overall cost of AML blasting is rather less sensitive to explosives cost than was the case for drilling cost. Some typical graphical relationships are shown for 55 feet overburden in single and double row cases in Figs. 9-11 and 9-12 respectively.

Although these graphs show only variable ANFO cost on the horizontal axis, the trends include the variation of slurry cost as well. In 55 feet of overburden for example (analyses EC3 and EC7 for single and double row blasts respectively) the increase in total cost per acre over this range of explosives costs was only about 25%. For single row blasting at 55 feet overburden depth there was an increased total blasting cost per acre of $235 per one cent increase in both ANFO and slurry cost. The corresponding increase for double row blasting was $230 per one cent increase per acre.

9.4.2.4. Variable Labor Cost

Results of the analysis of variable labor costs on overall AML blasting cost are contained in Appendix G-4. This analysis was carried out using a simple percentage of the base case labor costs used for other analyses. The range considered was plus or minus 15% of the base case costs of $30.00/hr (blasting engineer), $20.00/hr (field engineer) and $10.00/hr (laborer).

Graphical results from the analysis of variable labor cost for 55 feet of overburden in single and double row blasting (analyses LC3 and LC7) are presented in Figs. 9-13 and 9-14 respectively.

A five percent variation in labor cost produces a total cost per acre variation of about $230 in single row blasting, and $242 per acre when double row blasting is used for 55 feet of overburden. The impact of a 15% variation in labor cost was

FIG. 9-11 : AML BLASTING - COST MODEL

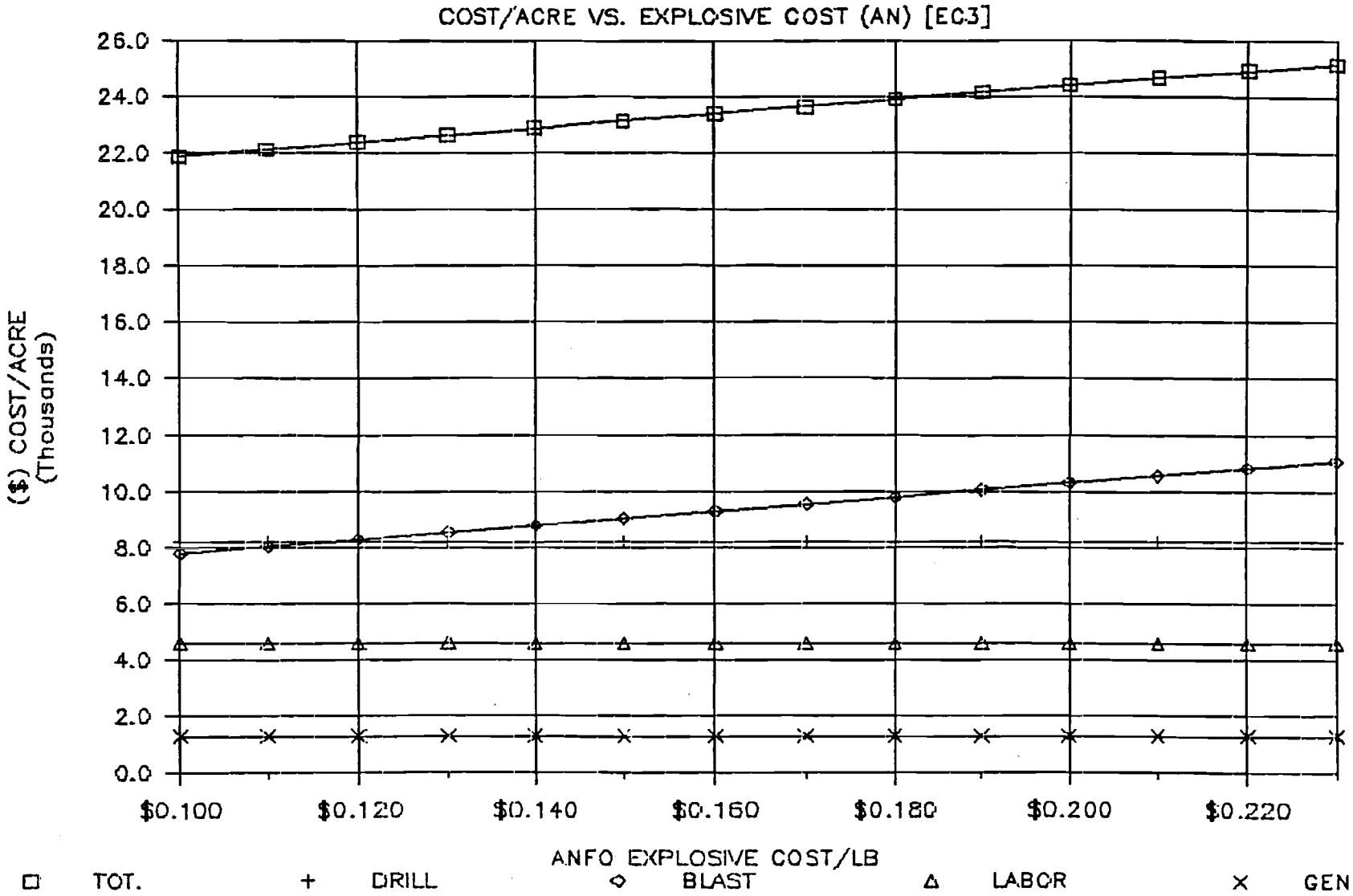


FIG. 9-12 : AML BLASTING - COST MODEL

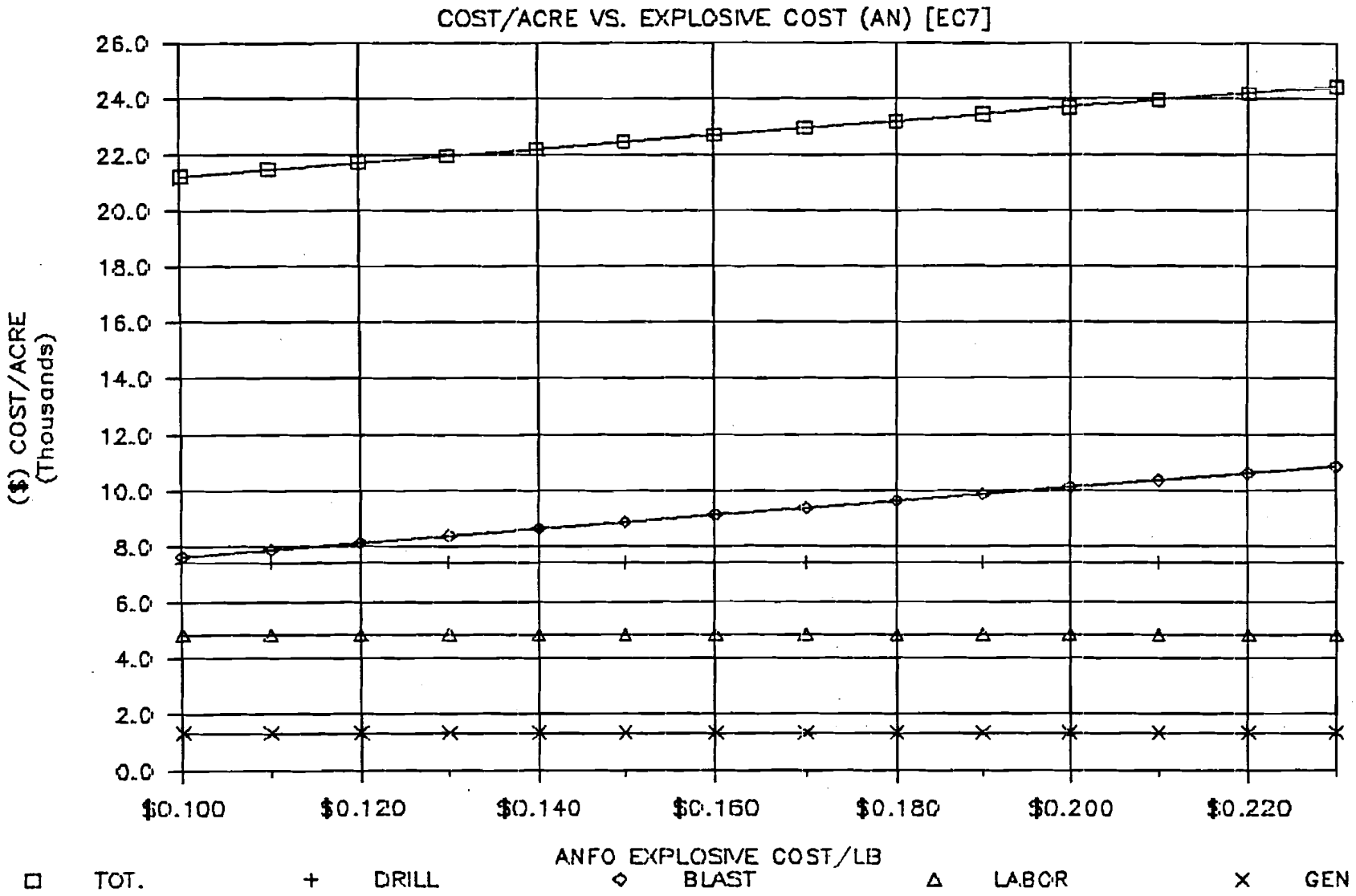


FIG. 9-13 : AML BLASTING - COST MODEL

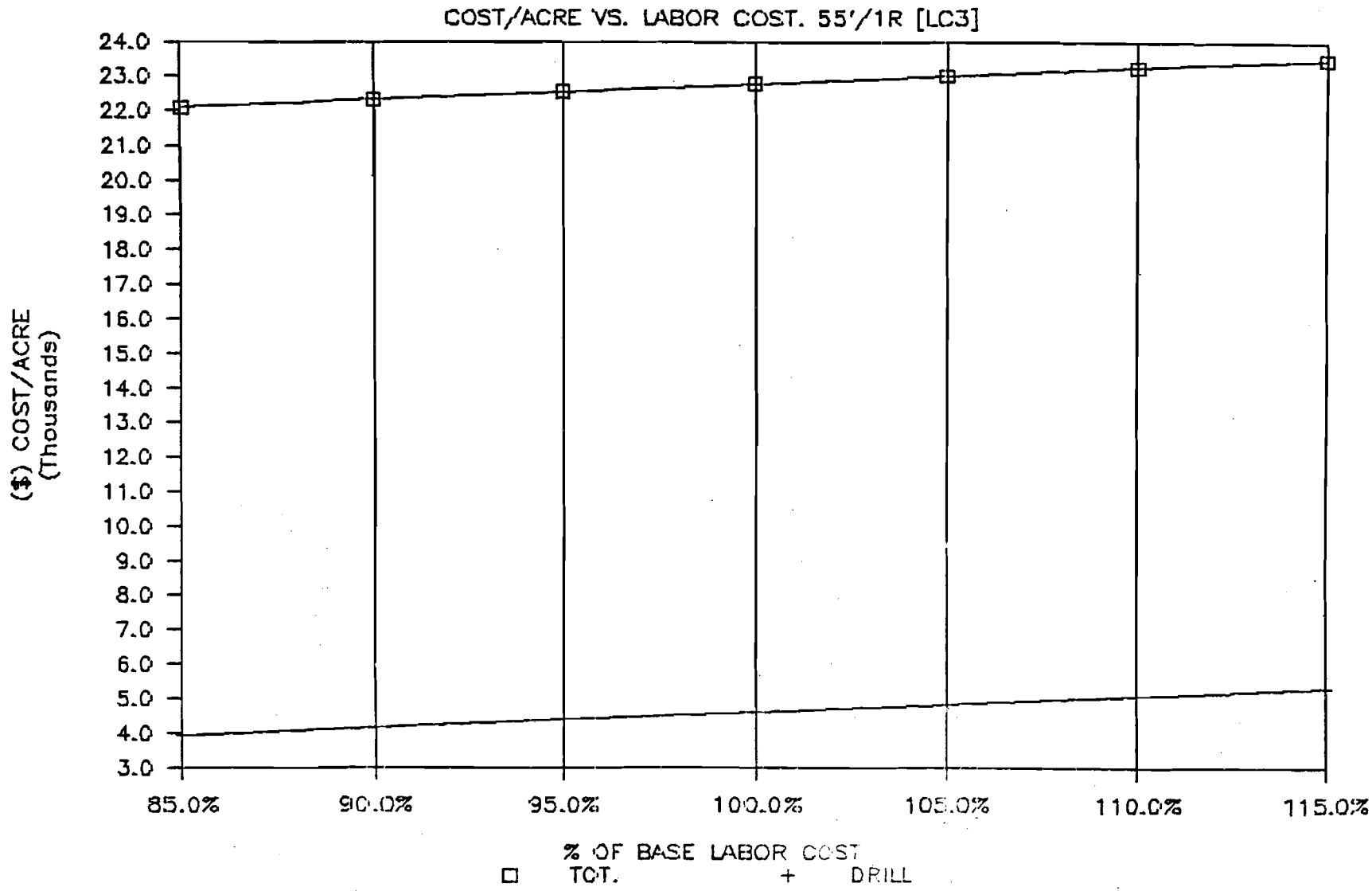
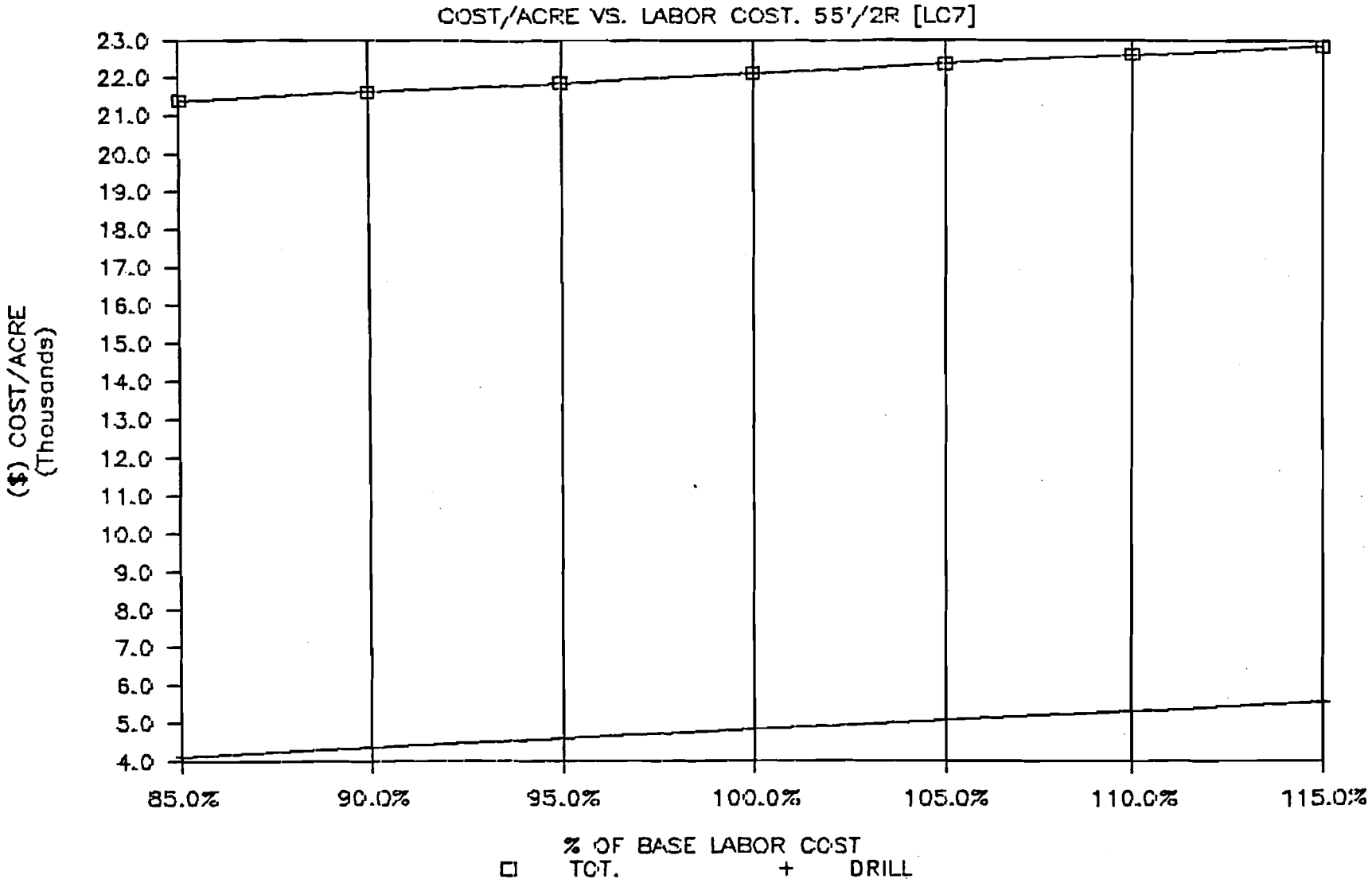


FIG. 9-14 : AML BLASTING - COST MODEL



reduced to only about a 3% variation in overall AML blasting cost per acre.

9.4.2.5. Variable Extraction from Site

As was expected, based on previous discussion and observed results, the analysis of AML blasting cost with percentage extraction from a site proved to be very significant. Since all blasting costs are pro-rated to a cost per acre, it is obvious that a higher extraction rate, which involves more blasts in an acre, will lead to a correspondingly higher total blasting cost.

The extraction ranges considered were as follows :

Single row blasting in 12 foot wide rooms:

- 20% (48' wide pillars) to 60% (8' wide pillars)

Double row blasting in 20 foot wide rooms:

- 20% (80' wide pillars) to 60% (13' wide pillars)

Results of this analysis are presented in Appendix G-5 of this report. Figs. 9-15 and 9-16 show graphically the results of analyses PE3 and PE7, for single and double row blasting respectively in 55 feet of overburden. There is observed a very wide variation in total blasting cost per acre, from $12,000/acre to $34,000/acre in single row blasting, and from $11,000/acre to $33,000/acre in the double row case.

Fig. 9-17 is presented to illustrate three different cases which are also included in the overall analysis. As was illustrated earlier, there is no saving made by using single row blasting unless this is achieved at a wider room width. In the upper part of Fig. 9-17, for example, it can be seen that for 20 foot wide rooms at 40% extraction (by area) there is a reduction in the number of blasts per acre from 4 to 2.5 if a single row of blastholes can be successfully employed.

If, however, a single row of blastholes is used to collapse 12 foot wide rooms, and the extraction percentage was still 40%, it is 66% more expensive than single row blasting in 20 foot wide rooms. This is because there are more blasts to be taken per acre for the narrow rooms which are, at the same percentage extraction, more closely spaced.

In the examples presented in Fig. 9-17 it can be seen that as the percent extraction increases from 40% to 60% for a 20 foot wide room this is due to a pillar width reduction from 30 feet to 13 feet. The number of double row blasts per acre for

FIG. 9-15 : AML BLASTING - COST MODEL

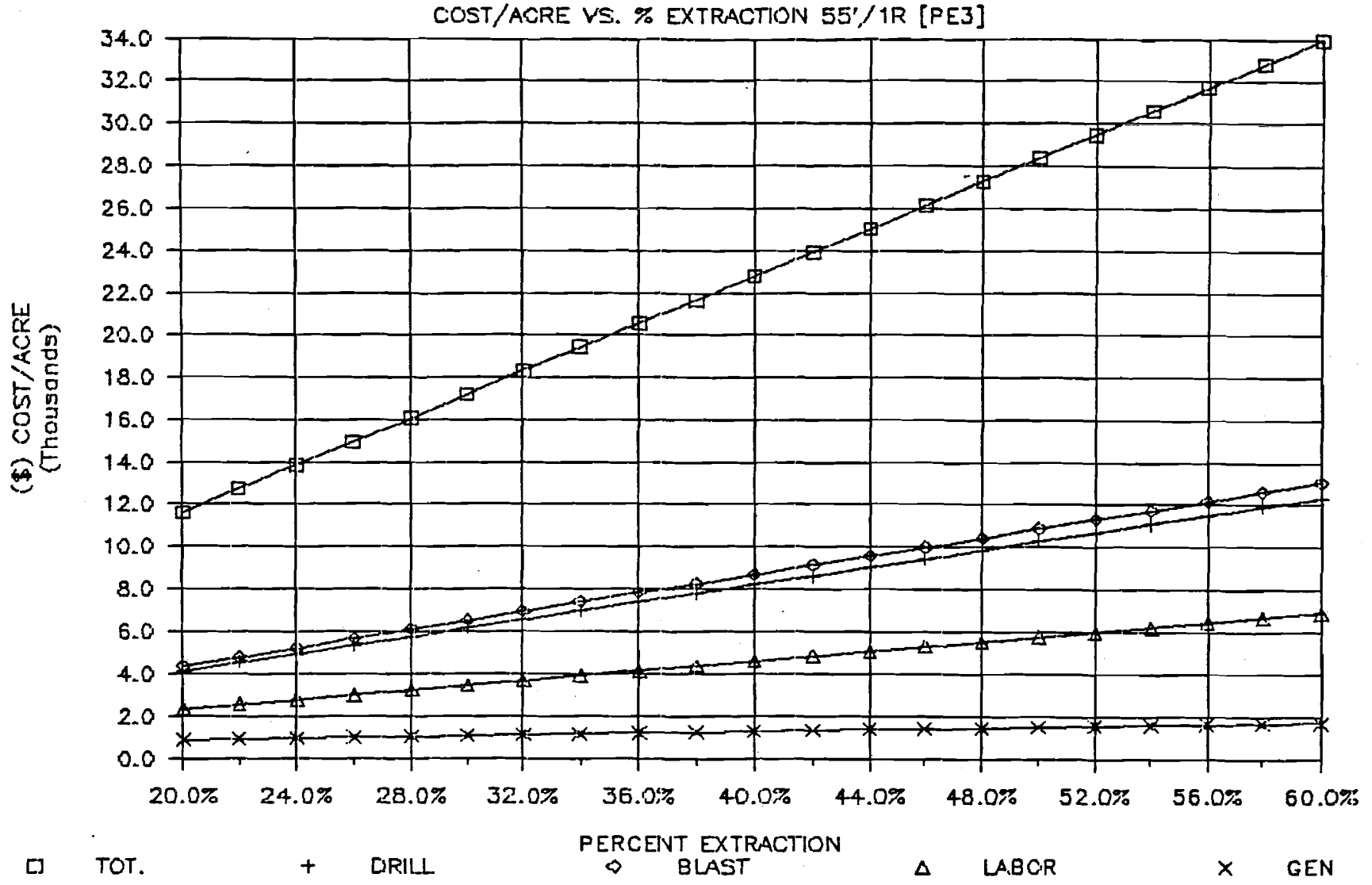
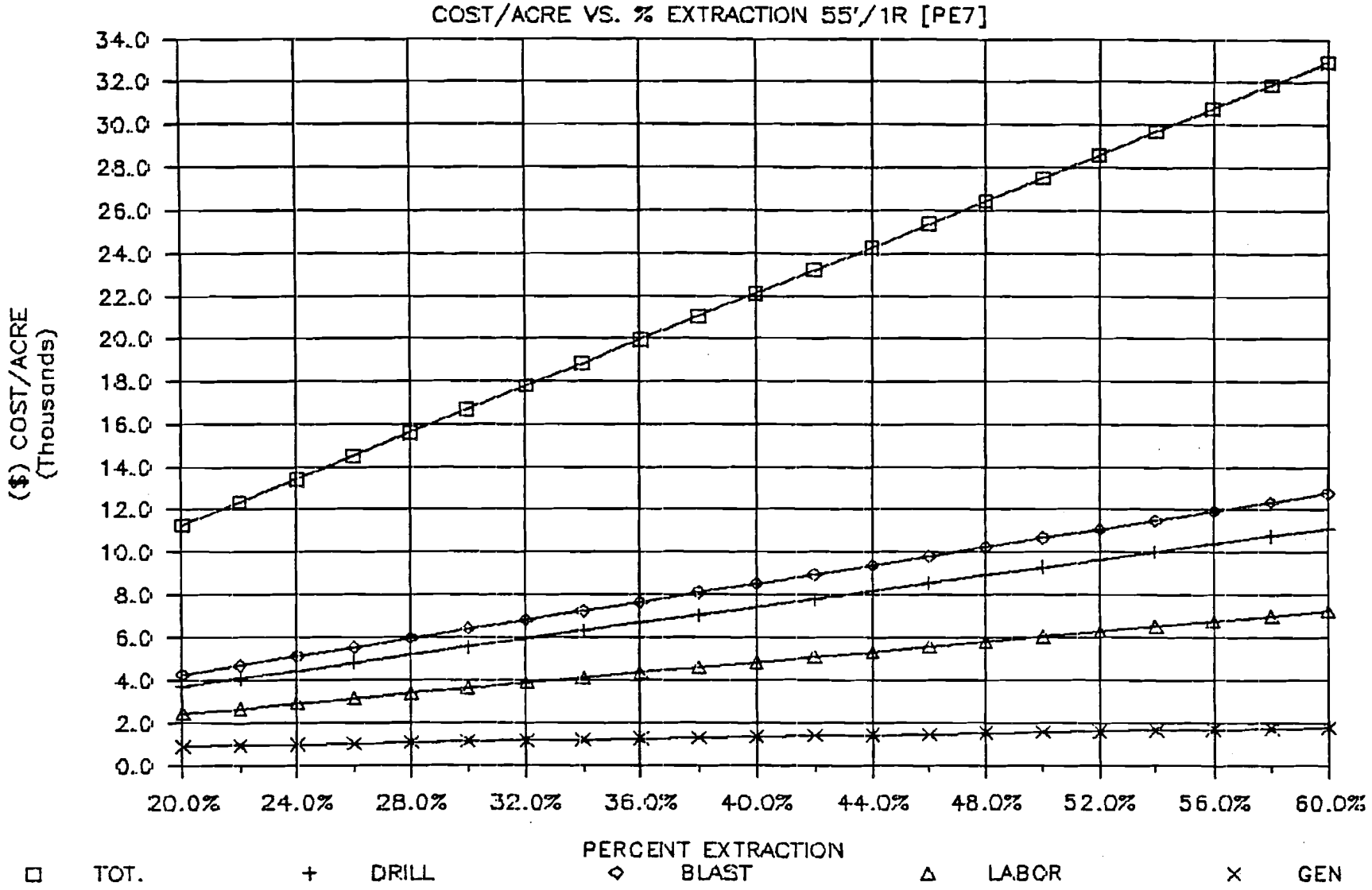
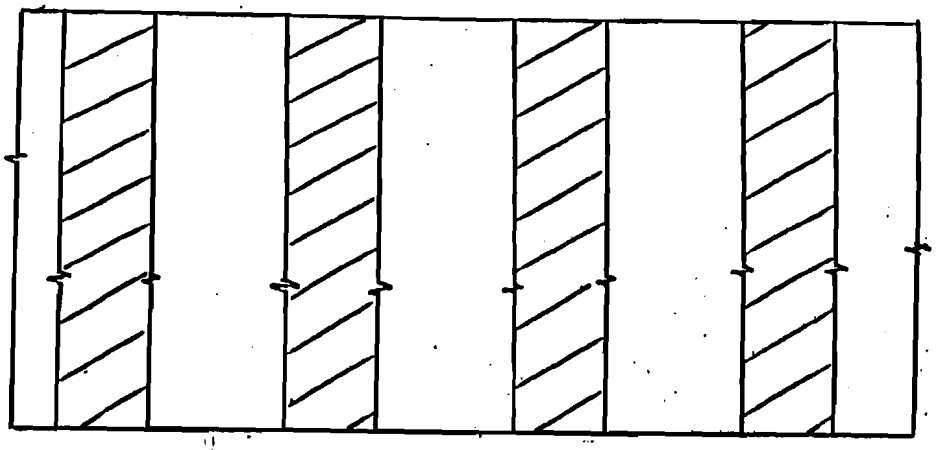


FIG. 9-16 : AML BLASTING - COST MODEL





20' wide rooms, 30' pillars
40% extraction : 0.40 x 43560 = 17,424 sq. ft./acre
DOUBLE ROW : 17424 ÷ 220 ÷ 20 = 3.96 blasts/acre = $19,800/acre
SINGLE ROW : 17424 ÷ 350 ÷ 20 = 2.49 blasts/acre = $12,500/acre

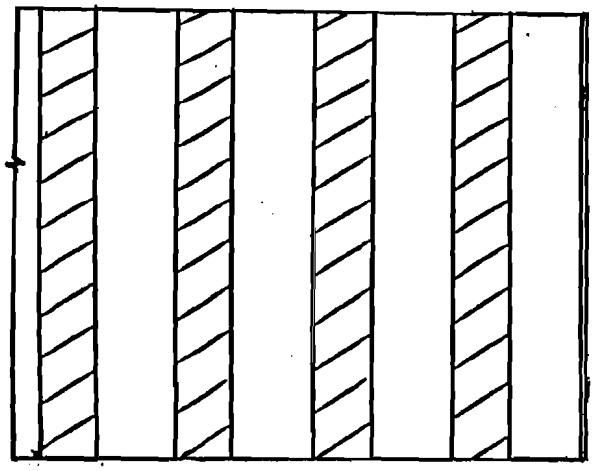
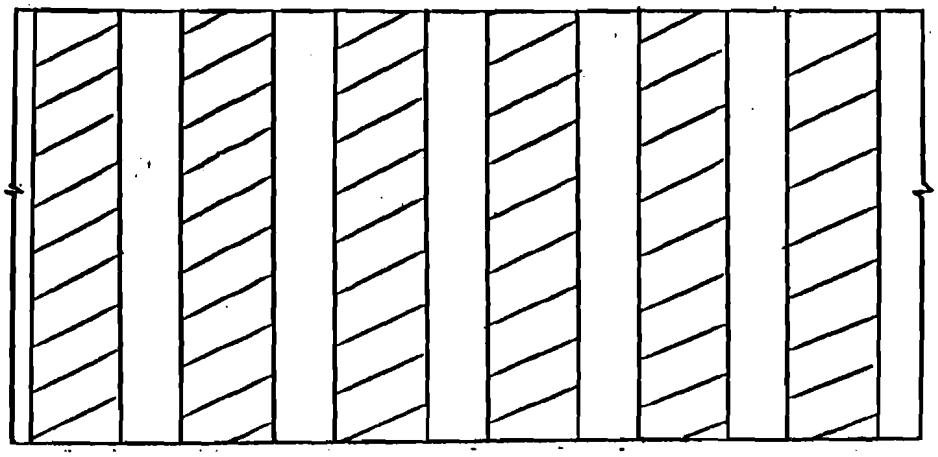


FIG. 9-17: ILLUSTRATING RELATIONSHIP BETWEEN ROOM WIDTH, PERCENT EXTRACTION AND BLASTING COST PER ACRE FOR 40% AND 60% EXTRACTION

12' wide rooms, 18' wide pillars
40% extraction : 0.40 x 43560 = 17,424 sq. ft./acre
SINGLE ROW : 17424 ÷ 350 ÷ 12 = 4.15 blasts/acre = $20,800/acre



20' wide rooms, 13' pillars
60% extraction : 0.60 x 43560 = 26,136 sq. ft./acre
DOUBLE ROW : 26136 ÷ 220 ÷ 20 = 5.94 blasts/acre = $29,700/acre
SINGLE ROW : 26136 ÷ 350 ÷ 20 = 3.73 blasts/acre = $18,700/acre

this configuration increases from 4 to 6, and the number of single row blasts goes from 2.5 to 3.7.

In analyses PE3 and PE7 (Figs. 9-15 and 9-16 respectively) the total cost per acre increases by about $5500 for each 10 percent increase in extraction for single row, and by about $5400/acre per 10% extraction for double row blasting situations. These increases are an order of magnitude higher than those which were encountered for incremental changes in the other variable site and cost parameters studied.

9.4.2.6. Variable Site Size

The only cost factors which are not independent of site size are the general project costs. Since these only represent a small percentage of the cost of a typical blast, the effect of variable site size on overall cost per acre is negligible. Trial runs were made using the Cost Model to evaluate this; results have not been documented.

9.5. COST COMPARISON WITH OTHER AML RECLAMATION METHODS

It was apparent from results obtained using the Cost Model that the cost of reclaiming abandoned mine land using blasting can be extremely variable. The most significant factor in determining a per-acre cost was the mining extraction from the site. Other important controls were depth of overburden and drilling and explosives costs. For complete reclamation of a blasted AML site it is probable that fill material would have to be brought in to restore the land surface for agricultural or other commercial use.

The amount of actual mining carried out at a site will have a very strong influence on reclamation cost whatever the technique employed. Drilling and explosives costs will obviously not be applicable factors in other methods. In this discussion main attention will be focussed on a common and simple method of AML reclamation - fill of existing sinkholes. This can be by dozer work at the site, or by trucking in fill from elsewhere. In either case, the cost per acre will be determined by the volume of sinkholes requiring fill.

The cost of filling will be determined by the depth, size and number of sinkholes. It was shown in Chapter 8 that for a given height of underground development, the depth of the resulting sinkhole may decrease with an increase in overburden depth. Therefore, deeper overburden may actually reduce the cost of filling sinkholes, though it will always increase the cost of blasting. The size (width) of the sinkholes will be controlled by the width of the rooms, and the number of these will depend on the spacing of underground development.

It is shown in Fig. 9-17 that at a 40% mining extraction the area of undermined land could be 17,450 sq.ft./acre if rooms were wide, and about 5% higher (18,300) if the rooms were narrow. If sinkholes are of fairly uniform depth, therefore, there will not be a great difference in work, and therefore in cost, involved in filling these, regardless of development width or the number of sinkholes.

It was established in Fig. 9-17 that the typical cost of a single or double row blast in AML work may be around $5000. It was also shown that for a similar overall area of undermined ground and the same percentage extraction in mining, development width was very critical in controlling blasting cost per acre. The cost of reclaiming the narrower rooms was over $30,000 per acre. The wider rooms could be blasted at about half this cost, and for as low as $13,000 per acre if a single line of blastholes could be employed.

It is very important to appreciate the above points before a reasonable comparison can be made between the cost of different AML reclamation methods.

Table 9-10 shows the typical cost of some different reclamation techniques that have been applied in North Dakota. For most reclamation methods the costs are very dependent on site characteristics, and can vary very significantly when similar techniques are applied to different sites. Thus it is extremely difficult to compare cost of different reclamation techniques without specific site information.

Unfortunately, the specific information required to make a meaningful analysis of Table 9-10 is not available. It was compiled from historic data, and in most cases site characteristics were not recorded. The most reliable information is probably that related to the rock or earth fill of individual openings and sinkholes, since much of this work has been carried out in North Dakota over the last 10 years.

Remote backfill was applied in one known case where sinkholes opened up in a trailer park. Obviously the use of blasting was impossible, and the problem could not be solved simply by filling in the holes that appeared. In this case a hydraulic fill, mixed with cement, was pumped underground in an attempt to fill the dangerously undermined area. The cost of this was extremely high, and it is not certain where all the fill went. Much of it may have flowed a considerable distance from the actual hazard site through panel entries and roadways.

Daylighting, using a small dragline or hydraulic shovel, or a back-hoe, appears to be a method comparable to blasting in that total elimination of hazardous areas of potential caving is possible. The cost would also appear to be comparable. The danger to men and equipment is a serious consideration, however. Like blasting, this method would also require fill for complete

TABLE 9-10

RECLAMATION TECHNIQUES AND RELATED COST

RECLAMATION TECHNIQUE	PROBLEMS	PER ACRE COSTS
Complete Reclamation		
Underground Mines.....	VO,P,S,IRW	
1. daylighting-without coal removal		
shallow mine.....		$ 10,000 - 20,000
deep mine.....		100,000 - 200,000
2. remote backfill.....		59,000 - 100,000
3. dynamic consolidation		
shallow mines.....		10,000 - 70,000
4. blasting.....		29,000 - 40,000
Reclaiming Individual Hazard by:		
1. Using rock or earth fill.....	VO,P,S	< $1,000
2. Blasting and/or Cement Blockage	P	3,000 - 5,000
3. Backsloping.....	DH	10,000 - 50,000
4. Fence Off Hazard.....	VO,P,DH,PWAI S,HEF,IRW,HWB	< $1,000

FOOTNOTES:

VO - Vertical Opening

P - Portal

S - Subsidence Prone Area

IRW - Unauthorized and Dangerous Disposal of Industrial or Residential Waste

DH - Dangerous Highwall

PWAI - Polluted Agricultural/Industrial Water Resource

HEF - Hazardous Abandoned Mining Equipment or Facilities

HWB - Unauthorized and Dangerous Use of a Water Body for Recreational Purposes

reclamation of the site.

Backsloping represents a potentially useful AML method if natural collapse has completely occurred and a land profile with gentle depressions is acceptable. The cost would again appear to be comparable to blasting followed by some grading work in blast-induced sinkholes.

The data for blasting related AML work in Table 9-10 refers almost exclusively to some work carried out by the North Dakota Public Service Commission near New Leipzig. Unfortunately, the type, size and extent of underground workings was not documented. The area was part of an existing landfill. The $30-40,000 cost per acre cannot, therefore, be compared specifically with those generated in the test blast program or from analysis with the Cost Model.

The cost associated with blast #20, employed to fill an individual sinkhole during the test blast program, was about $1200. While this exhibited excellent technical feasibility, the cost would appear to be prohibitive when compared to typical costs per acre using trucked-in earth or rock to fill sinkholes. This method may have some application in an emergency situation where the immediate closure of isolated subsidence features was required. Also, if a cheap source of fill is not available then blasting may be a good alternative.

The use of fill provides only a remedy for the existing problem. It is hard to predict how many times the same area may need to be filled before all subsidence has stopped. Therefore the cost is uncertain. In one case of which we are aware new holes opened up within weeks of the completion of an initial fill reclamation.

10. ENVIRONMENTAL ASPECTS OF AML BLASTING

10.1. INTRODUCTION

The purpose of this chapter is to briefly consider the environmental aspects of AML blasting work, based on observations from the test site used for the research project. A brief comparison will be made between blasting and other AML methods in this respect. Some brief comments on the potential land use for blasted AML land will be made.

An AML site where numerous sinkholes have occurred will have virtually no commercial use. Fig. 10-1 is a photograph of a typical site where subsidence is well-advanced. It is waste land, and represents a danger to the public. Some of the sinkholes contain water. The greatest danger from such sites, however, is the presence of potential sinkholes which are about to break through to surface. A likely mechanism for sinkhole formation was described in Chapter 8, with illustration of a newly formed sinkhole.

Blasting is a reclamation method that ensures that there will be no additional or unexpected subsidence once reclamation has occurred. If a site is simply reclaimed by filling existing sinkholes, there will exist the danger that additional ones will develop later. This is probably the greatest hazard of all that could exist at an AML site. A field reclaimed and replanted may not look like abandoned mine land at all. If the land is for public use, people and vehicles run the risk of falling into a structure that has caved to within inches of the surface since the initial reclamation effort. If the land-use is agricultural, there is risk to men and machinery even if the land is known by the users to be formerly undermined.

10.2. PRE-BLAST LAND USE

The Beulah site used for the testwork consisted of approximately 10 acres of North Dakota Game & Fish Management land. The surface vegetation consisted of tall grass (brome) with small interspersed clumps of shrubs and forbes. The site is easily accessible and open to the public. The primary use of this area was for public hunting, especially during pheasant and deer-hunting seasons.

Prior to the research project the area was considered extremely hazardous due to the presence of vertical openings. The site had been posted with warning signs, and the North Dakota Public Service Commission had contracted out several reclamation jobs on adjacent sites. This work consisted mainly of grading and filling existing sinkholes, followed by revegetation.

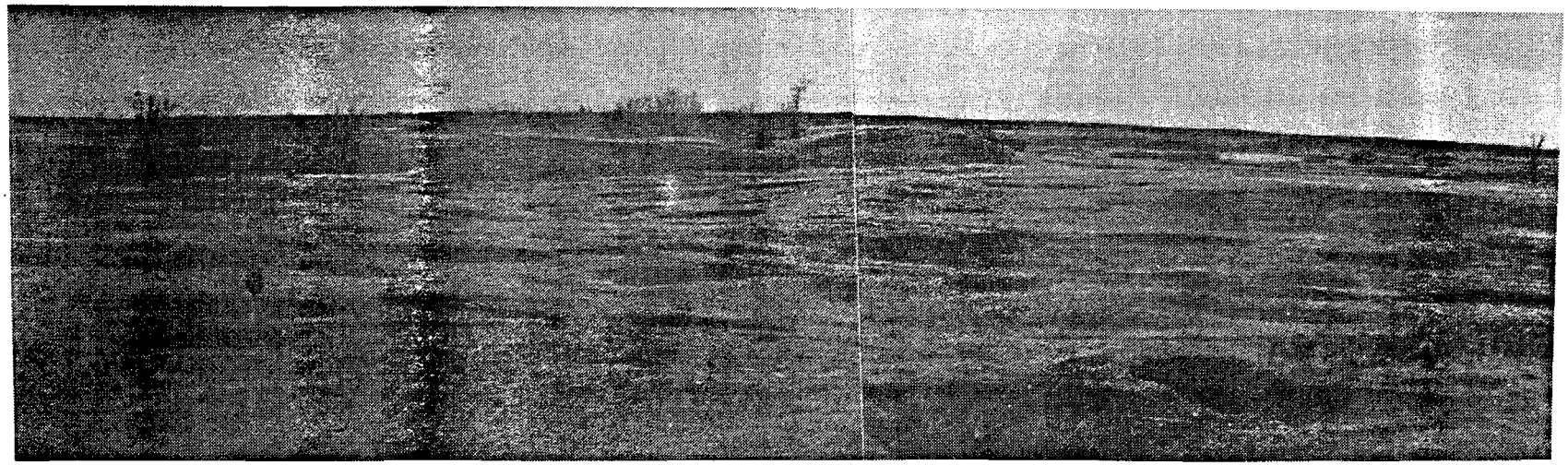


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FIG. 10-1: TYPICAL ABANDONED MINE LAND SHOWING ADVANCED STAGE OF SINKHOLE DEVELOPMENT

A general idea of the situation at the site can be obtained from Figs. 10-2 and 10-3. The first of these shows a near circular sinkhole that developed at the intersection of 12-foot wide access development. It is a gently sloping basin-like depression, and represents a fairly "mature" stage of sinkhole formation. This is backed up by the presence of vegetation in the depression, except for around its rim where presumably some minor slumping had occurred fairly recently.

Fig. 10-3 shows an elongated V-shaped depression which lies above, and is aligned with, one of the rooms in the southern part of the test site. Again the slopes are fairly gentle, and there has been considerable revegetation. It is possible that the small trees were present before subsidence occurred, and continued to grow as the land surface dropped. Both Figs. 10-2 and 10-3 were taken in the fall, and indicate the length to which the grass grew at the site during the summer months.

10.3. POST-BLAST LAND USE

Most of the blasting work carried out at the test site resulted in the formation of shallow surface depressions. In a few cases there was a few feet of surface heave at the ends of blasts, and in others this extended over part or all of the blast length. There was relatively little, if any, disturbance of topsoil in the former case, and none in the latter. Some typical post-blast profiles, photographed about six months after blasting work was completed, are illustrated in Chapter 4 of this report.

On completion of the program, observations by ourselves and others confirmed that, over the majority of the area, the overall wildlife habitat had been enhanced by the work. Following winter snowfall and the resulting spring melt, many areas which formerly had fairly steep slopes had been graded by natural cracking, slewing and water action. In some depressions ponded water was found.

By late spring/early summer vegetation had begun to establish itself. This was particularly the case for the vertical -sided sinkhole which was closed by blasting methods. This was also true for most of the depressions created. Fig. 10-4 illustrates a blast-created sinkhole immediately after blasting. The sides are near vertical at the rim, but darker topsoil can be clearly be seen in the depression.

Fig. 10-5 shows the same feature, photographed seven months later from further back, which indicates that there has been some spillage of material into the depression from its rim, reducing the overall slope of the sides. New grass is growing on chunks of topsoil which still contained roots, even in one case where the growing surface is in fact vertical. The part of the blast showing surface heave, which can be seen in the foreground

FIG. 10-2: TYPICAL INDIVIDUAL SINKHOLE WITH NEAR-CIRCULAR SHAPE SUCH AS MAY TYPICALLY FORM AT INTERSECTIONS OF UNDERGROUND DEVELOPMENT

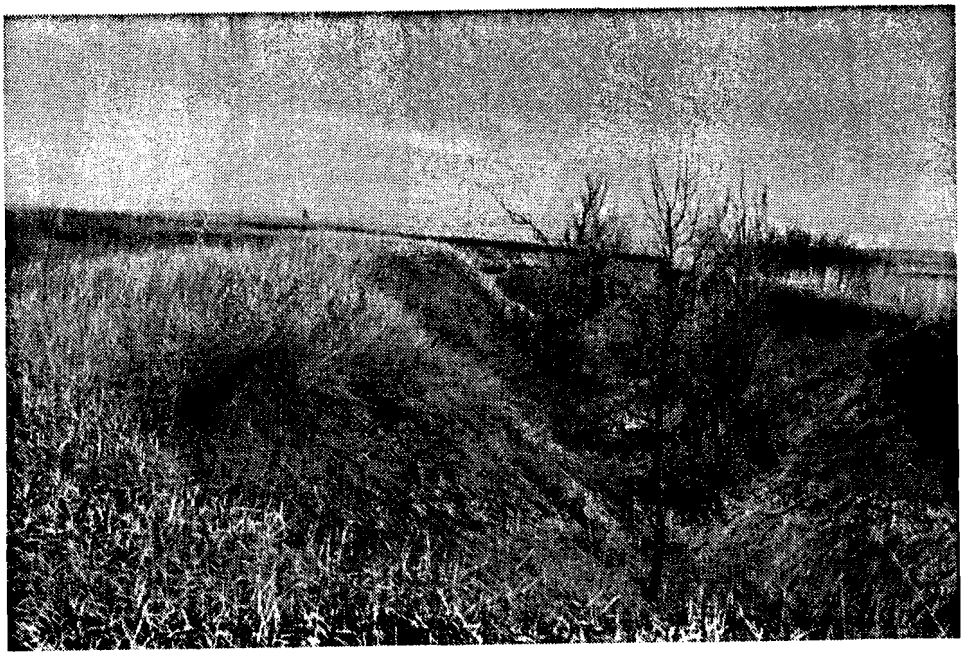

FIG. 10-3: TYPICAL "V-PROFILE" OF ELONGATED SINKHOLE ALIGNED WITH AN UNDERGROUND ROOM

FIG. 10-4: VIEW OF BLAST #1 (N-14) IMMEDIATELY AFTER BLASTING

FIG. 10-5: VIEW OF SAME BLAST (N-14) APPROXIMATELY SEVEN MONTHS LATER SHOWING DEGREE OF REVEGETATION

of this picture, has been totally revegetated except for the cracks.

It was evident by the spring of 1987 that, in many of the blast-created depressions, areas of microclimatic conditions had been created. In these areas, the soil, water and temperature regimes had been subtly altered to suit different forms of vegetation. A number of different plant species, in addition to the brome grass which had formerly covered the area, were noted. Natural vegetation of the site will take several years. It is expected that some woody or herbeous vegetation types will establish themselves. In areas where moist soil conditions prevail wet meadow or wetland vegetation should establish.

It is felt that the wildlife habitat has already been enhanced by the work, and that this can be achieved for other such sites without the use of equipment for recontouring and revegetation work. In addition the possibility exists to create ponds or wetlands, and to introduce plant types which the wildlife manager deems best suited to use by different types of wildlife. Such work could then be used to create more beneficial land use.

If commercial application is to be made to land reclaimed by AML blasting, it would probably be necessary to remove topsoil prior to blasting. This would only have to be carried out from above the rooms and entryways blasted. Dozed material could be banked up between rooms in pillar areas, if this can be practically achieved without creating serious access problems to blasting operations. Where rooms are located close together, or pillar failure is suspected, this might not be the appropriate course of action.

Some additional remote backfill may be required, to compensate for the depression depth. Guidelines for estimating approximate volumes were given in Chapter 8. In other cases material swell may be sufficient to fill the depression. In either case the topsoil could then be dozed back onto the final land surface.

11. CONCLUSIONS AND RECOMMENDATIONS

11.1. CONCLUSIONS

1. This research project has shown that the use of crater blast design techniques to cave-in underground mine workings is technically feasible.

2. Blasting is an AML reclamation method which minimizes the chance of additional subsidence after reclamation, and so enhances the long-term safety and utility of the site.

3. Crater blast design utilizes spherical explosive charges (charges in which the length does not exceed 8 times the charge diameter), with distances between charge centers and available free faces being scaled by the cube root of the charge weight.

4. The test site selected proved well suited for the research. It included variable cover, varied tunnel dimensions and it was well documented. It was Game and Fish Department land and changes to the topography, as this affected wildlife habitat, could be examined.

5. The performance of adequate preliminary work including site reconnaissance, data gathering and exploratory drilling will have a significant impact on blasting success.

6. Overall project cost will be minimized when detailed topographic and mine maps are available. Decreasing levels of information will increase preliminary exploration drilling cost and initial engineering design time and cost.

7. The preparation of composite maps incorporating both the topography and the mine workings is very helpful. This is especially true if the topographic maps result from recent aerial survey and include existing sinkholes.

8. Data from the exploratory drilling should be gathered carefully. Exploration will be the largest percentage of the total site selection, evaluation and exploration cost. The total cost of exploration will

be quite sensitive to drill cost per foot. It will be essential to budget adequate monies for exploration and this will be a function of site uncertainty.

9. ANFO should be used, whenever possible, as the explosive. It is inexpensive, easily loaded and reliable. Unless very hard strata are encountered over the workings it has adequate energy output (870 cal/gm) to fragment the material.

10. If water is a problem one should attempt the use of plastic borehole liners, which normally work well for hole diameters of 6-inches and above. The ANFO is loaded inside the liner which is sealed at one end. For cases where extremes of water are present or blast hole liners don't work then an emulsion, waterproof heavy ANFO or slurry ought to be considered. These products may also be useful if the strata are hard and maximizing the weight of explosives in the cratering deck is desired. This is because they have considerably higher densities than ANFO. Care should be taken to find a product which will be easily loaded into smaller diameter holes. This was a problem in the test program.

11. Initial design for the deck charges was based on cube root scaling with $d/W^{1/3}$ = 2.5 $Ft/Lb^{1/3}$. In general scaled depths of burial in the range of 2.0-2.3 $ft/(lb)^{1/3}$ were employed successfully. If a higher density slurry explosive was used in the bottom deck the associated SDOB would be in the order of 1.9-2.2 $ft/(lb)^{1/3}$. However, actual design had depths of burial that varied due to physical dimensions such as hole depth and to factors such as occurrence of rock layers and thick roof coal. Variations were not generally drastic and did not appear to affect results significantly.

12. The collar (explosive/stemming interface) was designed for $d/W^{1/3}$ of 3.1 $ft/lb^{1/3}$. This was intended to heave the surface but not create flyrock. Subsequently, to further protect the topsoil from disruption a scaled depth of burial of 3.25 $ft/lb^{1/3}$ was chosen. For 8-inch diameter holes $d/W^{1/3}$ = 3.5$ft/lb^{1/3}$ was used. It is concluded that in the geology experienced at the site a $d/W^{1/3}$ of 3.25 $ft/lb^{1/3}$ is quite adequate to provide heave and to avoid flyrock or surface bursting.

13. Initial spacing of the holes along a line was 2.0 times the depth of burial. This appeared too great.

Spacings were reduced to 1.5 times the depth of burial along the row, which worked well. It was subsequently found that, on the wide rooms, spacings of 1.85 times the depth of burial were adequate. Spacings between rows had to account for the dimensions of the openings. In 22 foot wide development the rows were 8 feet apart with 7 feet from each row to the pillar. For narrow entries spacings between holes of 1.5 times burial depth were initially tried. It was subsequently found that optimum results were achieved if this was reduced to around 1.1 times depth of burial. For narrow entries the blast pattern consisted of one row centered on the axis.

14. In general if wall-to-wall distances in the development to be blasted exceed 1.75 times the depth of burial, two rows of blastholes, on which hole locations should be staggered, are recommended. It was found, however, that a single row could in some cases be used for wider rooms, and a hole spacing of 1.4 times the depth of burial was found to be acceptable. This works well when there is adequate void beneath, and the large roof span results in an overall weaker structure.

15. Standard loading scales were established for six inch blastholes for the range of overburden depth (35-65 feet) experienced at the test site. This is a recommended practice to improve the speed and efficiency of loading operations.

16. An integral part of the application of crater theory to AML blasting is the use of decked spherical charges which detonate in sequence from the bottom up. The lowermost deck blasts into the underground void. Each charge blasts material into the void created by the previous charge in the detonation sequence. This can be achieved by use of down-the-hole millisecond delays connected to a detonating cord downline.

17. Millisecond delay times should be adequate to allow each cratering deck to relieve in advance of the next. In this study 25ms between decks in a blast hole was found to provide marginal performance. The use of 50ms delays, which gave a more systematic diagonal detonation sequence, provided greater relief and enhanced results.

18. Millisecond delays are also essential to the control of vibration. For this reason use of surface delays between blastholes is also recommended. In order to

reduce airblast effects, a reduced noise system such as Nonel is recommended.

19. In the test blast program 21 blasts were taken, which involved part of a 10 acre site area. In general six inch blastholes were used to collapse 22 foot wide rooms in a regular room and pillar operation. In two cases 8 inch diameter blastholes were employed. Two blasts were centered exclusively on 12 foot wide access development. Two blasts were employed partially or totally with the objective of filling in pre-existing sinkholes.

20. Line-up drilling along the room or entry is important to proper blasthole placement. Blasts should be staked out by a responsible person and the holes should be drilled on the stakes. Failure to do so will reduce the effectiveness of the blasting with possible bridging resulting. During blasthole drilling the drillers should record any variations in geology from the norm. Presence of water should be noted as well. Blastholes should be carefully measured before loading.

21. Seismic cones make good hole bottom plugs. They are placed fairly easily and are readily available. It is important to secure them using light twine so the load is well supported. Placement must be correct and this means, again, careful taping of the hole.

22. Deck loading requires careful operation with constant taping of the holes for correct explosive and stemming deck heights. Sloppy procedures here will lead to poor results.

23. Nonel surface tie-ins using 42ms noiseless trunkline delays were employed. These have the advantage of being of low noise, safe and easy to connect. The burial of delay elements, blasting cap and primacord pigtails is highly recommended to minimize airblast and noise.

24. The high-speed camera was an effective tool for analyzing blast effectiveness. Surface heave could be determined qualitatively and quantitatively. Millisecond delay accuracy was determined and the onset of caving could often be observed. Camera studies showed that the blasts consistently detonated in sequence. The 42ms surface delays had a mean delay time of 37.3ms.

25. The down-the-hole delays displayed more variation. This has been a common observation in the mining industry. The number 6 and 7 delays seemed to detonate at about the same time. However, the sample size was quite limited and general conclusions should not be drawn. The change from 25ms to 50ms delay between decks acted to eliminate problems resulting from the delay firing time variations.

26. The blasts for which surface movement was studied had total vertical displacements of 8 to 10 feet. Velocities were in the 10 to 20 feet per minute range. This would be typical. Movement of the top without flyrock was possible. Disruption of soils, although significant, was minimized. Such suitable plant growth material was kept near the top of the blasted material and some plant growth was noted the next spring.

27. The method of blasting studied leads to low charge weights per delay period and this means low levels of vibration. Square root scaling ($D/W^{1/2}$) was found to better represent the vibration data than cube root scaling. It is concluded that the detonation of multiple deck delayed cratering charges generates a ground disturbance similar to that of a linear column charge. For a scaled distance of 42 ft/(lb per delay period)$^{1/2}$ a vibration level less than 0.5 ins/sec can be maintained. This should eliminate most problems. Charge weights that can be detonated at this scaled distance are reasonable.

28. Airblast was low, being at least an order of magnitude less than that representing the onset of damage. Care taken in loading and connecting the blast was important in this regard.

29. Larger diameter holes will mean greater weights per delay period and therefore this may be a restriction on hole size when close to houses and other buildings.

30. Blasting near to structures is possible. This study shows blasting as close as 1,000 feet from buildings represents little problem. Blasting to within 400 feet of structures will be acceptable if care is taken. This should also apply to water wells and pipelines. It is assumed that the structure or facility is not undermined.

31. The slurry explosives exhibited loading densities much lower than expected. This was a field observation, later confirmed by detailed technical analysis of actual blast data. It appeared to be due to releasing the product from the bag and dropping it down the hole. For some reason these products did not compress or couple as well as expected. If a slurry is to be used care must be taken in selection. It may be that an emulsion or heavy ANFO will be more suitable. An important consequence of the above is that blasts should be designed based on the expected loading density of slurry type products, and not the bagged density as claimed by a manufacturer.

32. In the majority of cases the post-blast profile consisted of a V-shaped depression running along the center of the blasted room. In some cases there was surface heave over the blast area. When a depression is created it is possible to state definitely that a blast was successful in collapsing the underground structure. It is very important to realize, however, that surface heave will result from a successful blast if there was insufficient room in the underground volume to accommodate blast-induced material swell. Therefore, surface heave is not a definitive indication of an unsuccessful blast.

33. Measurement of the subsidence or heave produced by blasting was carried out for some blasts. A model was developed that allowed an estimate of the typical material swell encountered during testwork to be calculated. This may also be used to predict post -blast profiles for other work of this type. The results show when to expect depressions or heave based on variations in critical factors such as overburden depth, room height and swell factor.

34. Post-blast drilling indicated that in most cases where surface heave was experienced the voids had been filled. This exploration was completed where possible but did not occur for all heaved blasts. Visual and drilling evidence showed that two blasts were known not to be entirely successful. These showed bridging of the material at the position of the two upper decks. These were blasts #4 (N-1,N-2) and #9 (NC-7).

35. The test program was largely successful technically. A few problems were encountered as is expected in a research project. Those problems were mostly worked out by the end of the program through changes in

millisecond delay times, hole spacings, number of rows, explosives placement and so forth. It was found that the blast results correlated well with overburden depth, and the void depth available in the rooms. Closure of an individual sinkhole proved successful also. Great potential exists for the use of blasting in this area, especially in steep-sided sinkholes.

36. The exclusive use of ANFO in a blast provided equally good results as the use of slurry in the bottom deck. This can be done unless there is water in the bottom of the hole.

37. In rooms 22 feet wide it appears that one row of holes yields equally good results as two rows. The spacings between holes have to be reduced to 1.4 times the depth of burial. The resulting surface depressions will be deeper and not as wide.

38. Hole diameter should be governed by hole depth primarily. The goal should be to limit the number of decks to 4. This is less complicated to work with, the blast is less likely to choke and millisecond delay time variations (DTH) are less likely to be a problem. Where overburden cover exceeds 60 feet it may therefore be preferable to use larger diameter blast-holes than six inch. However, blast vibration may also play a role in this. It should also be noted that five decks were used successfully in a few cases.

39. The costs associated with AML blasting were analyzed in detail, and a model was developed for prediction of the cost per acre associated with work of this kind for different site and operational parameters. Analysis of actual blast cost data from the testwork program indicated that the cost per acre for blasting was variable. It was concluded that the parameters to which blasting cost was most sensitive included overburden depth, drilling and explosives cost, and whether one or two blasthole rows could be employed for a given development size.

40. Since reclamation costs are considered on a per acre basis, nearly all methods are very dependent on the amount of mining in an acre of ground. In a typical room and pillar operation this correlates directly to the ratio of room to pillar widths, and can be considered in two dimensions as a percent extraction by area. This is by far the most important factor in

determining the cost of AML blasting on a per acre basis.

41. In one scenario studied using the AML blast Cost Model a single 3 or 4-man blasting crew was considered for a 10 acre reclamation site size. It was determined that a typical blast that could be taken in a single day would be 220 feet long, if two rows of blastholes were required, or about 350 feet long in the single row case. Where 6 inch blastholes were used in 50 feet of overburden cover, a typical single or double row blast would cost in the order of $4500-$5000.

42. At a site where the rooms are 20 feet wide, and the room and pillar configuration was such that a 40% (by area) mining extraction exists, a typical double row blasting operation using the data described above would cost in the order of $20,000 per acre. If this same configuration could be blasted using a single row of blastholes, this cost reduces significantly, to around $12,500 per acre. However, if the rooms were narrower, at 12 feet, then the cost using a single row of blastholes is again around $20,000 per acre at the same 40% extraction. If the percent extraction is higher, at 60%, then for the wider rooms the per acre costs are $30,000 and $19,000 for double and single row applications respectively.

43. The above example illustrates that blasting costs can vary widely. The number and size of the underground openings present in a given acre of land are by far the most critical factors affecting the AML blasting cost of that site.

44. Blasting is an AML reclamation method that is cost competitive with other methods of area reclamation. It may be less competitive with individual feature reclamation. The depth to the works affects costs especially if drilling cost is high. It would also affect daylighting costs but would not affect remote backfill to the same extent. However, for the ranges of depths tested and drilling cost of $1.00/ft or less the method is comparable. With ANFO as the explosive and one row blasting on wide rooms, the method is quite cost attractive.

45. In most cases the use of blasting will require topsoil removal, regrading (possibly including fill), topsoil replacement and seeding. In some cases, however, such

as wild life habitat production leaving the area as blasted may well lead to micro-climatic and vegetation systems that enhance the habitat.

11.2 RECOMMENDATIONS FOR FURTHER WORK

1. Further experimentation is recommended using 8 inch diameter blastholes in overburden cover depths in excess of 60 feet. The technical feasibility of this should be investigated together with the increased levels of blast vibration that may result.

2. Further modifications to the upper explosive deck should be attempted, to avoid the possibility of bridging of the blast at this level. We have concluded that to effectively crater to the surface and also downward a double length deck should be attempted. A charge length of 12 times the diameter is proposed.

3. Great potential exists for the use of blasting to fill in individual sinkholes. It is suggested that this possibility should be studied in more detail.

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