

**PERFORMANCE MONITORING OF
ELECTRIC MINING SHOVELS**

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ABSTRACT

A P&H 2800XP electric mining shovel working in a Western Canadian surface coal mine was instrumented with microprocessor-based monitoring equipment, and its performance monitored as it excavated a series of test blasts. Hoist and crowd motor armature voltages and currents, dipper trajectories, cycle times and load weights were recorded. A diggability index has been established based on the responses of the hoist motor. Values of the diggability index correlated well with digging conditions as observed during monitoring and with muckpile fragmentation size distributions as determined by a photographic survey. This correlation establishes the ability of an instrumented shovel to diagnose the efficiency of ground preparation practices (blasting) by identifying variations in muckpile diggability. The data on dipper trajectory has demonstrated that variations in digging practices do exist amongst an experienced group of shovel operators, and that variation in trajectory significantly influences values of the recorded motor performance parameters. An approach is described to account for variations in digging practice on assessments of diggability. An allied investigation into the ability of time studies to define diggability, revealed dig cycle times to be operator dependant, and unrelated to levels of digging effort.

RESUME

Les performances d'une pelle excavatrice électrique de mine de type P&H 2800XP instrumentée à l'aide de transducteurs reliés à un microprocesseur ont été étudiées lors de sautages d'essai.

Le voltages et les ampérages des moteurs de levage et d'avance-retrait ainsi que les trajectoires des godets, la durée des cycles et les charges du godet furent mesurés et enregistrés.

Les réactions enregistrées à partir du moteur du treuil ont ainsi permis d'établir un indice d'excavabilité. Les valeurs de l'indice d'excavabilité obtenues démontrent une bonne corrélation avec les conditions et les difficultés d'excavation réellement rencontrées ainsi qu'avec la distribution granulométrique des fragments de roche telle qu'établie par relevé photographique.

L'emploi d'équipements instrumentés et des formules de corrélation suivant la variation d'excavabilité permettent d'établir un diagnostic sur l'efficacité des méthodes et des résultats des travaux de fragmentation (sautage).

L'étude des trajectoires du godet a aussi permis l'observation de nombreuses variations dans les pratiques et les méthodes d'excavation et ce, même parmi un groupe d'opérateurs de pelles mécaniques chevronnés. Ces variations influencent de façon significative les paramètres de performance des moteurs d'entraînement du godet.

L'auteur décrit une approche d'évaluation de l'excavabilité d'un terrain qui tiendrait compte des variations des différentes méthodes d'excavation. Une étude parallèle des temps et des durées des cycles d'excavation en fonction de l'excavabilité a révélé que les durées des cycles d'excavation dépendent la méthode utilisée par l'opérateur et qu'il n'y a pas de relations avec les niveaux d'effort d'excavation.

LIST OF SYMBOLS AND ABBREVIATIONS.

$|x|$ - absolute value of x

A - amps

B - blasthole burden

β - angle of crowd arm with horizontal - degrees

B_e - effective burden

BCM - bank cubic metre

CAE - crowd arm extension

cc - cubic centimetre

CCDI - crowd current diggability index

CDI - crowd diggability index

cm - centimetre

CR - cut ratio

CRMS - load cycle root mean square current

CS - coefficient of uniformity

CU - coefficient of sorting

cu.m - cubic metres

CVDI - crowd voltage diggability index

D_n - diameter at which $n\%$ of sample are smaller

DI - hoist motor based diggability index; increasing values of all diggability indices relate to increasing digging difficulty.

e - capacity of dipper (Pandey, 1974)

E - Youngs modulus

E.U. - energy units (of explosives)

FRMS - fill cycle root mean square current

ft - feet
gm - gram
H - hour
HCDI - hoist current diggability index
h.p. - horse power
HRP - hoist rope position
HVDI - hoist voltage diggability index
I - current
kg - kilogram
l - litre
m - metre
min - minute
MPa - mega pascal
ms - milli second
mV - milli Volt.
n - number of samples
RBS - relative bulk strength
RMS - root mean square current
rpm - revolutions per minute
S - blasthole spacing
SDS - shift diggability summary
 S_e - effective spacing
sec - seconds
STD - standard deviation
 Σ - summation
T - torque
t - short tons
TRK - truck

UCS - unconfined compressive strength

V - volts

VOD - velocity of explosive detonation

X - fragmentation size

$\bar{X}$ - mean fragmentation size

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1.0 INTRODUCTION.

1.1 OVERVIEW.

This thesis relates to research on the performance monitoring of electric mining shovels, at the Fording River Mine of Fording Coal Ltd., Elkford, British Columbia. Records of shovel performance, derived through microprocessor based instrumentation, have been analyzed and related to details of digging trajectory, bench geology, variations in blasting design parameters and the fragmentation characteristics of the post blast environment. Research has aimed at advancing a method for quantitative assessment of the blast results, which are viewed as a composite of not only the size distribution of the fragmented mass but also the inherent ease with which the mass is excavated i.e. its diggability.

Shovel performance monitoring field studies were conducted over a period from August 1987 to November 1988, during which time, two separate phases of research were completed. In the first phase - summer 1987, field studies involving the manual recording of shovel cycle times were conducted in a preliminary investigation designed to examine shovel performance in a range of mine environments. Details on shovel operation, mining and blasting practice, and bench geology were also studied. The results of the preliminary research defined the need for and requirements of, a more focused, concentrated investigation.

The second, final phase of the field research (June to November 1988) focused on a more comprehensive "integrated bench study", in which microprocessor based performance monitored data from both an instrumented electric mining shovel and blasthole drill, working the same mine bench, were combined. This thesis concerns itself primarily with an analysis of the shovel monitored data collected during the integrated bench study. Records of monitored shovel performance parameters have been

interpreted along with data on fragmentation size distributions, shovel production statistics, mine bench geology and blast designs. This has permitted establishing direct measures of the ease with which a fragmented rockmass has been excavated i.e. muckpile diggability.

Integrated Bench Study:

A P&H 2800XP electric mining shovel, equipped with a 30 cubic yard dipper, was monitored as it excavated three specifically designed blast - comprising in excess of 500 000 bank cubic yards of waste, in the bench selected for the integrated study. Monitoring established that the response of key shovel performance parameters was related to variations in muckpile diggability.

Drill monitored data from the test bench was used in conjunction with geophysical logging and core samples, to precisely characterize the stratigraphy of the mine bench. Microprocessor based shovel instrumentation has enabled the recording of the performance of the hoist and crowd motors of the shovel, the path or trajectory of the dipper through the muckpile for each load handled by the shovel, the weight of material moved per load and the various cycle times for each load. The performance of a single shovel was monitored as it excavated a series of blast zones within the test bench, each characterized by a unique set of blast design criteria (i.e. drill pattern, charge concentration). In this manner, a series of post-blast environments, each of which through their varied physical nature, offered unique levels of digging effort, were created from known geologic and blast design elements. A comprehensive photographic survey of the fragmentation size distribution handled by the shovel during each day of the integrated bench study, has provided a quantified aspect of muckpile conditions against which to compare the records of shovel monitored data.

This current research is unique in that a comprehensive shovel monitoring investigation has not been undertaken to such detail and in as controlled an experimental design. There are only two prior research efforts in the area of relating shovel monitored data to muckpile diggability, both in Australia (Williamson et al., 1983 and Mol et al., 1987). It is felt that the experimental design of prior research made it impossible to attribute variation in shovel performance solely to variations in muckpile diggability. In both cases the performance of more than one shovel model at a time, each working in separate uncontrolled geologic and blast environments was investigated. This compromised the ability to account for several uncontrollable variables. Additionally, in both studies only subjective assessments of muckpile fragmentation were made by operating personnel and no data on dipper trajectory was recorded.

The research underlying this thesis is the first to undertake monitoring of dipper trajectories. The records of dipper trajectory for each monitored shovel dig cycle have proven crucial in several regards. Prior research has assumed that variation in the digging practices of shovel operators is insignificant. It is suggested that this conclusion can only be substantiated by examining the dipper trajectories; this is the only record that will document the actual digging practice or tactics adopted by each operator. It has been demonstrated through the current research that variations in characteristic trajectories do exist amongst operators, and that the trajectory negotiated through a fragmented rockmass will impart a much greater degree of influence over monitored shovel motor responses, than have variations in the actual characteristics of the muckpile.

1.2 PROPOSED HYPOTHESIS.

The electric mining shovel is a machine designed to load material which has been prepared for loading through the blasting of an intact rockmass. The hypothesis of this thesis is that its performance can be related to the post-blast conditions, such that a relation between the ease with which the fragmented rockmass is excavated and the relative success of the ground preparation process can be established.

If the relationship between the shovel's performance and the physical characteristics of the muckpile can be established, then the shovel becomes a diagnostic tool for use in optimizing blast design. The hypothesis has been tested by accounting for variations in operating practice (dipper trajectory) and by conducting the investigation in a controlled geologic environment in which the resultant muckpiles are, as much as possible, a consequence of intentionally varied blast design elements. Variations in the monitored shovel performance can therefore only be attributed to changes in the ease with which the muckpile is excavated, i.e. diggability.

Responses of the shovel hoist motor have been used to derive quantified ratings of diggability. These have demonstrated good correlation with digging conditions as observed in the field during monitoring. An approach has been adopted to account for variations in dipper trajectory that were seen to exist as a consequence of variations in operating practices, and to have been motivated by changes in digging conditions i.e. degrees of fragmentation and looseness.

The significance of developing and validating this hypothesis is that at present there exist no objective methods of post-blast evaluation capable of providing continuous, quantified feedback on the success of blast designs. Many mining operations are reluctant to undertake blast optimization programs because

current methods of post-blast evaluation are either a) too costly, time consuming and complex or b) simplistic and subjective to the extent that they are of little value. Adapting the shovel as a monitor of blast conditions satisfies the requirements of what could be considered an ideal post-blast evaluation technique. It is comprehensive, inexpensive, objective and consumes no time. The shovel must interact with every bank cubic yard of a blast whether or not its performance is monitored.

Furthermore, it is logical to rate blasting efficiency on the basis of the ease with which muckpiles are handled by the shovel since the end ambition of the drilling and blasting process is to generate muckpiles readily managed by the loading equipment. In this sense shovel monitoring is seen as a tool particularly suited to the needs of the surface mining industry. It is also viewed as significant to the explosives industry, in advancing and validating more traditional techniques of blast design, optimization and post-blast evaluation i.e. high speed photography and computer modelling. In these areas, there exists a need for quantified measures of what has been termed "muckpile looseness" by the industry.

The deployment of the current shovel monitoring technology and interpretive procedures are seen as significant to mining operations such as the Fording River Mine which spends annually between \$8 000 000 to \$12 000 000 in explosives and accessories. This large surface coal mine employs both dragline and truck-shovel operations to remove 45-50 million tons of waste per year to produce annually 5 million tons of clean coal. The truck-shovel operations are responsible for over 70% of mine waste production. The ability to improve the use of explosive energy through optimizing blasting practices, will help to ensure the cost competitiveness of Canadian surface mining operations in world markets.

1.3 STRUCTURE OF THE THESIS.

The shovel instrumentation, monitoring and interpretive procedures adopted for this thesis are, to a degree, specific to the design of the investigated shovel; a P&H 2800XP Electrotorque electric mining shovel. An introductory component of chapter 2 will establish the importance, significance and likely longevity of the electric mining shovel in open pit operations. Seventy percent of electric mining shovels in use worldwide are P&H Electrotorque model shovels (Runno, 1989 pers. comm.). The technology developed through the current research is therefore well suited to widespread utilization. The remainder of the chapter will describe the design and operation of the P&H 2800XP electric mining shovel.

Key to the success of the current research has been the design of an appropriate shovel monitoring strategy. Chapter 3 investigates the nature of shovel monitoring instrumentation employed to collect data on specific shovel performance parameters. This has included the adaptation of existing micro-processor based technology in the form both of a commercial shovel production monitor manufactured by General Electric and a separate data-logger.

The nature of the environment in which the research has been conducted is addressed in chapter 4, which describes the geology, mining methods and blasting practices at the Fording River Mine.

Initial phase research involved a series of manual time studies conducted in a range of mine environments. Chapter 5 opens with a brief discussion of the key observations drawn from this phase of the research. A complete explanation of the methodology and conclusions drawn from the exercise are presented in Appendix B, which documents the nature and extent of the investigation. This phase of the research was regarded as preliminary, and served to clarify the appropriate design

criteria for the subsequent comprehensive field study, in which issues of the actual machine response to documented ground conditions would be addressed. The concept of the integrated bench study was therefore evolved, and work began on the design of a series of test blasts in a controlled geologic environment. The remainder of chapter 5 describes the extent of the study in terms of the definition of bench geology and design aspects of the test blasts investigated therein.

The intent of engineering separate blast zones, within an environment of known and relatively continuous geology, was to generate a series of muckpiles of unique characteristics, within which to compare shovel performance. In this sense, it has been the primary intent of the research to relate shovel performance to muck characteristics, not to relate the ensuing muck characteristics to aspects of blast design and the properties of the intact rockmass. This aspect is still of interest and is addressed in Appendix D, which discusses the fragmentation of rock by blasting. Controllable blast design variables such as drilling pattern, explosive strength etc., are briefly reviewed in order to attempt to relate how variation of these design criteria would theoretically influence muckpile conditions in each blast zone of the integrated bench study. The influence of uncontrollable blast variables, i.e. structural geology and physical properties of the rockmass, are also examined.

It was required that from each blast zone, some quantified measure of muckpile characteristics be available against which to compare shovel performance. Chapter 6 begins with a brief review of the post-blast evaluation techniques considered for this purpose. The method selected was that of a photographic survey of muckpile fragmentation. Details on the method's application, and how it has been adapted to suit the needs of the current research are described. Results are presented, based on image analysis of over 60 photographs of muckpile fragmentation, accounting for in excess of 60 000 fragments.

The integrated bench study is viewed as a significant advance over the only two prior investigations involving shovel performance monitoring. These prior works by Williamson et al., 1983 and Mol et al., 1987, are described in detail in chapter 7. Several deficiencies associated with the prior investigations are addressed with a view to detailing how the current research has approached alleviating these weaknesses, within the confines of a more controlled experiment - the integrated bench study.

The data collected from the integrated bench study is of unique scope. The manner in which it has been processed, interpreted and ultimately analyzed toward establishing the relationships between monitored shovel motor responses and diggability, is addressed in sections 8.1 to 8.8. The remainder of chapter 8 is dedicated to analyzing the significant influence of dipper trajectory, as it pertains to measures of diggability and assessing variations in operating practice. Attention is then turned to describing a method of incorporating the influence of varied dipper trajectories in ratings of diggability, i.e. correcting for variations in digging practices.

The thesis concludes with a chapter relating how the shovel monitoring technology developed through the current research, could be employed within a comprehensive, iterative blast optimization program. In chapter 9 the philosophy surrounding the integrated nature of the research is brought into perspective by illustrating how performance monitored data collected from both shovels and drills during normal operation, is incorporated within mine planning systems. Future work is planned therefore in developing a shovel monitoring system and software to allow continual records of diggability to be generated and incorporated within mine planning systems, in conjunction with records of explosives loading based on drill monitored data.

Recommended extensions of the research relate to greater involvement with explosives technology, especially in the areas of advancing the accuracy of computer blast modelling. Additional research is also recommended in order to assess the applicability of the current shovel monitoring technology towards operator training, equipment maintenance and automation.

1.4 NOTE CONCERNING UNITS OF MEASURE.

Metric units of measurement are used throughout the thesis except, for reasons of simplicity and clarity, in Chapter 8 "INTERPRETATION and ANALYSIS OF SHOVEL PERFORMANCE PARAMETERS". This is due to the following reasons: Shovel geometries and production capacities have typically been quoted by manufactures in imperial units (feet and short tons). The shovel instrumentation employed for the research was manufactured in the United States (by General Electric), and hence, reported all data in imperial units. Production statistics, as expressed by weight at the mine, are given in short tons (truck weigh scales also report load weights in tons). Production volumes are, however, reported in metric units - bank cubic metres.

Additionally, blast patterns at the mine are specified in metres, but blasthole loading instructions are, by convention, issued in feet.

2.0 ELECTRIC MINING SHOVELS.

The role of the excavator in open pit mining operations is typically for the handling (loading, casting etc.) of either waste or ore, which may or may not have been prepared for the specific handling process ie. - blasted or ripped material. In most instances primary loading equipment is selected in advance, in accordance with the nature of the material to be handled, and also considering the extent to which the material is prepared in order to make it suitably managed by the selected excavating equipment.

Several open pit operations operate with a range of excavating and loading equipment which may normally include one or more of the following - draglines, bucket wheel excavators, front end loaders and stripping shovels.

The mining shovel remains the primary excavation tool in most operations in which hard rock, requiring prior ground preparation such as ripping or blasting, is to be mined along benches not generally exceeding 15 m (50 feet) in height. These shovels can deliver the brute strength necessary to deal with the difficult digging conditions often imposed by this type of material - large, frequently interlocked fragments of higher than average strength waste rock units.

There are at present two types of mining shovels in broad use throughout the world - the electric mining shovel and the hydraulic shovel. Brief descriptions are offered for both to establish their relative merits. Since the early seventies several advances have been realised in the development and application of diesel-hydraulic technology to large mining shovels. There are those (Reid, 1988) who believe these new generation excavators will one day displace the electric mining shovel as the mining industries primary excavator-loader in

surface operations. Since the current research has been based entirely on data acquired through the instrumentation and monitoring of a modern electric mining (cable) shovel, it is important to establish the short and long term view of the relevance of the machine to surface mining.

2.1 HYDRAULIC SHOVELS.

Through Figure 2.1.1, it is clear that the fundamental design of an hydraulic shovel combines the principal benefits of both an electric mining shovel and a front end loader, to produce a very versatile machine, well adapted to selective mining methods. The design of the excavator permits a "wrist action" of the dipper allowing for efficient separation of waste from ore, thus reducing subsequent costs of dilution in processing the end product (Singhal, 1986). When first introduced to the industry units were generally equipped with buckets of up to 20 cubic meters only, the newer models are now available with buckets to a maximum capacity of 34 cubic meters.

2.2 ELECTRIC MINING SHOVELS.

While hydraulic shovels offer particular advantages in maneuverability (i.e. digging tactics), they are not as well suited to lengthy excavations (decades) in hard digging environments, as are the traditional electric mining shovels.

In most open pit operations electric shovels are the principal excavators. They can provide high availability while working 24 hours a day, often yielding 20 or more years of service under harsh operating conditions before being retired.

Key to the acceptance of these excavators in most operations is their ability to handle the most difficult digging conditions. Here, the electric mining shovel excels where other excavators

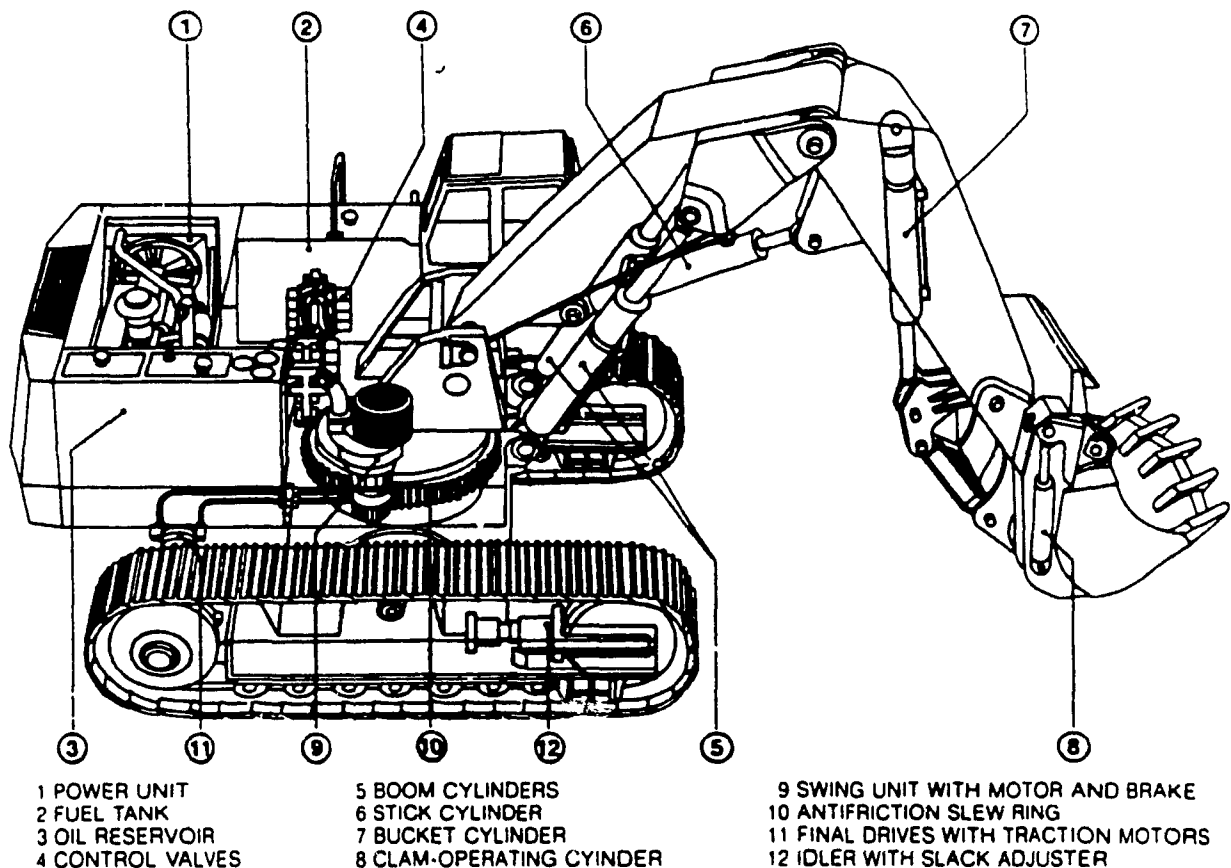


Figure 2.1.1 - Configuration of a hydraulic shovel (Source; Demag Corp.)

fail; the handling of large, poorly fragmented interlocking rocks. They are also suited to medium and easy digging conditions, and when employed in these situations they are often refitted with dippers having a larger than rated capacity for the

equipment (Cherones, 72). The harder the digging, typically the coarser the fragmentation of the material being excavated, the increased stresses imposed on the dipper by this material, results in a need to employ smaller more rigidly engineered dippers in these environments. Conversely, in environments where fragmentation is improved larger dippers may be employed on a machine rated for hard digging conditions, resulting in increased productivity.

The conditions described above are typical of most open pit or surface mining operations in which the material handled requires some form of previous ground preparation ie. blasting. This draws into focus the natural combination of electric mining shovels with surface operations in which ground is typically prepared for excavation through the blasting process.

One of the more significant advantages electric mining shovels offer over hydraulic shovels is their overall dependability. The operational availability of principal loading equipment is of primary concern to a mine. The electric drive mechanisms of electric mining shovels, coupled with simple design, have proven more reliable than the diesel drives of most hydraulic shovels, resulting in average availabilities in hard rock mining of 90-95% (Bulin, 1973). The complexities of hydraulic excavators make them less reliable instruments, especially so in hard mining situations. Scheduled maintenances on hydraulic equipment is more frequent than on their electric counterparts, furthermore, since these pieces of equipment are much smaller than electric mining shovels access to faulty components is more difficult, resulting in longer down time for repairs or maintenance. Because of the size of electric mining shovels their internal and external components (motors, drives, gearing etc.) need not be crowded, this makes for easy access to those elements requiring attention, which in turn has a direct bearing on how long a piece of equipment will be "down" for preventive maintenance or repair.

2.3 ELECTRIC MINING SHOVEL DESIGN.

In the following sections the design, mechanical and electrical and proper operation of a P&H 2800XP (as monitored during the research for this thesis) will be addressed.

2.3.1 The P&H 2800XP - General Characteristics:

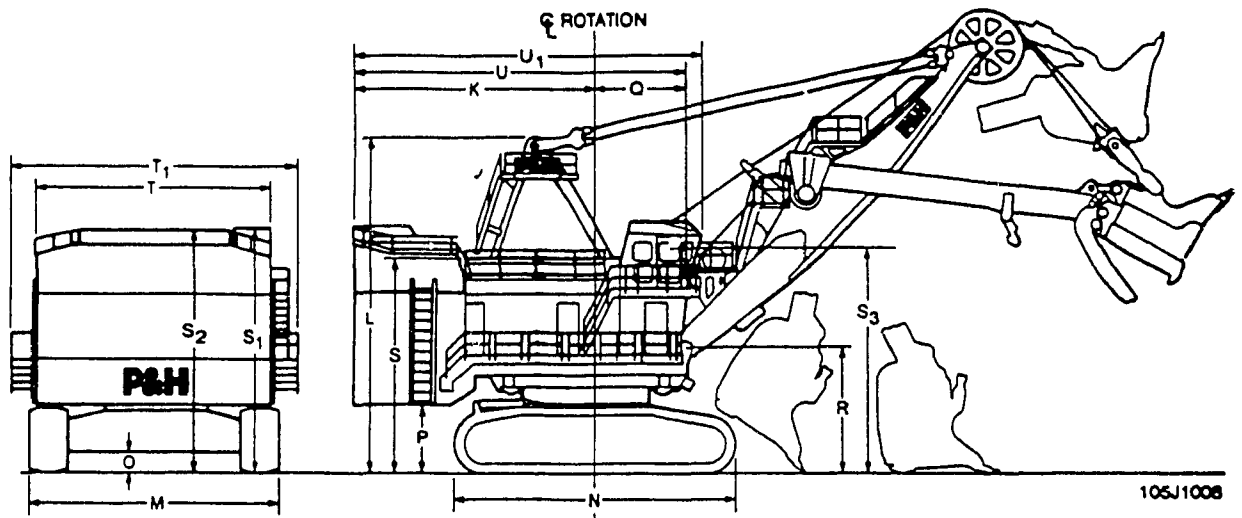
The P&H 2800XP is manufactured by Harnischfeger of Milwaukee, Wisconsin, U.S.A.. It is the second largest electric mining shovel manufactured by the company, the P&H 5700 is the largest in the world, with a dipper capacity of up to 75 cubic yards (57.3 cu. m). The 2800XP is available with or without dual propel motors, the unit investigated in this thesis was equipped with dual motors. The design and operating specifications for the 2800XP are given in Figures 2.3.1 and 2.3.2 respectively. The particular model investigated during the research for this thesis was equipped with a 30 cubic yard (23 cu. m) dipper.

2.3.2 P&H 2800XP Electrical Systems:

A comprehensive analysis of the shovels electrical systems is well beyond the scope of this thesis. However, much of the data accrued during the research for this thesis, pertains to the instrumentation and monitoring of the shovel's electric motors. It is intended therefore, to provide a fundamental overview of the shovel's main electric systems and their design.

The P&H 2800XP is powered by P&H's "Electrotorque Control"¹ system for electric mining shovels. Electrotorque control is a patented system which employs solid state electronics to produce controlled delivery of adjustable voltage DC main operating power, from the pit supplied AC power to the shovel. The main operating DC power is then supplied directly to extra heavy duty,

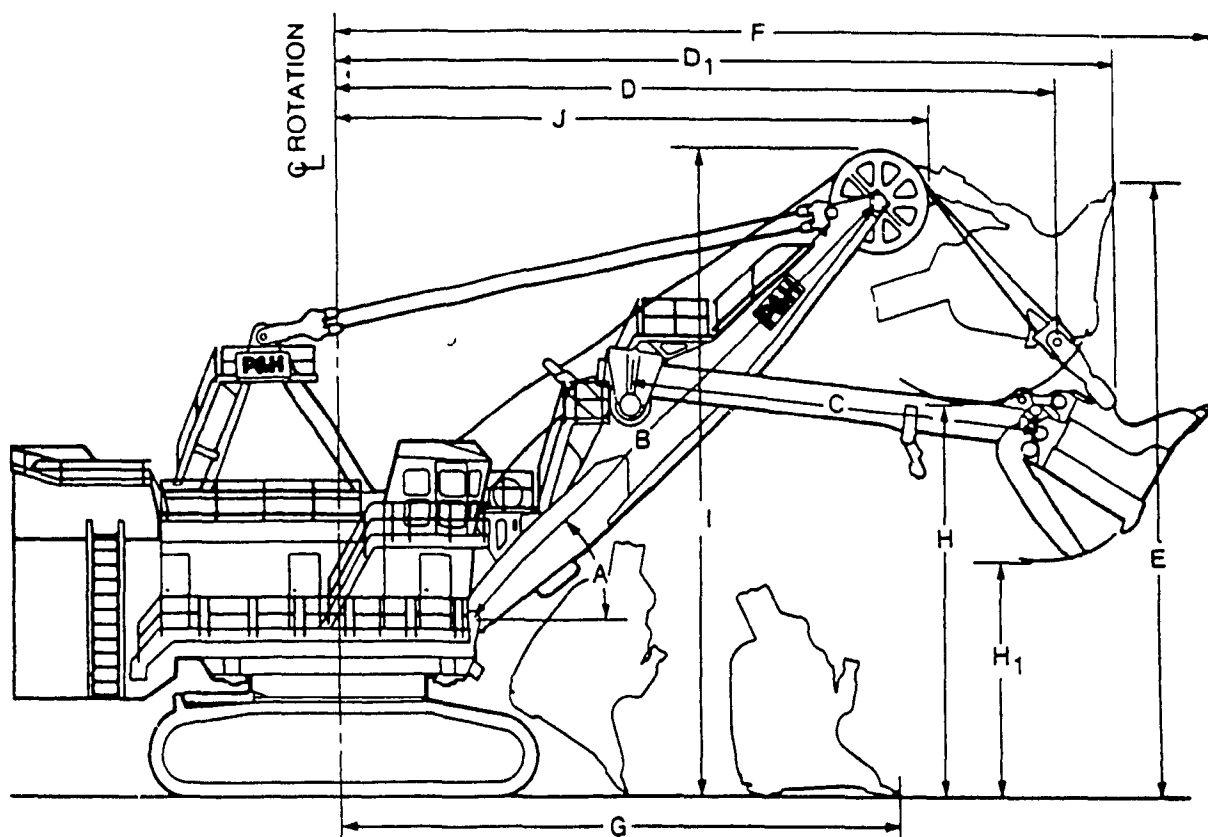
¹. The words "Electrotorque Control" and "Magnetorque Control" are registered trade marks of Harnischfeger, U.S.A.



General Dimensions

K	Radius of Rear End	29 ft. 4 in.	8.94 m.
L	Maximum Height Over Gantry	39 ft. 5 in.	12.01 m.
M	Overall Width of Crawlers (56" Shoes)	29 ft. 8 in.	9.04 m.
N	Overall Length of Crawlers	33 ft. 4 in.	10.16 m.
O	Ground Clearance	2 ft. 6 in.	.71 m.
P	Height — Ground to Bottom of Counterweight	8 ft. 4½ in.	2.46 m.
Q	Center of Rotation to Boom Foot Pin	11 ft. 6 in.	3.51 m.
R	Height — Ground to Boom Foot Pin	14 ft. 10½ in.	4.44 m.
S	Height — Ground to Top of Main Cab	25 ft. 8½ in.	7.84 m.
S ₁	Height — Ground to Top of Operator's Compartment	30 ft. 9 in.	8.99 m.
S ₂	Height — Ground to Top of Rear Canopy Section	27 ft. 2 in.	8.92 m.
S ₃	Height — Ground to Operator's Eye Level	25 ft. 8½ in.	7.77 m.
T	Width of Cab	28 ft. 0 in.	8.53 m.
T ₁	Width of Cab and Stair Platform	34 ft. 4 in.	10.46 m.
U	Overall Length of Main Cab	39 ft. 2 in.	11.94 m.
U ₁	Overall Length of Cab and Operator Compartment	41 ft. 0 in.	12.50 m.

Figure 2.3.1 - P&H 2800XP Design specifications (Source; P&H).



105J1464

Working Ranges

		45°	45°
A	Boom Angle		
	Dipper Capacity (Nominal)	33 cu.yd.	25.2 m. ³
	Dipper Capacity (Range)	33-60 cu.yd.	25.2-45.9 m. ³
B	Boom Length	58 ft. 0 in.	17.68 m.
C	Effective Dipper Handle Length	34 ft. 0 in.	10.36 m.
D	Dumping Radius at Maximum Lift	64 ft. 6 in.	19.66 m.
D ₁	Dumping Radius (Maximum)	67 ft. 6 in.	20.57 m.
E	Height of Cut (Maximum)	59 ft. 8 in.	18.19 m.
F	Digging Radius (Maximum)	77 ft. 7 in.	23.65 m.
G	Floor Level Radius	53 ft. 3 in.	16.23 m.
H	Dumping Height (Maximum) - Door Open	36 ft. 11 in.	11.25 m.
H ₁	Dumping Height at Maximum Radius - Door Open	19 ft. 0 in.	5.79 m.
I	Clearance Height of Boom Point Sheave	60 ft. 2 in.	18.34 m.
J	Clearance Radius of Boom Point Sheave	57 ft. 2 in.	17.42 m.

Figure 2.3.2 - P&H 2800XP Operating specifications (Source; P&H).

velocity controlled DC work motion drive motors; hoist, crowd, swing and propel (P&H tech. bull. X-818 and X-849).

The Fundamental configuration of the Electrotorque Control system is illustrated in Figure 2.3.3. The main incoming pit AC power is stepped down by a main transformer to practical working voltage of 600 volts. This 2000 KVA main transformer supplies power for the motion motor armatures, through two three phase 600 volt secondaries (Figure 2.3.4).

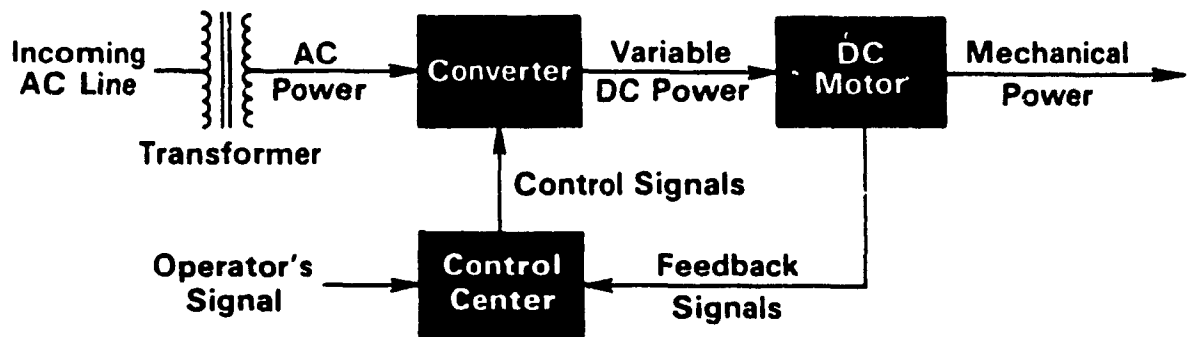


Figure 2.3.3 - Basics of the Electrotorque control system.

The stepped down 600 volt AC power is then routed to the converter cabinets for the conversion of power for the motor armatures. Here AC power supply is converted to DC operating power, and is supplied in a controlled manner to

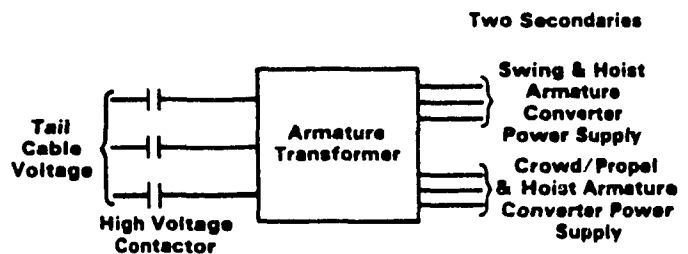


Figure 2.3.4 - Secondary transformers for work motion motors.

the motion drive armatures of the shovels main motors.

A control centre regulates DC power delivery to each of the main motors. This computer-like control centre responds to operator commands for various motions, by comparing operator demands with the speed (voltage) and torque (current) feedback signals from the respective motors at the time of the command. The motor then adjusts to voltage and current outputs synonymous with motor speeds and torques requested by the operator.

It is important to state that the P&H 2800XP is one of the most common electric mining shovels in use today. Approximately 75% of all electric mining shovels used in the surface mining industry are manufactured by P&H Harnischfeger, of which 95% are Electrotorque controlled models. Other major manufactures of electric mining shovels; Bucyrus Erie and Marion, have held 15% and 10% of the market respectively, during the past decade (L. Runno, pers. comm., 1989). This is significant in that seven of every ten electric mining shovel in use worldwide, employ electrical and mechanical design systems identical to those of the shovel investigated for the purposes of the current research. The monitoring strategy (i.e. instrumentation) and interpretative procedures established during the current research, are therefore, well suited to widespread utilization.

The remaining shovels manufactured by P&H are the Magnetorque controlled models; 1400 to 2100 series. These are smaller shovels with nominal dipper capacities ranging from 4.5 to 25 cubic yards (3.4 to 19 cu. m)

2.3.3 Basics of Shovel Operation.

The complete loading cycle of a shovel involves three separate actions or sub-cycles - digging, swinging to the truck, dumping the load in the truck and swinging back to the bank (muckpile). Of these, the only true work producing element is the digging cycle, which represents the amount of time the dipper spent filling in the bank. The swing and dump cycles merely provide a materials handling service in transporting the excavated material to the haulage vehicles.

The dig component itself is a product of two elements - hoisting and crowding. The hoist action is responsible for pulling the dipper up through the muckpile, while the crowd acts to provide initial thrust energy on the dipper to force it into the toe of the muckpile (Figure 2.3.5). During hoisting the crowd action maintains the dipper, through minimal effort, at a suitable depth of penetration in the bank throughout the digging cycle. If the crowd element for each cycle is equivalent i.e. same optimal depths of penetration, then the hoist cycle is the primary element behind shovel productivity. That is, the amount of available hoist power during the dig cycle will determine how much and how fast material is loaded in the dipper by the shovel (P&H tech. bull. X-730 and X-735). The effective digging power of the shovel is determined by the hoist cable drum torque and the feet per minute of drum circumferential speed:

Digging power = Hoist drive h.p. = torque X speed per unit time.

There are four classes of digging generally recognized by the open pit mining industry (after P&H tech. bull. X-642):

1. Very Hard Digging: Hard rock overburden requiring heavy blasting and yielding coarse fragmentation with considerable fragment interlock. Typical weight of 4000 lbs. or more per cubic yard in the dipper.
2. Hard Digging: Typical rock overburden, blasted material yielding moderate fragmentation and considerable fragment interlock. Materials which weigh between 3000 to 4000 lbs. per cubic yard in the dipper.
3. Medium Digging: Dry materials requiring little or no blasting and having weights of between 2000 and 3000 lbs. per cubic yard in the dipper.

4. Easy Digging: Free running loose materials such as coal, exceptionally well fragmented from blasting and having material weights per cubic yard in the dipper of up to 2000 lbs.

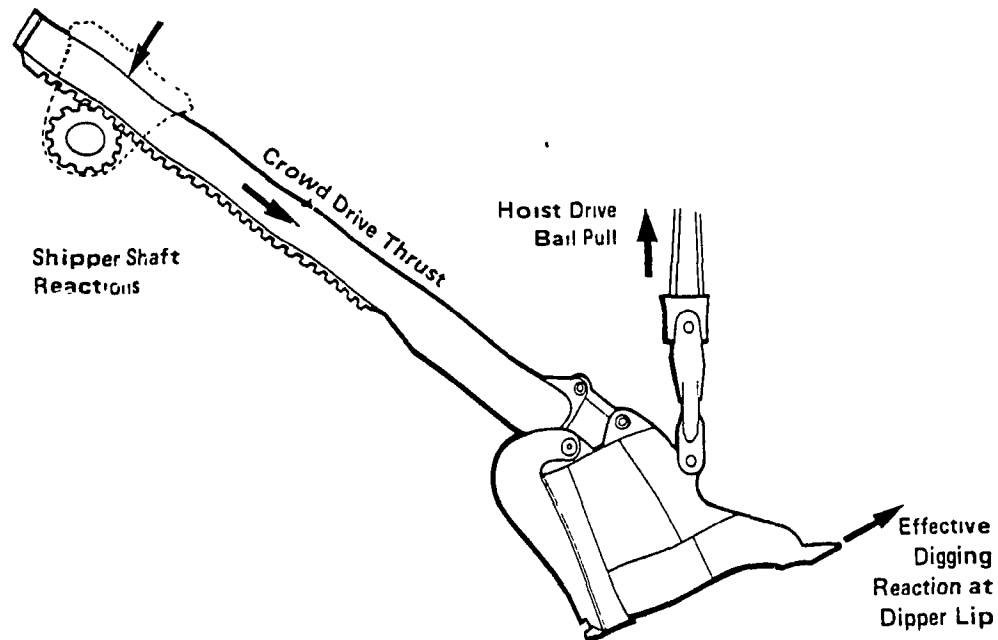


Figure 2.3.5 - Hoist and crowd forces during dig cycle (Source; P&H)

The above digging classifications have been established on the basis of two bail pull (hoist motor Torque) effort factors; 1) weight of the fragmented material and 2) the additional bail pull required to overcome fragment interlock. The material weight after it has passed into the dipper is a calculable static bail pull load. The added bail pull required to free interlocked fragments is a dynamic, transient load imposed above and beyond that required to handle the static material weight. This is an incalculable element which varies greatly between the range of easy to very hard digging conditions. Note that once out of the bank only the static material loads act on the dipper (P&H tech. bull. X-642).

The transient dynamic loads imposed on the dipper by material fragment interlock are managed by added digging power drawn from the system in the form of surges of mechanical energy. The incidence of which is flagged by Torque peaks, equivalent to the maximum bail pull of the hoist motor.

Dipper Filling.

The ability of the hoist motor to maintain high bail pull at close to constant speeds, is significant to the manner in which the dipper fills during its travel through the muckpile. Upon initial penetration into the muckpile the forward pitch of the dipper lip directs material along the dipper bottom. As the dipper rises through the muckpile, and the angle with which the dipper lip intersects the muckpile changes from nearly horizontal to nearly vertical, material continues to readily flow downward and toward the back of the dipper. Initially, the flow of material from the dipper lip to the back of the dipper results in good fragment distribution throughout the dipper body. Dippers fill easily for the first 75 to 90% of their rated capacity for easy to very hard digging respectively. To fill the remaining 10 to 25% of the dipper requires a high thrust against the lip to force the fragment flow into the final open space of the dipper back.

2.3.4 Hoist Motors.

The 2800XP uses two identical extra heavy duty D.C. mining shovel motors for hoist drive power. The motors are placed one fore and one aft of the hoist drum gear as shown in Figure 2.3.6, and develop a total of 1450 h.p. at 475 Volts, with a peak H.P. of 2250. Two smaller motors are used in place of a single larger motor because they offer lower mechanical inertial resistance to changes in their state of motion. The kinetic energy stored as rotational energy in a spinning motor armature increases with the square of its radius. Hence, the lower inertial resistance of two smaller armatures working in unison can accommodate faster acceleration and deceleration, bringing the hoist drum to maximum speed quicker than the single larger armature of a single motor

of equivalent horsepower.

Figure 2.3.7 shows that the armatures of the two hoist motors are connected in series. The configuration results in equal sharing of current and hence, equal production of torque from each motor.

The lowering of the dipper is by gravity controlled, modulation of the hoist motor. During dipper lowering the kinetic energy from the controlled gravity fall of the dipper, is converted by the hoist motors, working in a regenerative capacity, into usable A.C. current fed back into the mine main A.C. line.

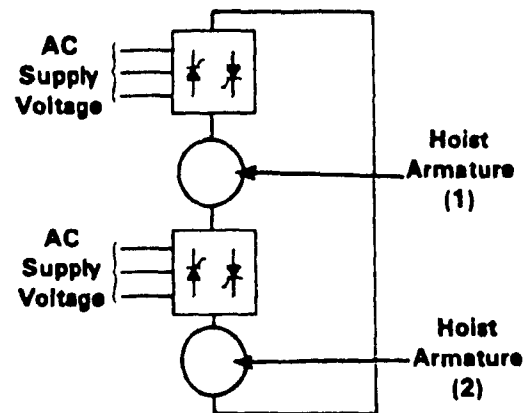


Figure 2.3.7 - Series connection of hoist motors (Source; P&H)

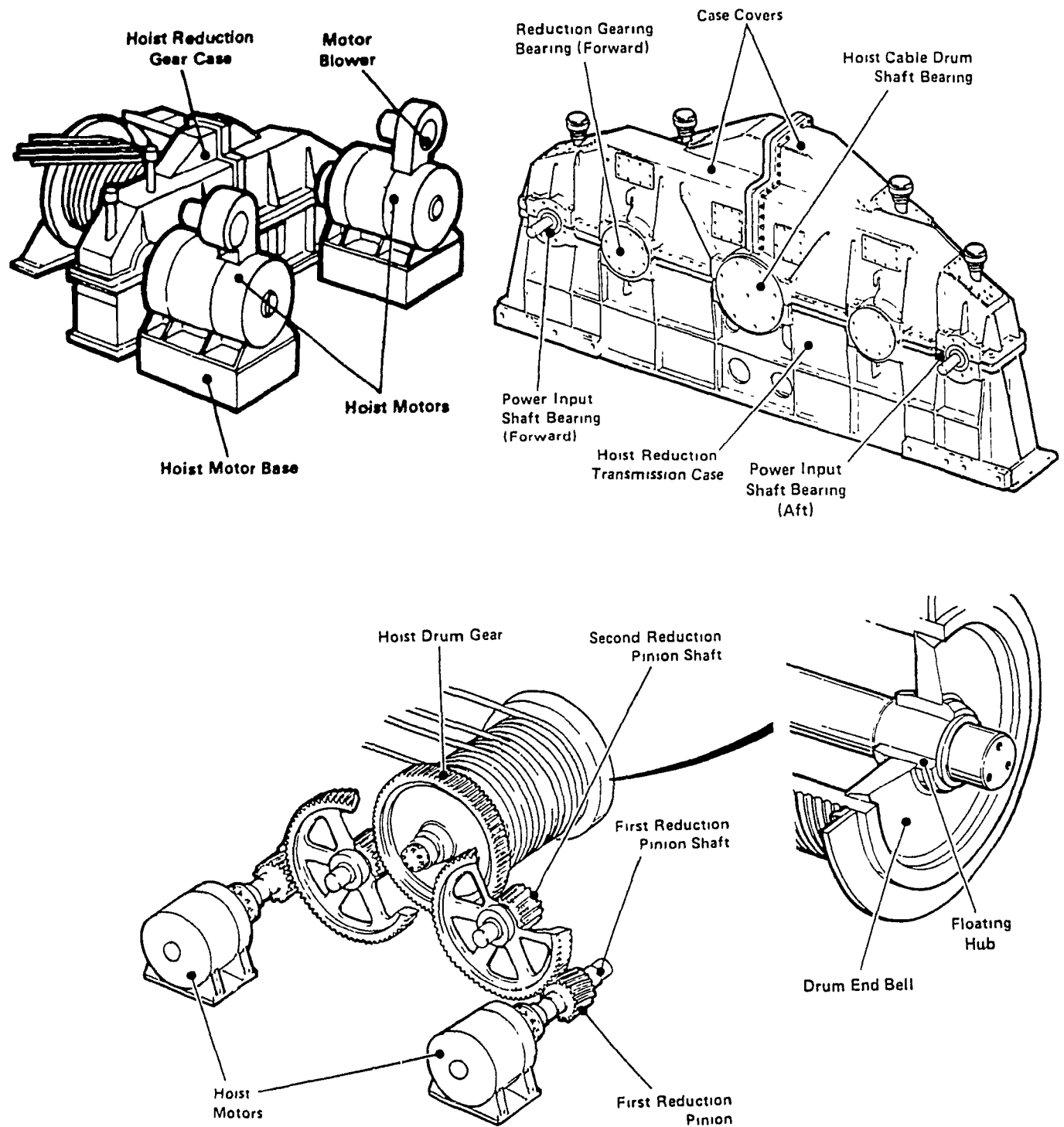


Figure 2.3.6 - Hoist motors and drive assembly (Source; P&H).

2.3.5 Crowd Motor.

The crowd action of the shovel is powered by a single heavy duty D.C. high torque motor, which at 475 Volts develops 400 H.P. and peak H.P. of 600. The motor is mounted near the base of the shovel boom (Figure 2.3.8).

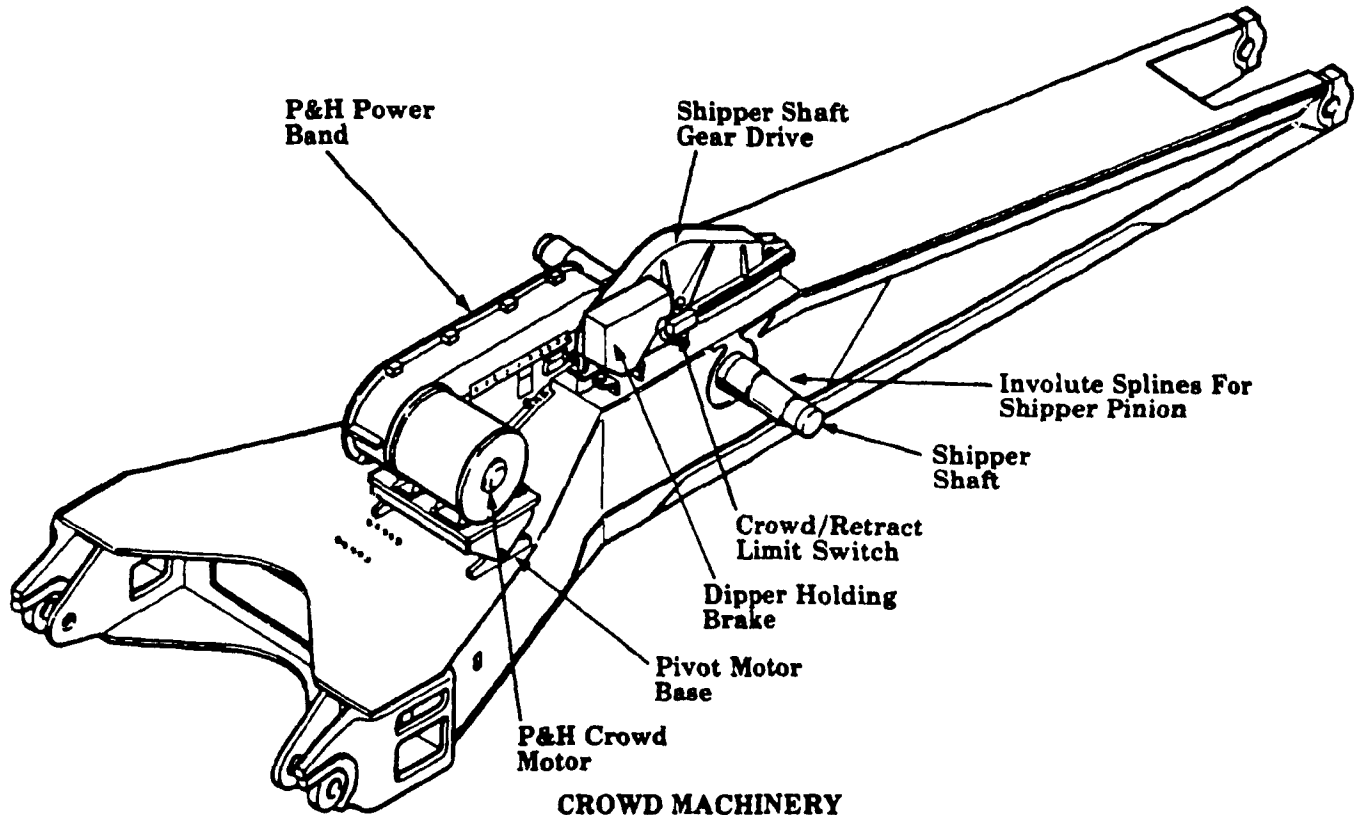


Figure 2.3.8 - Crowd drive assembly (Source; P&H)

It is important to appreciate the magnitude of the transient strains imposed upon the crowd motor during the crowd cycle. Thrusting the dipper into a mass of irregularly broken material and often striking un-fractured projections will result in frequent massive shock loadings to the entire crowd drive machinery and motor. In order to diminish the damaging effects of

sudden impact loading, a V-belt transmission is used to transfer power from the crowd motor to the shipper shaft gear train (Figures 2.3.9 and 2.3.10). The belts absorb the high amplitude energy of the shock from sudden impact loads, dampen it in milliseconds and transfer the dissipated energy to the crowd motor and machinery at a significantly reduced level (P&H tech. bull. X-693)

A concern in the proper operation of the shovel is that operators do not apply excessive crowd during the dig cycle. This practice will result in an increased incidence of dipper stalls in the bank, imparting undue stresses to the electrical and mechanical machinery. To prevent this practice, all P&H Electrotorque model shovels (model series 2300, 2800 and 5700) are equipped with a control governor that limit the amount of available crowd force as a function of hoist force used. If an operator is using less than 60% of available hoist power no more than 50% of total available crowd force can be accessed. Crowd force increases linearly with hoist force once greater than 60% of available hoist power is engaged. Apart from preventing dipper stall in the face from too low a travel speed through the bank, the governor limits the application of excessive crowd into the toe of the muckpile at the onset of the dig cycle.

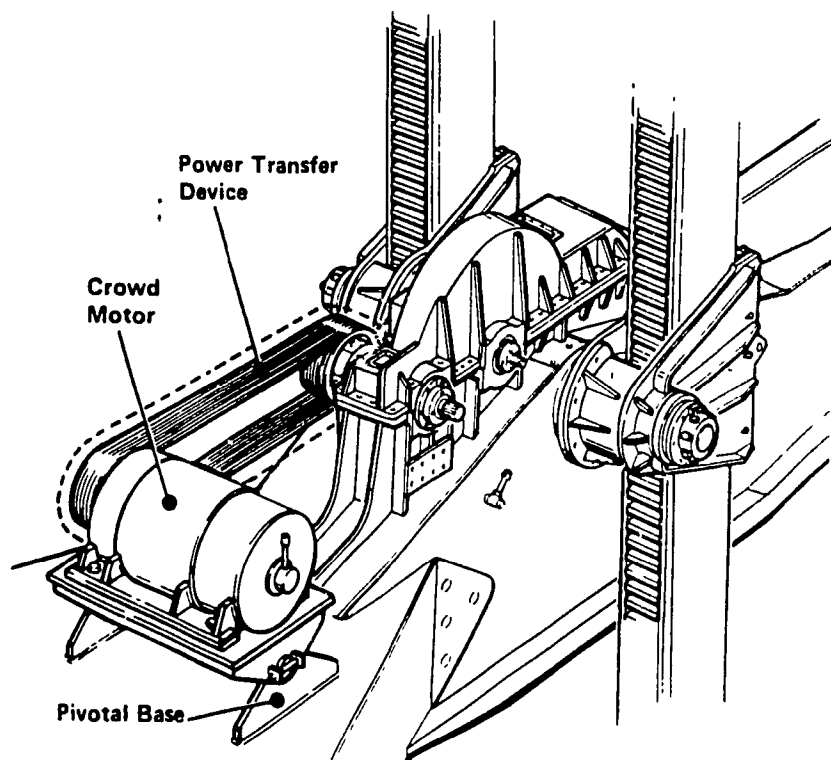


Figure 2.3.9 - Crow power band transmission (Source; P&H)

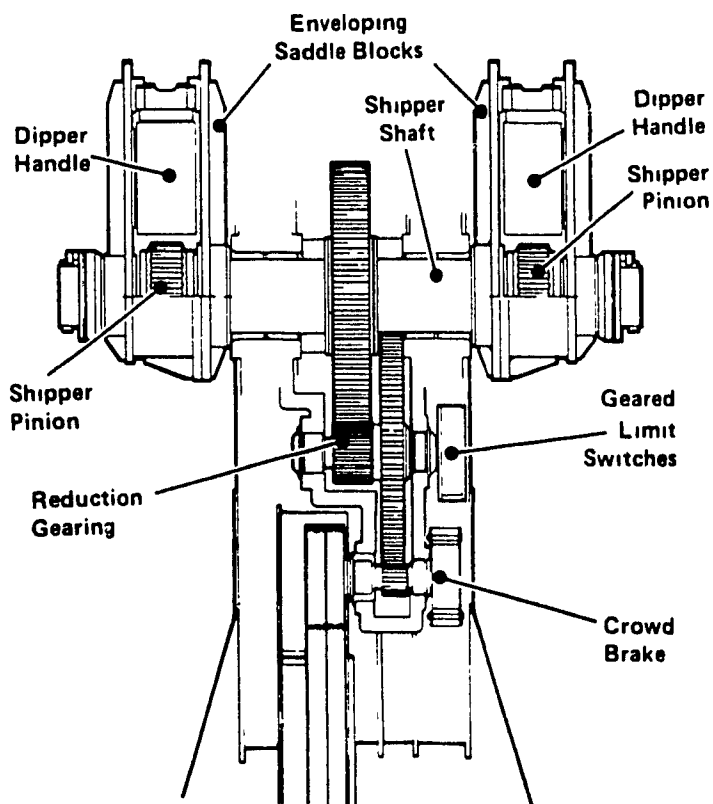


Figure 2.3.10 - Crow drive mechanism (Source; P&H).

3.0 INSTRUMENTATION:

3.1 INTRODUCTION.

The single greatest concern in the development and progression of this project has been the acquisition of shovel performance data. Shovel performance as it pertains to this research is defined as a quantifiable measure of the shovel's behaviour while engaged in the digging portion of the shovel production cycle. More precisely, instrumentation was required to record the action of the dipper in terms of its fundamental force components - hoist and crowd. In addition, a knowledge of the dippers position in the bank during the dig cycle permits resolution of these forces. As discussed in the previous chapter, the dig component of the shovel cycle is the one most closely related to fluctuations in productivity as a result of variations in the nature of the material the shovel is digging. Instrumentation has therefore centred around recording the actions of the excavator while engaged in the dig cycle component of the production cycle.

In accomplishing the instrumentation requirements, two separate systems were implemented. The first and most comprehensive system employed was a commercially available shovel monitoring system - General Electric's SPM-8000 Shovel Production Monitor. The SPM-8000 provided most of the data required for the research with the exception of crowd motor performance data. The overall complexity of the SPM-8000 precluded its alteration to incorporate the collection of crowd motor data. A second system, independent of the SPM-8000, was designed around a commercially available data logger (the Polycorder by Omnidata) to collect data on the crowd motor.

In addition, a video-camera, mounted on a specially designed bracket in the operators cab, was engaged in conjunction with the systems discussed above. This served to establish a visual record

against which potential ambiguities in the data collected through monitoring systems could be compared. Furthermore, individual cycle times for other aspects of the shovel production cycle have been determined through subsequent time studies conducted on the video taped records. In all therefore, three separate monitoring devices are seen as contributing to a complete instrumentation of the shovel, from which all aspects of shovel behaviour have been studied. A discussion of each of the systems employed follows.

3.2 GENERAL ELECTRIC'S SPM-8000 SHOVEL PRODUCTION MONITOR.

The SPM-8000 is a micro-processor based shovel production monitoring system designed for use on board D.C. powered electric mining shovels. Several operating parameters are monitored by the system for conversion to data of benefit to both mine production and maintenance personnel. The shovel operator has available, real time feed back on dipper weight, swing angle, load time and present weight of material in the truck currently being loaded, for each dipper pass. Monitored data can be retrieved from the system and stored on pocket sized data cartridges. These non-volatile solid state memory cartridges stored up to 128Kb of monitored data for subsequent analysis. The data stored in these cartridges is transferred to mine computer facilities through a data card reader supplied with the system.

The entire SPM-8000 system is designed around the processor console. The processor console is built around a CPU (central processing unit) comprising a 16 bit Intel 8086 processor and an Intel 8087 co-processor. The processor console is encased in a sealed NEMA 12 gasketed enclosure of dimensions; 36"H x 24"W x 20"D. The unit is braced against shock and vibration and effectively protects against dust, grease and moisture. The systems power supply is protected from voltage fluctuations and noise.

The fundamental configuration of the SPM-8000 system is represented in Figure 3.2.1. Various sensors for position, motor EMF and current, relays etc., provide real time data to the systems processor console. Here, the systems software converts sensor monitored data to information recognizable to the operator such as dipper weight. The operators console, on which this information is displayed, is equipped with a slot into which a data cartridge may be inserted. The data collected on the cartridge is processed on a PC in the mine offices.

A supplement to the design of the SPM-8000 system was the addition of a dot matrix printer to which data from the processor console may be spooled. The data sent to the printer is the production statistics for each shovel load cycle ie. dipper load weights and cycle times. The scope and nature of this data is unique and is not available through either the operators console or through the data collection cartridges.

The SPM-8000 is a very complex and comprehensive monitoring system. It is intended therefore, to address only those facets of its operation that have been engaged to assist in the outcome of this investigation. The systems features that will be described in more detail are; production data acquisition and sensor data acquisition.

3.2.1 Production Data Acquisition.

Production data refers to the data processed through the processor console and sent to the dot matrix printer. As stated the production data is printed out for each complete shovel loading cycle and includes information on the following aspects of the shovel operation:

- 1). Time at which readings were taken.
- 2). Load cycle time.
- 3). Fill cycle time.
- 4). Swing time.
- 5). Dump time.
- 6). Delay time.
- 7). Propel time.
- 8). Swing angle.

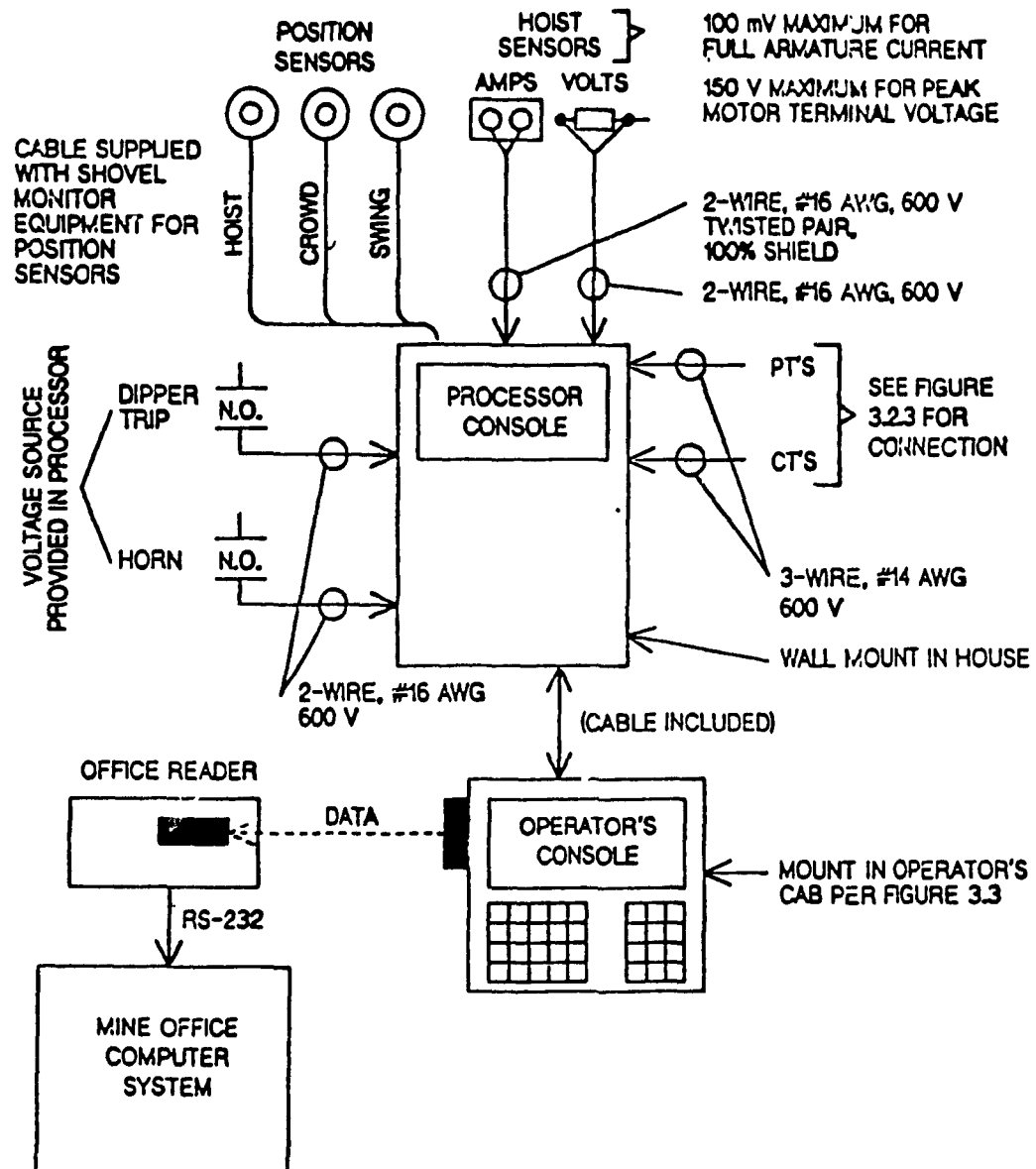


Figure 3.2.1 - General Electric SPM 8000 shovel monitor system configuration.

- 9). Load weight.
- 10). Fill RMS current for the cycle
- 11). Cycle RMS current.

A sample portion of the print out of the above production data is presented in Figure 3.2.2. Note that there exists a blank column under the heading "TRK". This column is ideally where the number of the truck being loaded during a load cycle would be registered. The only time the operator need interface with the shovel monitoring system is to enter the number of the truck he will be loading, upon the arrival of that particular truck. Complicating factors such as the lack of a clear view to those trucks (and hence their numbers) loaded on the operators left however, prevented the entry of truck numbers into the system. The lack of such data prevents determining truck cycle times. This minor deficiency is of little consequence to the current investigation which concern itself with an analysis of excavator performance rather than the cycle times of haulage equipment.

Several other ambiguities may be apparent upon a first inspection of the data presented in Figure 3.2.2. It may appear that load and fill time are measures of the same event with two separate times. Load time is a time interval recorded on the basis of the dipper sticks (crowd arm) position with respect to the horizontal. The interval is initiated after dumping on the return of the dipper from the truck at the instant the dipper sticks fall below -35° (with respect to the horizontal). Below this pre-set angle, all actions of the shovel are taken as associated with the loading process. Shovel fill time on the other hand is a component of load time, and is a measure of the time the dipper is engaged in the actual digging process. This interval is initiated whenever sensors on the hoist armature voltage indicate an EMF in excess of 100 volts. The fill and load time intervals are terminated when the shovel begins its swing to the truck. The swing times indicated in Figure 3.2.2 are the intervals for each cycle during which the shovel rotating at greater than 3° per second.

*** Reset truck weight key pressed ***
8/18/88

GENERAL ELECTRIC SHOVEL PRODUCTION MONITOR

TIME	TRK	LOAD	FILL	SWNG	DUMP	DLAY	PRPL	FFMS	CRMS	SW AIR	WGT
11:29:39	..		21.4	..	0.4	11.7	113.6	1145	1158	60.5	41.2
11:30:11	..	17.6	11.4	10.5	3.0	0.0	0.0	1313	1009	54.3	33.4
11:30:38	..	14.5	9.3	9.8	3.1	0.0	0.0	1260	968	44.7	32.5
11:31:06	..	14.4	7.9	10.2	3.4	0.0	0.0	1258	961	60.3	36.6
11:31:41	..	17.8	7.7	12.7	3.5	5.2	0.0	1014	784	53.8	23.9
11:32:48	..	16.7	9.5	22.3	4.7	14.3	0.0	1185	702	50.5	32.7
11:33:15	..	14.9	7.8	9.0	2.8	0.0	0.0	1320	1024	35.0	39.2
11:33:47	..	14.7	7.2	10.1	3.0	0.0	0.0	1165	942	45.4	34.9
11:34:12	..	14.9	8.8	11.4	3.2	0.0	0.0	1263	945	56.8	31.8
11:34:40	..	17.6	8.6	10.6	3.3	0.0	0.0	1121	896	50.3	31.5
11:36:02	..	26.7	21.5	32.8	3.3	6.7	13.1	1143	797	59.3	30.5
11:36:29	..	15.7	9.0	8.2	2.6	0.0	0.0	1422	1036	27.1	39.8
11:36:57	..	16.5	8.0	8.6	3.1	0.0	0.0	1403	1099	41.6	35.1
11:37:24	..	14.3	8.3	9.2	3.7	0.0	0.0	1337	948	50.7	34.0
11:37:59	..	15.3	8.6	14.7	3.5	2.1	0.0	1489	1072	61.8	41.3
11:38:26	..	13.4	7.6	8.0	2.7	0.0	0.0	1414	1091	43.9	42.7
11:38:57	..	15.9	8.0	10.1	3.3	1.7	0.0	1444	1039	56.0	37.7
11:39:21	..	11.9	6.2	8.0	3.4	0.0	0.0	1265	125	57.4	31.1
11:40:09	..	12.9	7.2	14.3	4.6	15.8	3.0	1241	930	45.5	37.4
11:40:34	..	13.5	7.9	7.7	3.4	0.0	0.0	1411	1024	51.8	40.4
11:41:04	..	15.0	7.3	9.5	3.5	2.3	0.0	1317	991	50.7	31.5
11:41:32	..	14.8	9.8	10.0	3.4	0.0	0.0	1551	1094	50.7	37.4
11:42:16	..	27.1	22.1	13.5	3.9	0.0	0.0	835	791	40.4	17.1
11:42:41	..	13.6	8.3	9.0	2.9	0.0	0.0	1339	909	47.9	36.6
11:43:11	..	14.7	7.0	10.4	3.8	0.0	0.0	1425	1067	57.1	39.8
11:43:40	..	14.5	8.9	10.8	3.5	0.0	0.0	1284	950	57.0	39.2
11:44:10	..	16.1	9.1	7.3	4.5	0.0	0.0	926	741	57.1	37.1
11:45:05	..	17.7	11.5	18.7	4.0	17.7	0.0	1540	1044	57.1	37.0
11:45:34	..	16.5	8.0	9.0	3.1	0.0	0.0	1313	1030	57.1	37.1
11:46:04	..	14.1	10.5	0.0	2.4	0.0	0.0	1354	85	57.1	37.0

Figure 3.2.2 - Sample of General Electric production data.

With the data provided from the printout it is possible to calculate the complete cycle times for each load handled by the shovel. This can be accomplished in one of two ways. First, by subtracting the time at which a record was established from the record before it. Or secondly, by summing the times for the three elements of a complete shovel cycle - load time plus swing time plus dump time. One would not include fill time since it is already incorporated as part of the load time.

There are three significant disadvantages associated with the data sent to the printer by the processor console. The first is a result of the positioning of the printer itself. The printer, an Epson FX-80, a simple non commercial unit, was installed mounted vertically on the back wall of the shovel machine house next to the processor console. While the printer was installed in a protective metal enclosure (non-gasketted), the rigors imposed on it from the surrounding environment caused it's failure on several occasions. The main impediment to its proper operation was the high concentrations of particulate matter in the air (coal and rock dust) that would accumulate on the metal bar along which the print head travels. The frictional resistance caused by this accumulation was sufficient to stop the print head from moving, resulting in the printer switching itself off-line. The production data for each load is memory resident only until the next load replaces it with the current values. Hence, if no printer is available to record the down loaded data for each load, the data is permanently forfeited. It was important therefore to ensure the proper operation of the printer throughout monitored periods. This was accomplished by cleaning the metal bar along which the print head travels with an aerosol contact cleaner for electronic components. These "cleanings" had to be conducted every half hour.

The second problem associated with the printed production data was that its collection was temporarily halted during periods of monitoring other aspects of shovel performance. When a data cartridge was inserted for data collection on position

data and motor performance, (section 3.3), streaming of production data was temporarily halted.

The final difficulty affiliated with the printed data is that it requires complete rehandling in order to analyze it. There are up to 280 loads handled in a typical monitoring period of four hours. This translates into close to 3000 figures per day of monitoring, that have to be manually reentered into a suitable spreadsheet or data base in order to compute any statistics based on the data collected by the printer. The production data in its rehandled format is presented in chapter 10 - data analysis.

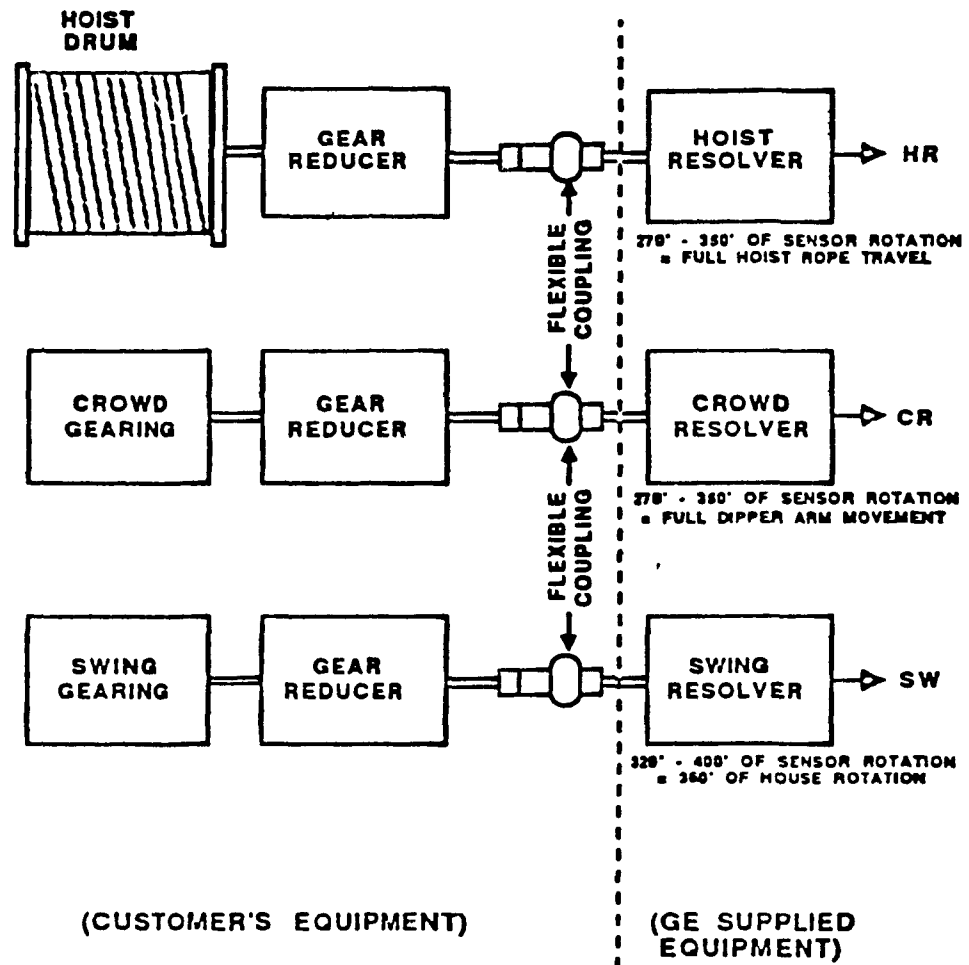
3.3 SENSOR DATA ACQUISITION.

The SPM-8000 is equipped with sensors to measure both position and power parameters of the shovel during its operation. Position sensors measure the hoist, crowd and swing motions of the shovel. Power or force, is measured through sensors for hoist armature voltage and current. The dipper trip and horn sounding are monitored through relay contacts.

3.3.1 Position Sensors.

Hoist, crowd and swing positions sensors are all highly linear synchro resolver-devices which have no effective dead band. These resolvers have an accuracy of 0.02% which allows determining dipper position to within 1/8 of an inch per 30 feet of hoist rope travel. Positions are determined ratiometrically by comparing output signals. The resolvers are therefore immune to noise as well as signal fluctuations from line voltage drops.

Hoist and crowd resolvers are connected to gear reducers which are in turn connected to intermediate crowd or hoist gears (Figure 3.3.1). The gear reductions are selected so as to permit at least 270° of resolver rotation but not more than 350°, to account for full travel of the hoist rope or the crowd arm. The hoist resolver permits determining the length of the hoist rope at any time. The crowd resolver determines the extension of the



SHOVEL MONITOR POSITION SENSOR INSTALLATION

Figure 3.3.1 - Configuration of G.E. position transducers.

crowd arm.

The swing sensor is attached to a gear reducer that allows a full 360° rotation of the shovel to be represented by between 320° to 400° of sensor rotation. The system determines and uses only relative swing angles.

3.3.2 Electrical Sensors.

The SPM-8000 confines the instrumentation of electric motors to the hoist drive. The system monitors hoist armature voltage, hoist armature current and hoist field current. The input signals are filtered and amplified to +/- 5V and current shunts are designed for a maximum signal of 100mV.

Hoist armature current is measured across a current shunt. This low level millivolt signal is sent to the processor console where it is isolated, conditioned by an analog module in the processor, and converted to a standard analog +/- 5VDC signal. The SPM-8000 uses the monitored hoist armature current as a primary signal from which dipper loads are determined.

Hoist armature voltage is measured through a resistor bridge. The analog signal is treated as per the hoist current to produce a standard analog level +/- 5VDC signal. The SPM-8000 use the hoist armature voltage signal to determine hoist acceleration and direction.

Hoist field current is monitored through a current shunt and processed as per the hoist armature current to yield an analog +/- 5VDC signal. Hoist field current is monitored to determine when the hoist motor is operating in a weak or strong field mode. The weak field mode corresponds to the cycle component during which controlled gravity lowering of the dipper occurs.

The full nature of the signals monitored by both the position and electrical sensors of the SPM-8000 will be addressed in detail in chapter 10 - data analysis.

3.4 PRODUCTION DATA HANDLING.

Production data collected through the systems printer requires extensive rehandling in order to make use of it. This data as mentioned, is only collectable in a printed format. In order to analyze this information, the data collected from each day of monitoring, must be manually re-processed into a usable computer format. For the purposes of this investigation a Lotus spreadsheet has proven an adequate means of data manipulation. Once entered into the spreadsheet, the statistics which can be derived from the production data are:

- 1). Total monitored time.
- 2). Total time loading.
- 3). Total time filling.
- 4). Total time swinging.
- 5). Total time dumping.
- 6). Total delay time.
- 7). Total propel time.
- 8). Percentage of time spent loading.
- 9). " " " " filling.
- 10). " " " " swinging.
- 11). " " " " dumping.
- 12). " " " " in delays.
- 13). " " " " propelling.
- 14). Average total shovel cycle time.
- 15). " load cycle time.
- 16). " fill cycle time.
- 17). " swing cycle time.
- 18). " dump time.
- 19). " delay time.
- 20). " propel time.
- 21). " swing angle.
- 22). " fill RMS current.
- 23). " cycle RMS current.
- 24). " load weight.
- 25). Total number of loads (dipper passes)
- 26). Total tons hauled during monitored period.
- 27). Total tons hauled per hour and/or tons handled per minute of monitored time.

It is known (based on mine production records, past monitoring experiences (time studies) and operator input), that

for at least 95% of the trucks, four dipper loads are required to approximate the 170 ton rated capacity of the trucks. With this knowledge it is possible to compute the following statistics based on the monitored production data:

- 28). Average load per truck (based on four passes).
- 29). Average load time per truck (minutes/truck).

3.5 SENSOR DATA HANDLING.

Sensor data, both position and electrical, is monitored solely for processing by a subsystem of the SPM-8000 which is responsible for determining weights of suspended dipper loads. This subsystem or sub-menu is called the "Weighment Diagnostics Menu". Access to all sub-menus of the system is restricted to authorized personnel, who must enter a pass-code in the form of a series of keystrokes at the operators console. The Weighment Diagnostics Menu is typically engaged only by General Electric field engineers to observe various aspects of the weighing system while in operation. One of several options available to the user through the Weighment Diagnostics Menu is - "Cartridge Data Capture". This option allowed the capture of real-time sensor data for later analysis. It should be noted that when this option is engaged, all other data capture and processing by the system is inhibited.

The sensor data is collected on the read-write data cartridges as discussed earlier. The cartridge is inserted in a slot in the side of the operators console, while in the systems "Cartridge Data Capture" mode. A series of key strokes then instructs the system to stream all sensor data, at 100 millisecond intervals, to the data cartridge. These 128K cartridges can store approximately 10 minutes of sensor data. The data scanned and collected every 100 milliseconds includes the following:

- Hoist rope position.
- Crowd arm extension.
- Dipper trip relay.
- Hoist armature voltage.
- Hoist armature current.
- Swing position.
- Hoist field current.

The above data is retrieved from the data cartridges by a data cartridge reader connected to a personal computer. The personal computer, loaded with SPM-8000 software for the purpose, reads the raw sensor data into a data file format. These data files are always of fixed size 132096 bytes of binary data. This is the most convenient format in which to store the monitored data. The sensor data requires further processing in order to analyze it. A facility in the SPM-8000 data cartridge software processes the data files and converts their contents to ASCII format. These ASCII files are usually larger than 350Kb for ten minutes of monitoring sensor data. The ASCII data is assigned a .PRN extension permitting import into a Lotus spreadsheet where it can be analyzed.

3.6 THE "OMNIDATA POLYCORDER".

The overall force exerted by the shovel during the dig cycle is the product of two forces - hoist and crowd. The SPM-8000 provided comprehensive instrumentation on the hoist motor, but because of its design (ie. determining suspended loads from hoist motor response) it required no crowd motor instrumentation. It was necessary therefore, to incorporate into the overall monitoring strategy, a secondary instrument dedicated to monitoring crowd motor responses. The instrument employed for this purpose was the "POLYCORDER" manufactured by Omnidata International Inc. of Logan, Utah, U.S.A..

The POLYCORDER is a combination of a portable computer, data-logger and data entry terminal functioning under its own menu

driven operating system and programming language. It is able to read and store data from a variety of sensor input devices. The unit is equipped with a serial RS-232-C communications port through which data stored internally on solid state RAM (16K) can be transferred. The unit is portable measuring 8" x 5" x 3", is water and dust proof and can accommodate up to 10 A/D inputs simultaneously.

3.6.1 Programming the Polycorder.

It was desired to programme the Polycorder so that it would automatically scan sensors connected to the crowd motor. The data collected on crowd motor performance by the Polycorder was to be used in conjunction with the SPM-8000 data to form a comprehensive data set on all shovel performance parameters. It was necessary therefore, to insure that the two data sets (Polycorder and SPM-8000) could be meshed with one another, or synchronized in the same real-time frame. The G.E. sensor data is not "flagged" with an associated time for each reading. The best way to insure correlation of the two data sets was to duplicate the recorded signals. It was decided to monitor hoist armature voltage and current by the polycorder in addition to crowd armature voltage and current. The hoist motor traces common to both instruments could then be used as a means of correlating crowd motor responses with the SPM-8000 hoist and position traces.

The Polycorder is programmed in its own operating language called "POLYCODE". Polycode is a series of instructions, which can be given in alphabetic or numeric form to sequence a program. Given in Appendix A, is the programme written in POLYCODE to scan four analog inputs (hoist armature voltage and current and crowd armature voltage and current) and record the readings in a data file.

3.6.2 Instrumentation of Hoist and Crowd Motors with the Polycorder.

The P&H 2800XP electric mining shovel is equipped with a comprehensive control system. The control system, housed in its own cabinet in the machine house, contains all electrical components associated with the control of each motion motor - hoist, crowd/propel, and swing (Figure 3.6.2).

Each work motion has a separate control centre with individual armature logic control, structured around a series of solid state printed circuit boards. Each of the circuit boards is housed in a protective moulded frame, with test points exposed on the frame's front. These test points are used primarily as a convenience when testing or trouble shooting by maintenance personnel. A series of armature control logic circuit boards for the hoist motor is presented in Figure 3.6.3. There are ten separate circuit boards in the complete logic control strategy for each of the four work motions.

The advantage afforded by such a complete control system made it possible to instrument for hoist armature voltage and current as well as crowd armature voltage and current directly from the control frames for the respective motors.

The circuit diagrams for hoist armature control frame and the crowd/propel control frame are presented in Figures 3.6.4 and 3.6.5 respectively. An examination of the hoist armature control frame diagram (Figure 3.6.4) reveals that the armature voltage feedback signal for the hoist motor can be accessed through test point 18 on the hoist voltage regulator circuit board (QALB 214). The armature feedback signal for hoist is available through test point 6 on the hoist current regulator circuit board (YXR 104).

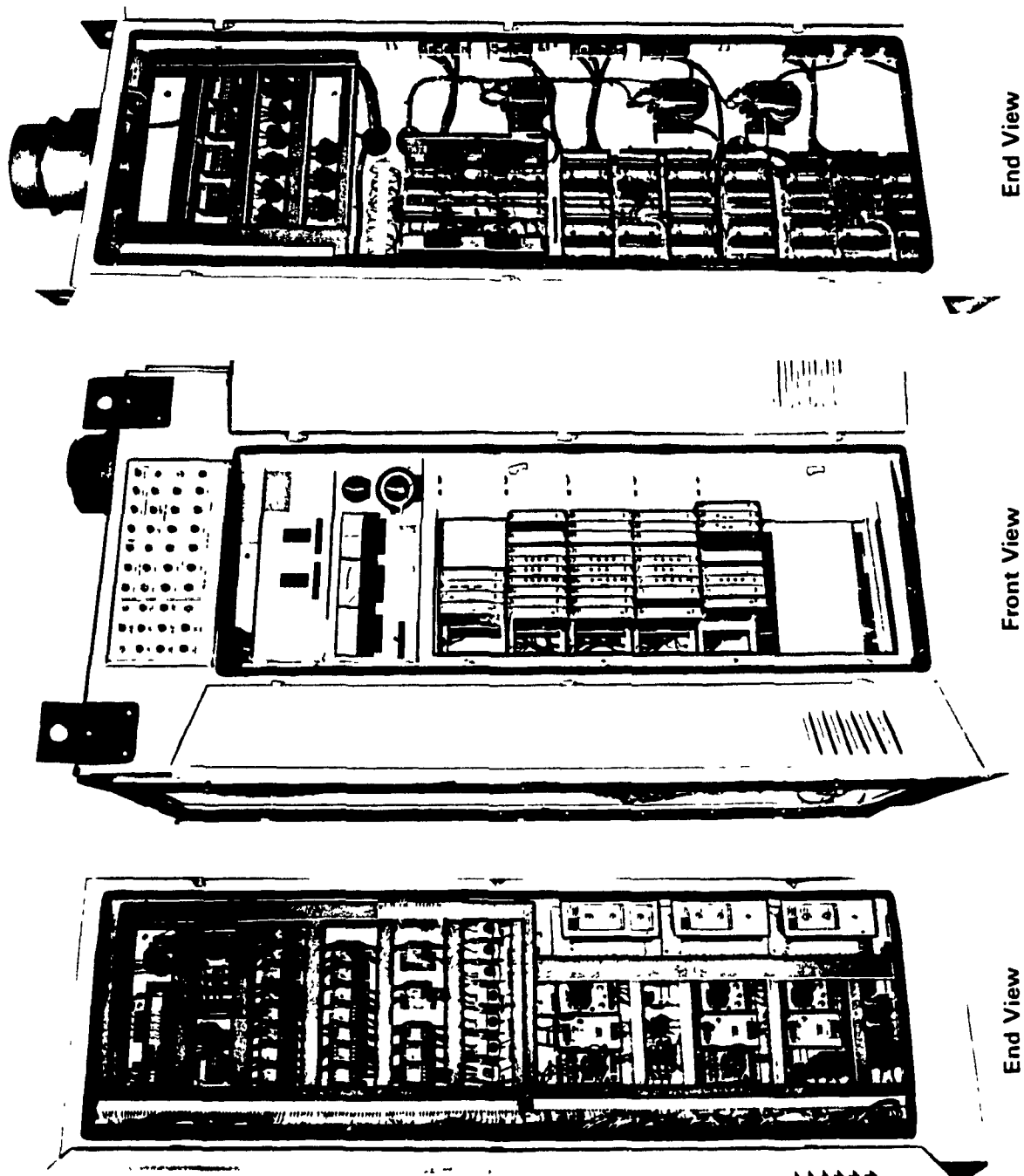


Figure 3.6.2 - Shovel control cabinet.

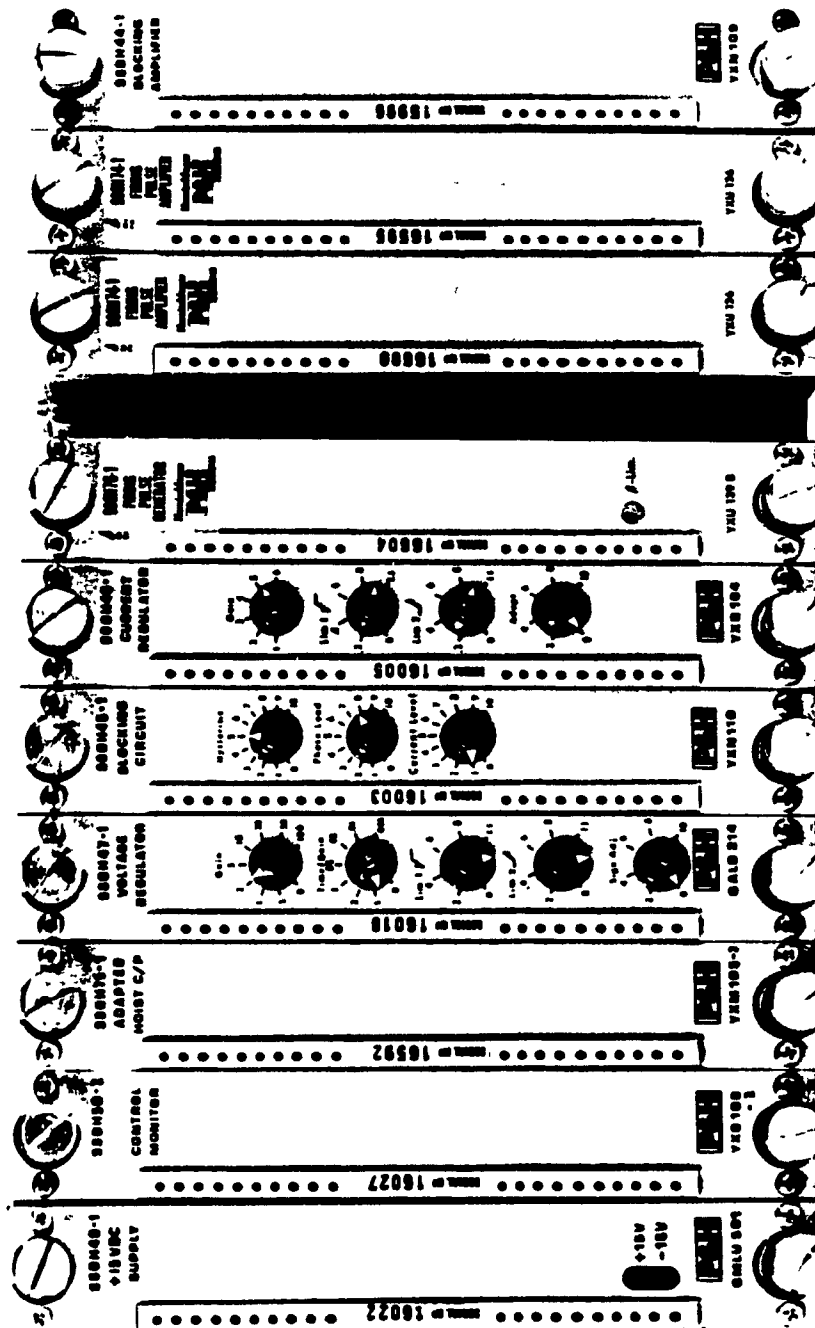
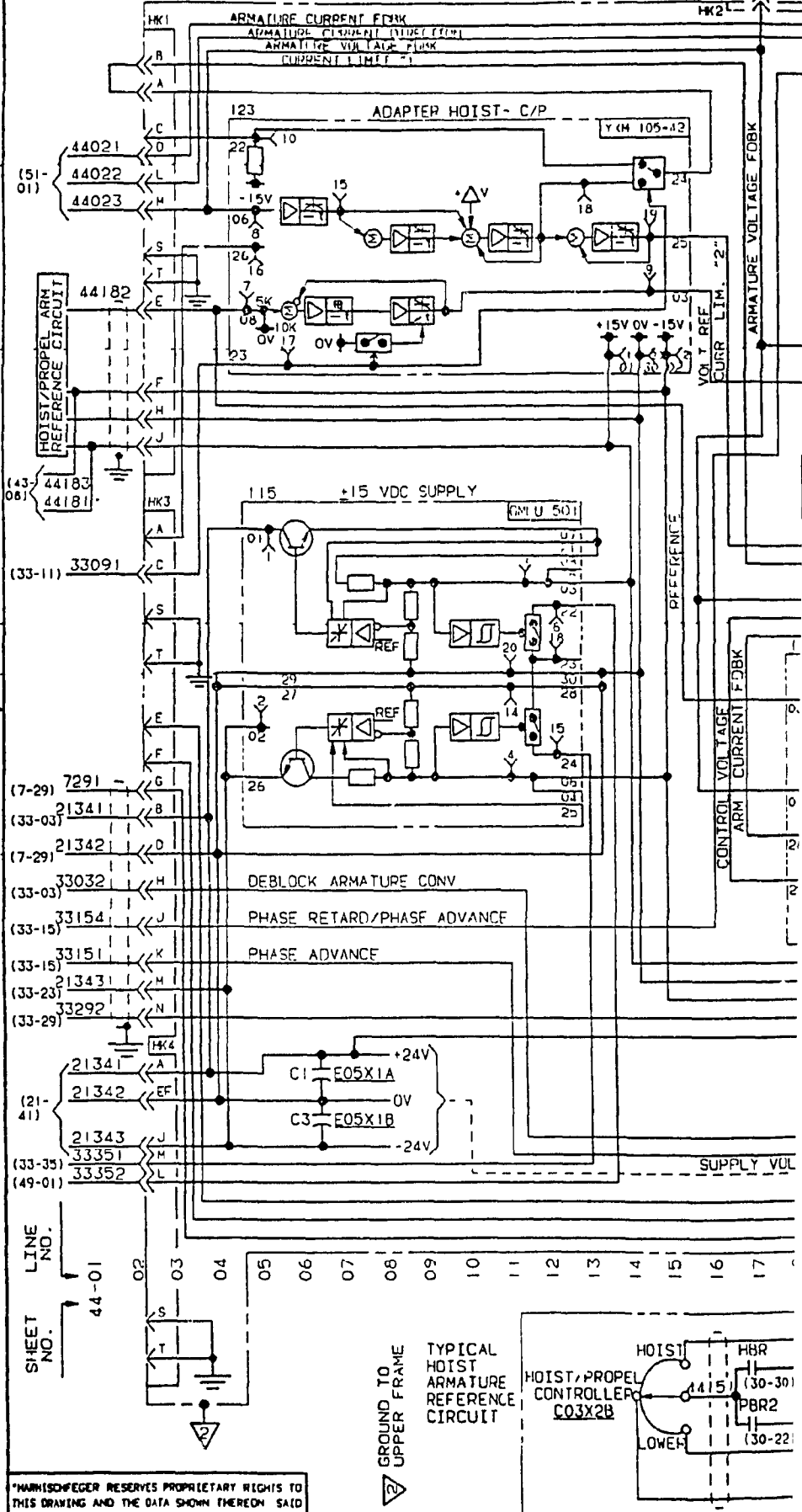


Figure 3.6.3 - Hoist motor control frame.

1

2

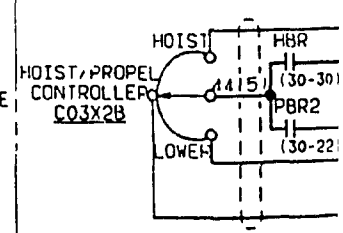
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GROUND TO UPPER FRAME

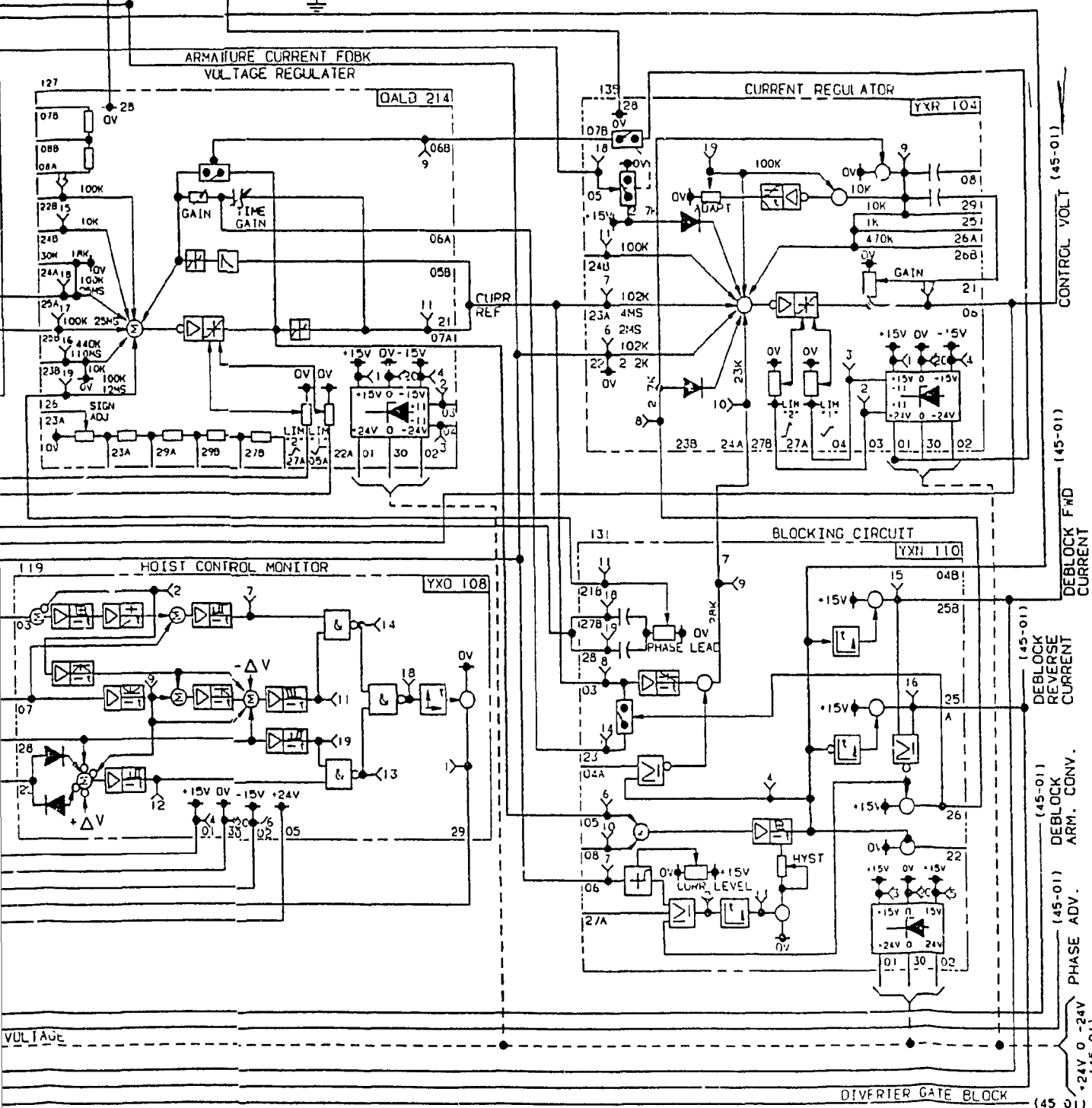
TYPICAL HOIST ARMATURE REFERENCE CIRCUIT



17-45) 21342 7351 (7-45) 21342 (7-45)

E05000

HOIST ARMATURE CONTROL FRAME



VOLTAGE

DIVERTER GATE BLOCK

FORDING RIVER

S.N KJ0028		PART ACTIVITY	
DATE 04-25-81	RELEASE DATE 8-7-85	REPETITIVE	NON REPETITIVE
BASIC LIST NO		Harnischfeger	
SIMILAR TO		P&H	
MILWAUKEE WISCONSIN			
TITLE SCHEMATIC DIAGRAM			
HOIST ARM CONTROL FRAME			
2800 ELECTRO II			
101E16830			

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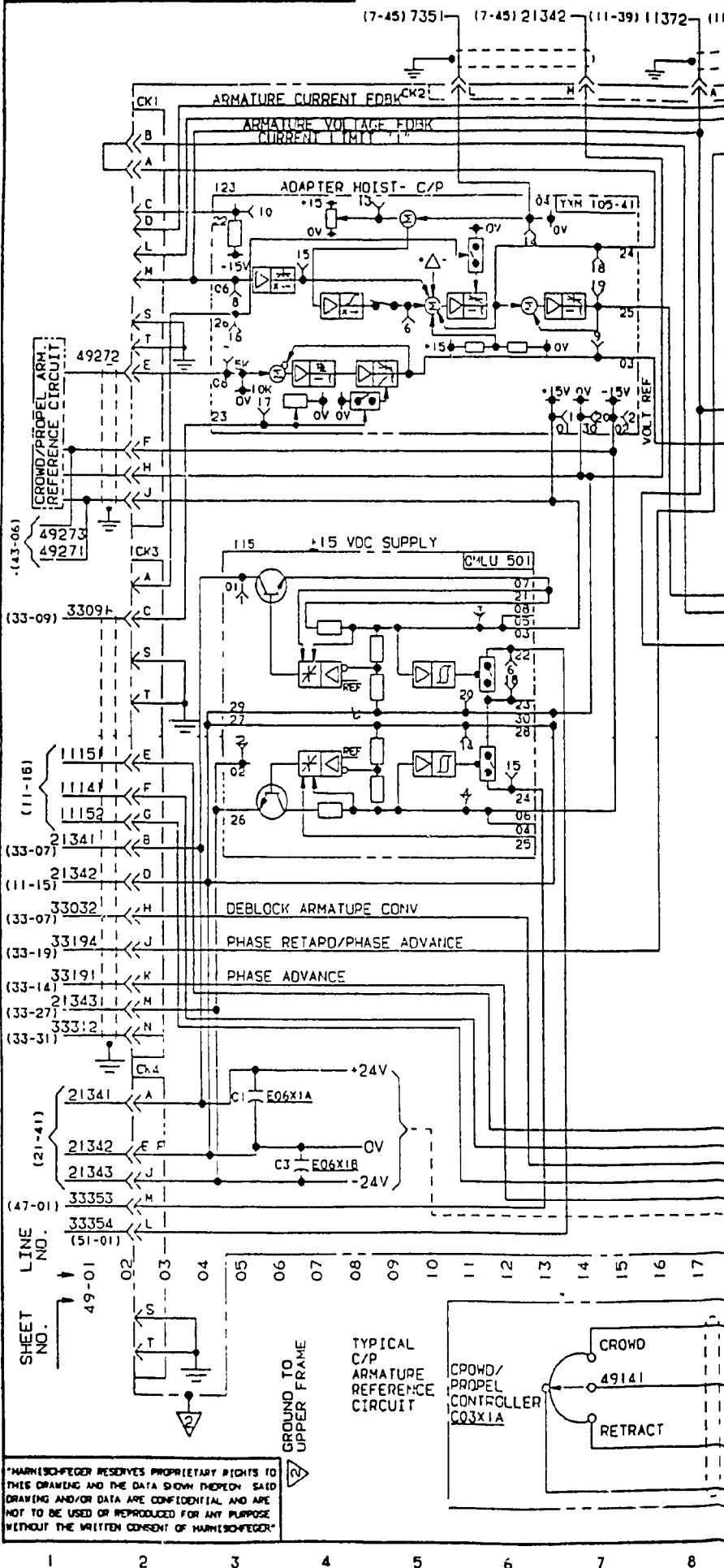
Figure 3.6.4 - Circuit diagram for hoist armature control frame.

Figure 3.6.5 - Circuit diagram for crowd/propel armature control frame.

1

1

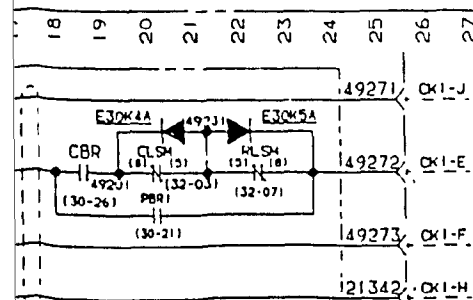
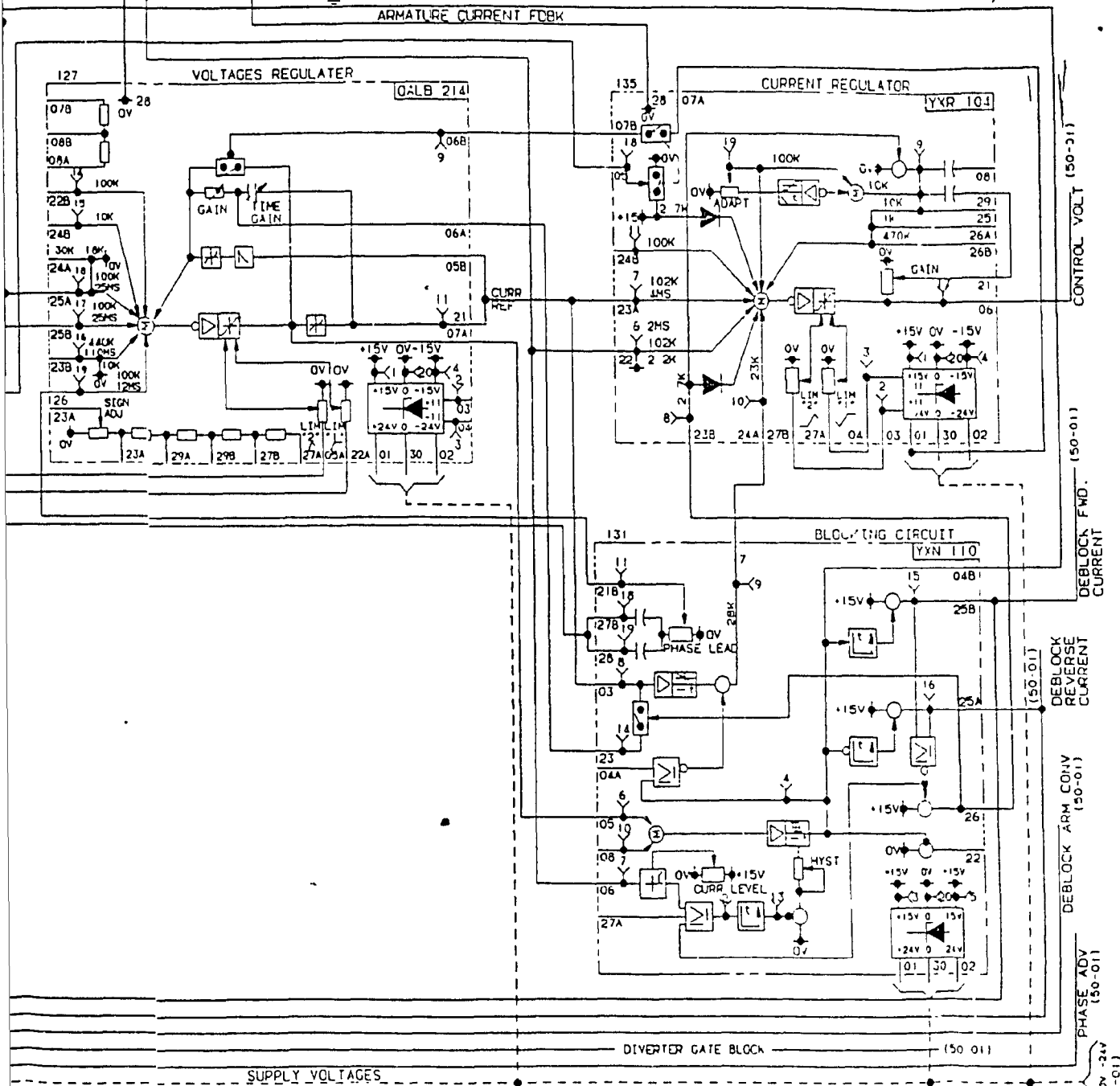
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E06000

CROWD/PROPEL ARMATURE CONTROL FRAME



CAD/CAM DRAWING - DO NOT MANUALLY CHANGE

FORDING RIVER

S.N KJ0028

C.C.

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POSITIVE **NON-DEPT**

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MILWAUKEE WISCONSIN

ATIC DIAGRAM
PHATYPE CONTROL FR

ELECTRO II

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CONTINUED

The equivalent signals are found for the crowd motor on the diagram for crowd/propel armature control frame (Figure 3.6.5). Crowd armature voltage is accessible through test point 18 on the voltage regulator (QALB 214), and crowd armature current is available through test point 6 on the current regulator circuit board (YXR 104).

The signals available from the above control circuits are fully processed (clean and free of noise) as they serve as primary feedback signals for the shovels control system. The availability of these reliable feedback signals alleviated the need to monitor armature voltages and currents directly from the crowd and hoist motors. This would have required extensive additional equipment to properly process and transmit these raw signals to the polycorder. The armature feedback signals from the control logic circuits are all $\pm 10V$. The polycorder however is only capable of handling analog inputs in the range of $\pm 5V$ (or $50mV$). It was necessary therefore to have a voltage divider constructed (by mine electrical maintenance) to step down the incoming 10 volt signals from the control cabinet to a voltage manageable by the Polycorder.

The signals from the control circuits are treated (by the voltage divider) and collected by the Polycorder in the following manner - referring to the block diagram in Figure 3.6.6. A serial cable, one end of which has five of its 25 wires exposed and fitted with copper leads, is connected to the appropriate test points on the armature control frames for hoist and crowd/propel. The fifth lead is connected to test point 20 on the hoist armature voltage regulator (QALB 214), which serves as a common to ground the four monitored signals collected by the Polycorder. The remaining end of the serial cable is connected to the analog input voltage divide, by way of a 25 pin serial connector (female to male). The voltage divider has been designed to accommodate up to eight 10V analog inputs and step them down to 2.5V (8 separate four way voltage dividers) for reading and collection by the Polycorder. The four signals, stepped down to 2.5V each are

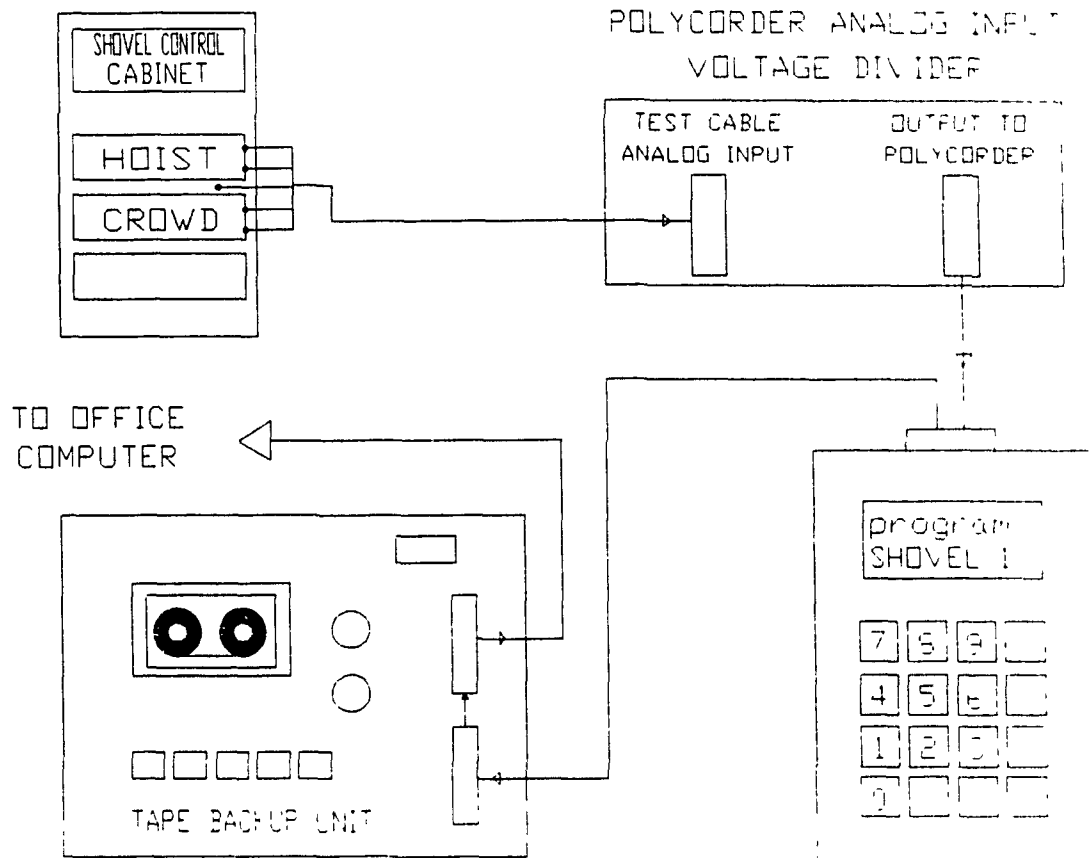


Figure 3.6.6 - Block diagram of Polycorder adaptation.

routed from the analog input voltage divider to the polycorder by a second 25 pin serial cable connected to the Polycorder's serial sensor port.

The second serial cable from the voltage divider to the Polycorder is 50 feet in length. This was required in order to permit operation of the polycorder from the same location as the SPM-8000 (the operators cab) and hence ensure that the two instruments are engaged as close to simultaneously as is possible.

3.6.3 Polycorder Data Handling.

The Polycorder with its maximum memory of 16Kb of RAM will exhaust itself after approximately 10 minutes of monitoring. In

this ten minute period the polycorder will record values for each of the following; hoist armature voltage and current, crowd armature voltage and current and the time the readings were taken to the nearest 100th of a second. An example of the data for each scanned interval is given below:

<u>HV</u>	<u>HC</u>	<u>CV</u>	<u>CC</u>	<u>S ms</u>	<u>HrMn</u>
-2.51	1.12	2.29	0.46	3669	1428

The data going from left represents readings in volts for; hoist voltage (HV), hoist current (HC), crowd voltage (CV) and crowd current (CC). The last two columns refer to the time of day at which the readings were recorded. Referring to the number 3669, this corresponds to 36 seconds and 69 one hundredths of a second. The final column is the hour and minutes for the reading - 14:00 hours and 28 minutes. The complete time for the reading is therefore, 2:00 PM 28 minutes 36 seconds and 69 one hundreds of a second.

The above data represents one line of scanned data. The Polycorder with 16Kb of memory can accommodate approximately 27 pages of such data with 50 lines per page. For reasons to be discussed below, no more than 17 pages of data were employed. This translates to, for a typical monitoring period of 10 minutes, 850 lines of sampled data. From this it is evident that the Polycorder is capable of a maximum sampling interval of only 820 milliseconds. All readings were recorded consistently at this sampling interval, there was no variance by the instrument from this sampling rate. The effect of this slower sampling rate (recalling the SPM-8000 sampling rate of 100 ms) is that traces of the recorded data will be smoother than that recorded by the G.E. monitor.

The maximum number of pages of memory used by the Polycorder was limited to 17 for two reasons. First, 17 pages corresponded to 10 minutes of monitoring which was about the maximum amount of monitored data that could be stored on one of the SPM-8000 data

cartridges. Since the two instruments were often engaged simultaneously, it made sense to terminate them at the same time as well. Secondly, the data collected by the Polycorder had to be down loaded between monitored studies. A tape backup unit was employed to backup monitored data from the Polycorder to clear its memory for successive studies. The tape backup employed was an industrial unit, steel encased and protected from vibration. The unit had a maximum baud rate of 1200 for dumping data to tape. This meant that even when using a 60 minute computer tape, the tape would exhaust one side after backing up only 17 pages and 2 lines of Polycorder stored data. It was decided, given the nature of the environment under which the studies took place, that it would not be practical to stop dumping the data to tape, flip or replace the tape with another, and then resume dumping data to tape. In all it was concluded that 17 pages of data per side of tape backup was adequate and corresponded well with the data collected by the G.E. monitor.

3.7 VIDEO CAMERA.

The third and final piece of instrumentation to form the complete monitoring strategy was a portable video camera. It's purpose was to record all of the events that transpired during a day of monitoring aboard the shovel.

It was desired, that during analysis of the traces of data collected from the SPM-8000 and the Polycorder, that there be an unequivocal means of correlating machine responses (traces) with digging conditions and/or operator practice. There are potential situations that could be falsely interpreted from the recorded traces of machine performance. A primary application of the video camera is therefore to insure that any ambiguities in the recorded traces of motor behaviour could be rectified through review of the actual dig cycles.

Given the range of instruments that had to be attended to during the monitored periods, it was important that the video

camera function under minimum supervision. Two hours was the maximum period the video camera was to run. This meant using one video cassette per day at the standard recording speed mode to insure optimum picture quality. A 120 V power supply was available in the operator's cab so the camera did not require battery changes. It was elected to install the camera inside the operators cab to protect from damage. The view from the cab was extremely favourable and little advantage would be gained through risking the camera by positioning it outside.

The video record established for each day of monitoring has been used to establish shovel cycle times. A program has been developed that will allow direct entry of the following aspects of shovel operation into a computer database - dig, swing to truck, swing to bank, dump time, propel episodes and delays. The program is designed such that the person viewing the video tapes need only hit select keys on a computer keyboard to register the times associated with each event. For example, a viewer observes a typical loading sequence of dig, swing to truck, dump and swing back. To record the times for each action the following sequence of keys would be struck:

1. <D> To initiate the computers internal clock and have the program begin recording the interval of time between striking the <D> key and the next key <T>. This interval of time will represent dig time.
2. <T> By striking <T> the user instructs the program to store the time interval between striking <D> and <T> as dig time. It also instructs the computer to begin recording the next interval of time between striking <T> and <U> as the swing time to the truck.
3. <U> Striking <U> instructs the program to store the swing to truck time. It then initiates timing the interval between striking <U> and as the dump time.
4. Striking instructs the program to store the dump time, and begin recording the final cycle - swing back to the bank. This corresponds to the interval between striking and <D> to indicate the start of a new cycle.

The cycle times determined from the video tapes provide a means of relating these (essentially) manually established cycle times with those cycle times printed out by the SPM-8000.

Recall that the SPM-8000 when engaged to monitor sensor data, will not record any other data such as cycle times. It was important therefore to ensure that most periods of sensor data monitoring were during the two hour periods of video tape recording. In this fashion the voids created in the record of cycle times could be filled with times derived from the video tapes.

The video tape records will prove especially valuable in determining cycle times for those studies conducted in the absence of the G.E. monitor. A portion of the research was conducted without the benefit of the G.E. monitor.

3.8 SUMMARY.

A methodology was developed employing three separate instruments for data collection, to form a complete shovel performance monitoring strategy. Comprehensive records of dipper hoist and crowd power demands, as well as the capacity to track dipper trajectory were acquired by continuous monitoring systems as described above. Use of the G.E. shovel production monitor in its conventional capacity has provided data on dipper load weights, with which to compile shovel production statistics.

A continuous coordinated visual record of digging and handling events were also made possible thorough video tape records of shovel activity. These permit a more informed interpretation of the recorded dig cycles, and also served as a permanent detailed visual record for use in cycle time studies.

4.0 FORDING COAL LIMITED.

4.1 INTRODUCTION.

This chapter will provide a brief description of the mine property and mining methods followed at the Fording River Operations of Fording Coal Limited. This is the location of at which all of the studies discussed in this thesis were conducted.

The Fording River Operation is located in the southeastern corner British Columbia close to the Alberta border (Figure 4.1.1). The principal products of the large surface coal operation are a high quality metallurgical coal, used by the steel industry in the manufacture of coke, and a thermal coal for use primarily by power utilities. Production began in 1972 and since then the mine has been removing 45-50 million tons of waste per year to achieve an annual production of over 5.5 million tons of cleaned coal. The mines current reserves are placed at 330 million tons of cleaned coal, in fifteen mineable seams ranging in thickness from one to several meters (Gold et al, 1987).

4.2 MINE GEOLOGY AND STRUCTURE.

The coal seams on the property fall within the Mist Mountain Formation of the Elk River Valley's Kootenay Group (Williamson, 1984). Of the fifteen mineable seams in the deposit, 10 have a thickness of more than six meters, the remaining seams vary in thickness from one to six meters. The mine does not attempt to recover seams less than one meter thick.

The active mining area covers approximately 10,000 acres and has been influenced by two synclines lying to the east and west of the Fording River Valley, in which the mine plant and

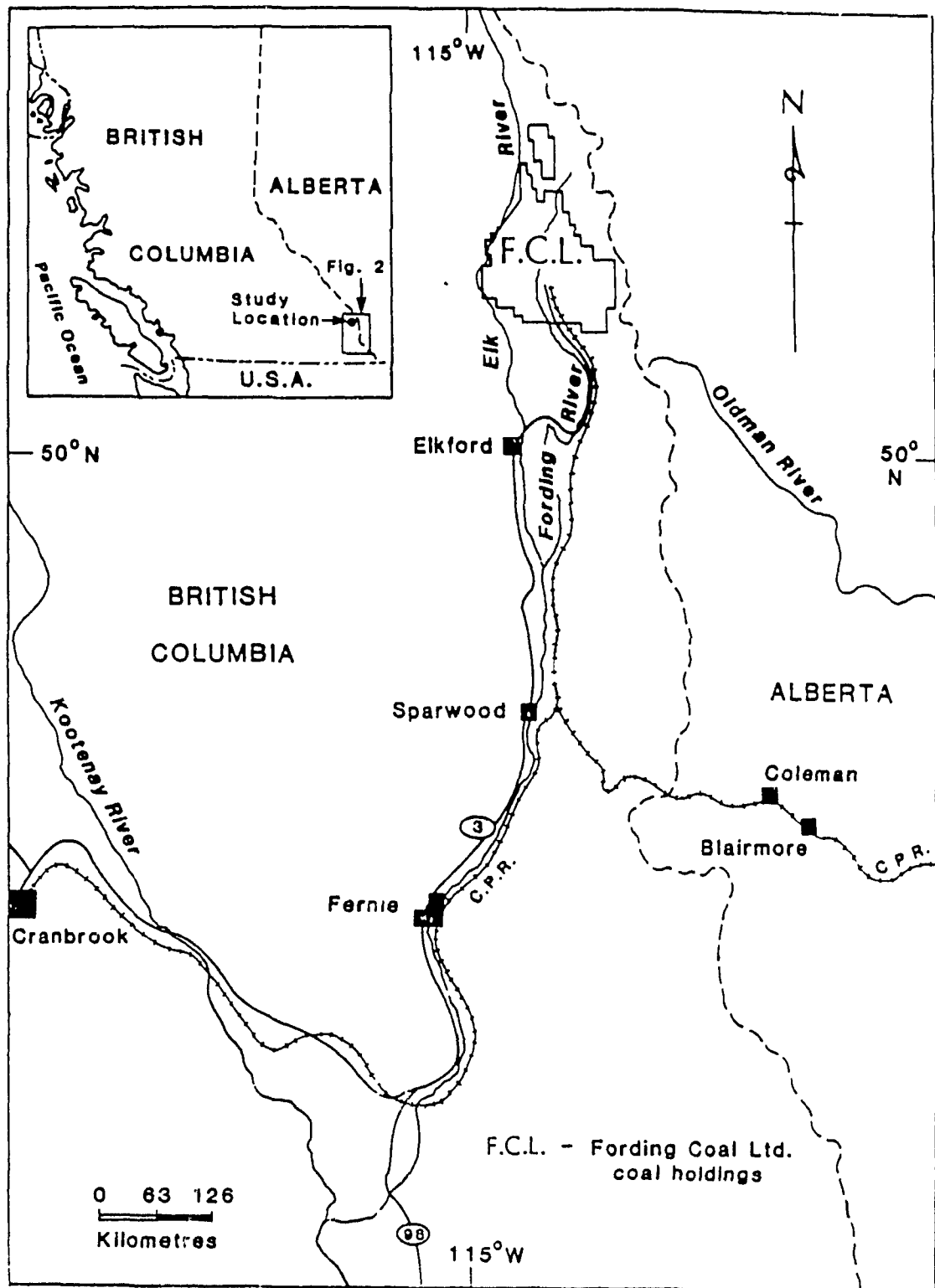


Figure 4.1.1 - Location map of Fording Coal's Fording River Operations (Donald, 1934).

facilities are located (Figure 4.2.2). The coal seams are separated by alternating sequences of sandstone (average UCS strength of 136 MPa), siltstone (UCS=131 MPa) and mudstone (UCS=56 MPa), coal has a low strength value of 21 MPa (UCS). These strength values have been averaged from samples tested from the Eagle Mountain side of the property. Strength determinations on core samples taken from the mine bench selected for the integrated study will be reported on in section 6.4.

There are two predominate joint sets (S_1 and S_2) which are essentially orthogonal to bedding. Joint spacing ranges from 10 cm to several meters in massive units. The seams dip at an average of 17 degrees to the east on the Greenhills side of the property and at 17 Degrees to the west on the Eagle Mountain side.

4.3 MINING METHODS and EQUIPMENT.

Production is achieved by both truck/shovel and dragline operations. The Greenhills side of the property is mined by a Marion 8400 dragline equipped with a 46 cubic meter bucket and is credited with 20% of total annual production. The Eagle Mountain side of the property is mined exclusively by truck/shovel operation which accounts for 80% of mine waste production. A fleet of seven P&H electric mining shovels, four equipped with 23 cubic meter buckets, three with 11.5 cubic meter buckets, mine 15 meter high benches by digging along hanging wall waste to expose coal seams. The shovels are serviced by a combined fleet of 60 Wabco Haulpak trucks. Of the 60 trucks, 46 are 170 ton capacity and are typically used to haul waste, the remainder are smaller 120 ton trucks used for coal haulage.

4.3.1 Drilling and Blasting.

Production blast holes are drilled by a fleet of three B.E. (Bucyrus-Erie) 45R's and Four 60R's, large diameter, electric, rotary blasthole drills. The smaller 45R's are used to drill 10

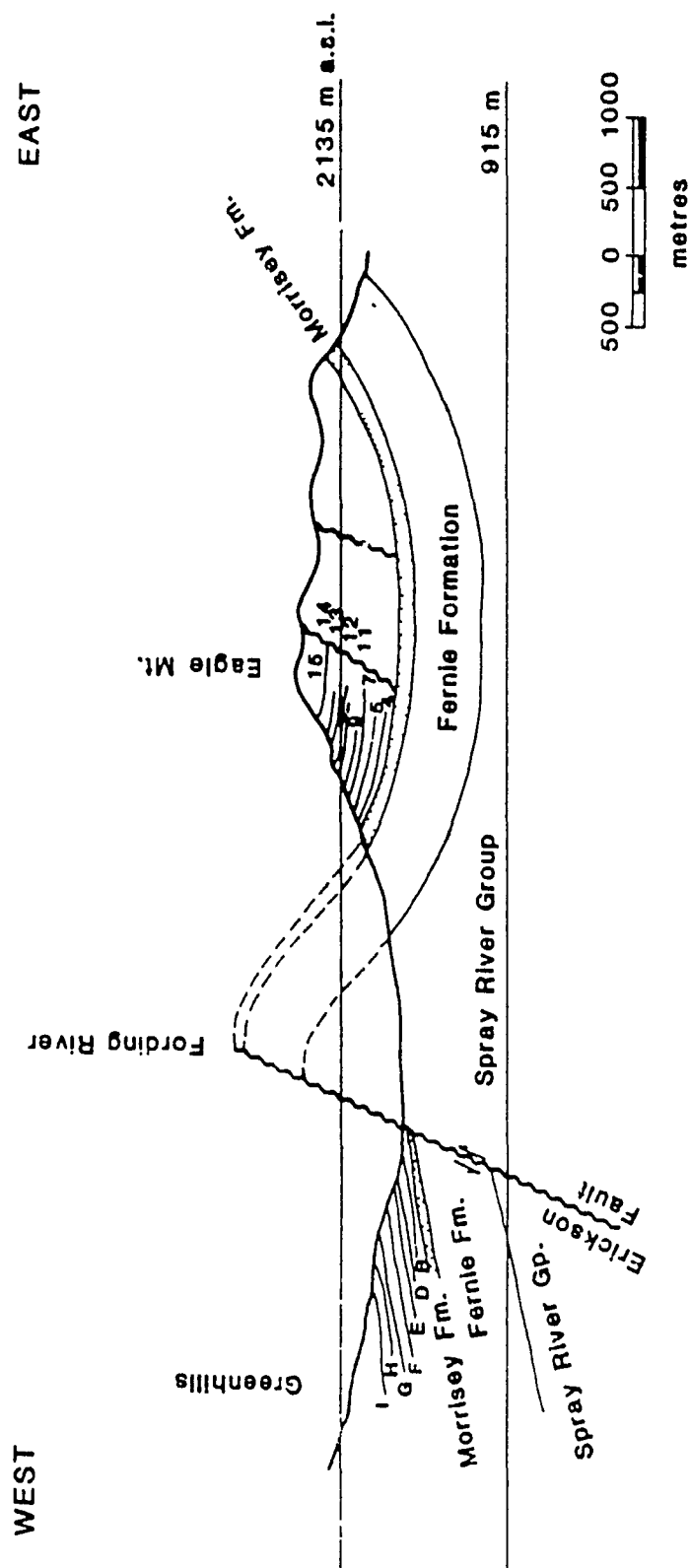


Figure 4.2.2 - Cross section of general mine geology at Fording River Operations.

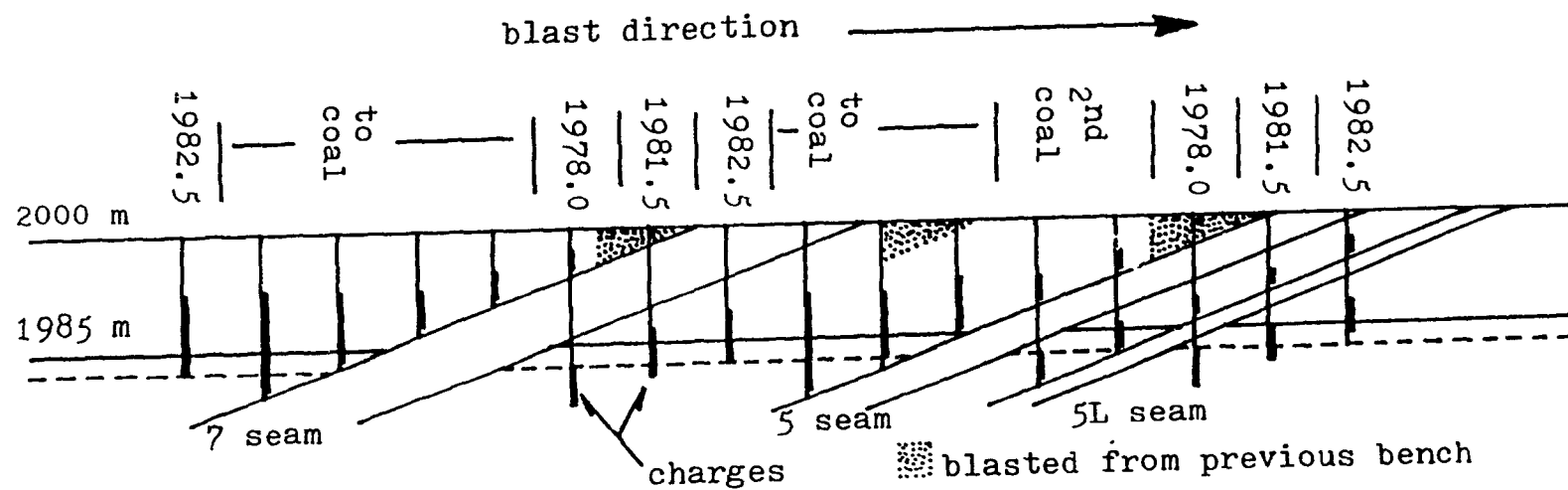
5/8" (270 mm) diameter holes in square patterns of dimension 8.5-9 m (burden = spacing). The 60R's drill 12 1/4" (310 mm) diameter holes in standard square patterns of dimension 10-10.5 m (burden = spacing).

In a continual effort to improve blasting practices, mine engineering is currently evaluating the benefits of blasting from an equilateral triangular drilling pattern. As will be explained in chapter 6, this pattern should return improved results as it allows for optimal distribution of explosive energy in two dimensions. At present only the larger 60R's are being used to drill equilateral patterns, with side dimensions; of spacing = 10.7 m and burden = 9.3 m.

Blasts at Fording are designed for benches that are either, straight waste, or those that contain one or more mineable coal seams (seams > 1m thick). In the case of straight waste, holes are drilled to uniform subgrades of between 2 and 2.5 m, with standard collars of 5.7 and 7.3 meters respectively for the 45R and 60R drills.

The standard practice for blasting benches that contain one or more mineable coal seams is shown in Figure 4.3.1 the most noteworthy elements of which are: Blasting above and below coal, standoff distances, breaking the hanging and footwall wedges of the preceding bench from the one above and, the blast direction.

Holes are drilled to the hangingwall of seams for up to 6 m below the 1985 bench elevation, and for up to 7 m below the footwalls at the same elevation. This approach permits blasting the footwall waste wedges of the next bench from the one above. If these wedges were blasted from the same bench elevation (ie. exposed at surface), short closely spaced holes would be required to minimize throw and dilution.



DESIGN PARAMETERS

Spacing x Burden
Subgrade
Collar

60 R (310 mm Ø)

10m x 10m
2.5m
7.3m

45 R (270 mm Ø)

8.5m x 8.5m
2.0m
6.7m

Figure 4.3.1 - Basic instructions for blast pattern loading.

The standoff distance for charges placed above coal is always 0.6 m (2 feet). Standoffs for below coal loading are variable and dependant on the position of the seam within the bench, see Figure 4.3.2. Where the seam is deep within the bench, the overlying mass provides sufficient restraint against differential heaving of coal and waste units, allowing charges to be placed within 2 m of the seam. However, as the seam nears surface, this confinement is diminished and charges must be dropped in order to control dilution.

Referring again to Figure 4.3.1, note that the blasts are fired for throw in an up dip direction. This implies that the blast is initiated from the right of the diagram, with the throw from successive rounds in an up dip direction towards the diagram's right. This firing sequence is made possible by the low shear strength of the coal waste rock interface which allows the rock to relieve itself up dip, into the effective free faces provided by the broken waste wedges.

The mining sequence following a blast is presented in Figure 4.3.2. As previously stated, mining is by digging along hangingwall waste to expose coal seams. Initially a shovel equipped with a 23 cubic meter bucket (a P&H 2800 or 2800XP) digs waste to the toe of the seam and loads it into 170 ton haul trucks. Once at the toe, in order to control dilution, a dozer is brought in to push the blasted waste wedge down along the seams upper surface to the shovel which remains at the toe. This operation is refereed to as "hangingwall dozing". The dozer is equipped with a ripper to break those thinner partings, less than 2.4 m, that were not loaded. With the seam full exposed, a smaller shovel (P&H 2100) equipped with an 11.5 cubic meter bucket is employed to load the coal into smailer 120 ton trucks. The entire sequence is repeated for the remaining seams in the bench.

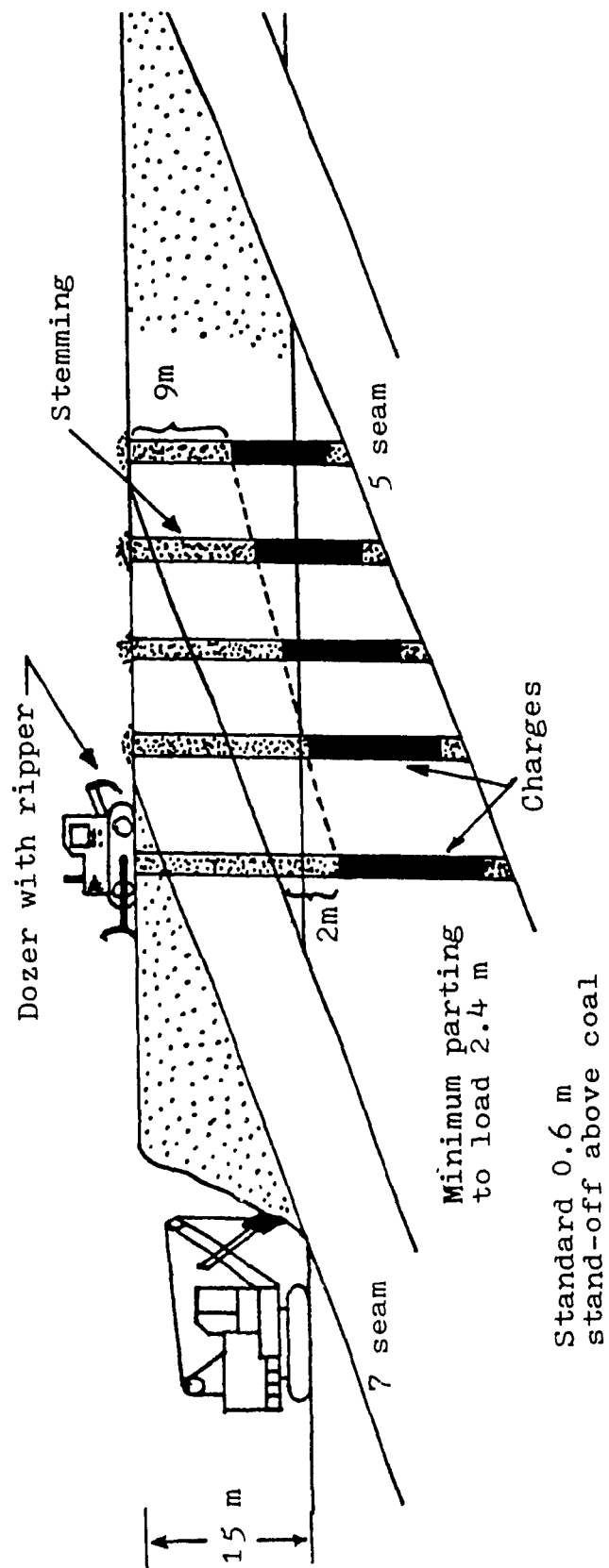


Figure 4.3.2 - Mining sequence and stand off distances for loading above and below coal.

It should be stated that the blasts investigated in the second field period, the integrated study, were only straight waste. The blasts were excavated in their entirety by a single P&H 2800XP, equipped with a 23 cubic meter bucket. The manual time studies conducted in the first period of the project however, (Chapter 5) involved studying the actions of P&H 2800 and 2800XP's (both equipped with 23 cubic meter buckets), as they handled hangingwall waste, footwall waste and hangingwall dozed waste.

4.3.2 Blast Tie-Ins and Sequencing.

Approximately 75% of the blasts at Fording employ both surface and down hole delays (DTH). When down hole delay-detonators of 340 ms are used, inter row surface delays of 50 ms are used along the firing lines and free faces, with 75 ms surface delays along the safety lines and confined faces. Light 25 grain/foot primacord surface lines are used in down hole delayed blasts.

In surface delayed blasts, 50 grain/foot primacord firing lines are connected to 60 grain/foot "Scuf-flex" down hole lines. Inter row delays are 35 ms along the firing lines and free faces and 50 ms along the safety lines and confined faces.

4.3.3 Explosives.

As of January 1987, all explosives used at the mine have been emulsion based heavy ANFO's. Table 4.1 below, lists the emulsion based explosives used by the mine. The dry hole product is a Heavy ANFO, manufactured by C.I.L. under the trade name "SuperAN". It is to be noted that the bulk strength of the product increases as the percentage of explosive emulsion matrix in the blend is increased. The wet hole product is a 100% emulsion blend, also a C.I.L. product called "MAGNAFRAC". The bulk strength of this product is accomplished through the addition of aluminum as indicated.

Table 4.1 - Explosives.

DRY HOLES

SuperAN	2495	20% emulsion	RBS* = 109
SuperAN	2505	30% emulsion	RBS = 120
SuperAN	2510	35% emulsion	RBS = 125
SuperAN	2515	40% emulsion	RBS = 130

WET HOLES

MAGNAFRAC	9000		RBS = 118
MAGNAFRAC	9015	3% Al	RBS = 146
MAGNAFRAC	9020	10% Al	RBS = 164

* Strength relative to ANFO at 0.84 gm/cc RBS = 100

5.0 INTEGRATED BENCH STUDY - GEOLOGY AND BLAST DESIGNS.

5.1 INTRODUCTION.

This chapter will address the experimental design followed during the course of the investigation. The research has been conducted in two phases: A preliminary phase investigation, which helped to define the requirements of a more comprehensive study during the second, final phase of the research: "The Integrated Bench Study".

5.1.1 Preliminary investigations: manual time studies - summer 1987.

This section is presented in order to briefly describe the nature of work done in advance of the more comprehensive shovel instrumentation studies, reported on in the remainder of this thesis. The results of this preliminary research are significant in that they motivated and defined the required approach undertaken during the more sophisticated/comprehensive investigation of the second field period - the integrated bench study. A detailed description of the preliminary research; methodology, data analysis, results etc., is offered in **Appendix B.**

Field work on the project began in the summer of 1987, at which time it was elected to undertake a manual time study of shovels operating in a range of mine operating environments. The intent was to derive a diggability index for each operating environment in which the performance of the shovel was studied (Hendricks et al., 1988). Even if it were not possible to establish a set of digging indices, the recorded data would at least provide an indication of the extent to which an indirect measure of machine performance (dig cycle time) was related to environment (as defined by geology and blast design) and muckpile

digging conditions.

It is now known, based on the results of the shovel instrumentation studies, that dig cycle times are operator dependant and show minimal variation with respect to changes in muckpile diggability. In this regard, the manual time studies are of limited relevance to the current research, other than representing a logical and legitimate first approach at defining the relations between shovel performance and diggability.

Statistical analysis of the time study data proved confounding. It was not possible to establish any sound and consistent relations between dig cycles times and environment characteristics. Analysis of the data was plagued by a multitude of variables, each of which could not be accurately parametrized and accounted for (i.e. several shovels each with a unique set of operators, working in multiple environments)

Key deficiencies of the preliminary research were identified as:

1). There existed no direct means of relating mean dig cycle times with actual measures of digging effort. A key assumption of the preliminary research was that shovel performance, affected by varying degrees of digging difficulty, will be reflected through changes in mean dig cycle times. Longer dig cycle being indicative of more difficult digging conditions. This has since been proven false (on the basis of conclusions drawn from the integrated bench study employing shovel instrumentation), mean dig cycle times show little variation between easy and hard digging conditions.

2). There was no way of assessing variations in the operating practices amongst operators. It was necessary to assume that all operators perform equally in equivalent digging conditions. This too, has since been proven false.

3). Dig cycle times alone revealed little about actual shovel productivity. Dig cycle times may have varied as a consequence of an operator's attempts to increase dipper fill factors. However, it was possible that a shovel working a particular environment may have a longer mean dig cycle time than in another environment, where it may have been more productive.

The results of the preliminary study defined the need for a more comprehensive investigation involving actual instrumentation and performance monitoring of the shovel. This, in order to confirm that specific aspects of shovel operation (performance parameters), that relate directly to digging effort and are responsive to variations in muckpile diggability, could be identified. It was also concluded that the monitoring trials should be conducted in a controlled environment in order to minimize several uncontrollable variables associated with variation in geology from one site to another. Towards this end, a single mine bench was selected, in which local variation in geology and structure were fully explored. A series of blast were then designed in said bench, in order to generate a series of blast zones in which to compare shovel performance. Fragmentation distributions from each blast zone were determined and thus provide a quantified measure of blast efficiency, against which to correlate shovel performance.

Finally, by refining the scope of the experiment (isolating a single environment) the performance of only one shovel and set of operators need be considered.

The remainder of this chapter will describe in detail, the geology and structure of the mine bench in which the performance of the instrumented shovel was evaluated. Following which the design characteristics of the blasts fired in said bench will be discussed.

5.1.2. Integrated bench study - summer 1988.

The main ambition behind shovel monitoring is to establish a means of relating shovel performance to muckpile characteristics; fragmentation and looseness. Towards establishing these relationships separate blast environments, each characterized by unique blast design criteria, were created within a single mine bench. The effect of varied blast designs within a given environment will theoretically result in differing muckpiles, with characteristics unique to the blast design elements responsible for their creation. Excavator performance may then be related to the resulting muck and possibly to the blast design variant.

It was originally intended to monitor the performance of the shovel as it excavated two blasts in the test bench; EZ#3 drilled in a square pattern and EM#1 which employed an equilateral drill pattern, Figure 5.1.1. However, unforeseen circumstances resulted in a change of production scheduling and much of blast EZ#3 was excavated in advance of the study start date. To compensate, a third blast of design characteristics equal to blast EZ#3, was fired in October of 1988 during which time a second, unscheduled field monitoring period was conducted.

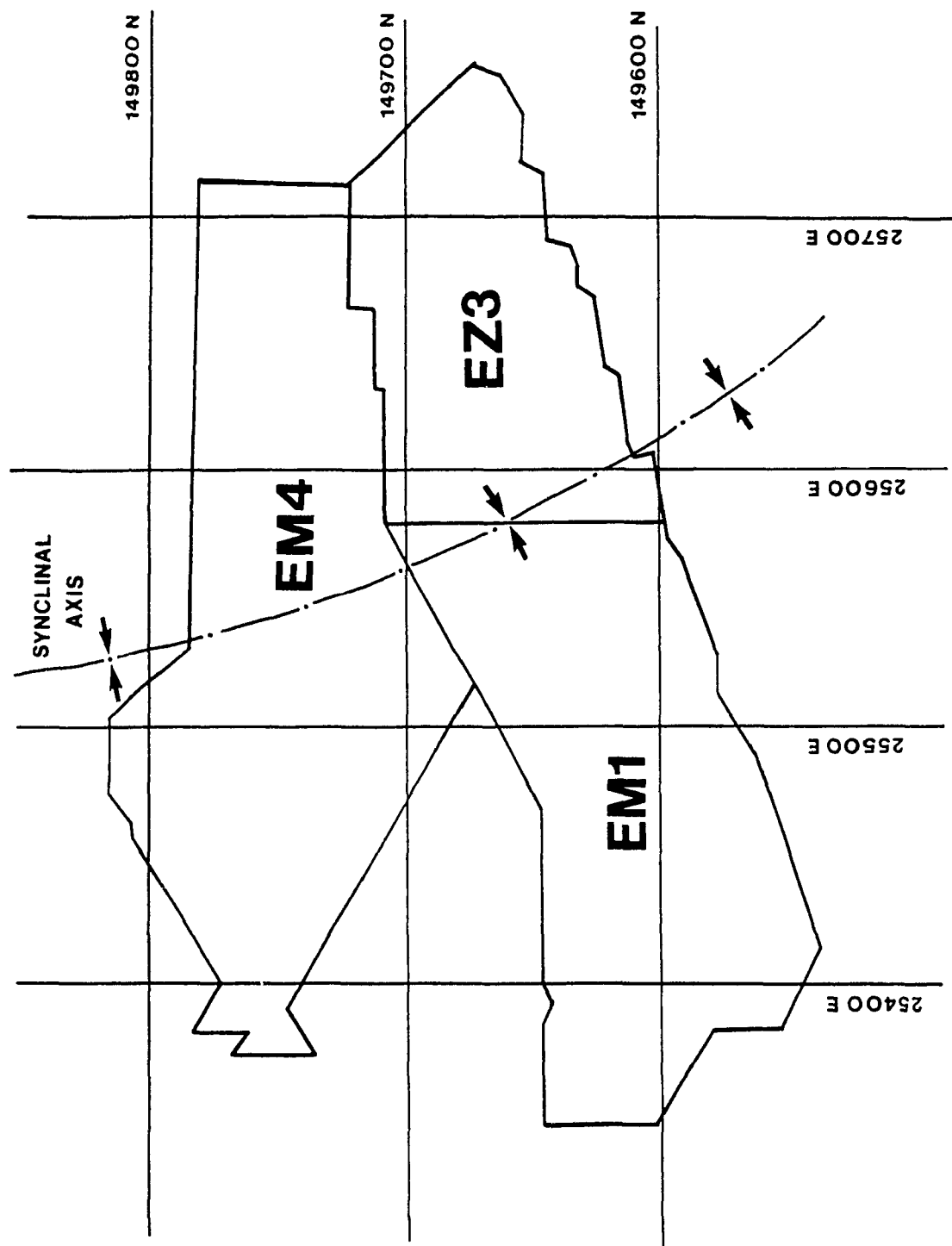


Figure 5.1.1 - Positions of blasts EZ#3, EM#1 and EM#4 in bench 2240.

5.2 GEOLOGY OF THE TEST BENCH AREA.

The mine bench selected for the integrated study was located at the 2240 meter elevation level of Eagle Mountain, the bench will here within be referred to as "bench 2240". The bench, as with all others on the property was 15 meters in height (50 feet), and extended from the 2240 level down to the 2225 level of the mine.

Recalling from a general description of mine geology and regional structure (Section 4.2), the property is characterized by two synclines, one on either side of the Fording River Valley. The area of Eagle Mountain's bench 2240 studied falls directly on the synclinal axis of the Eagle Mountain syncline (Figure 5.1.1). Hence, the geologic setting is characterized by a series of very shallow dipping waste rock units - sandstone, siltstone and mudstones inter-spaced in some sections by a series of coaly stringers.

As described in the introduction to the thesis, performance monitoring of the drill used to drill off the test blasts, provided extensive data on bench geology. Records of drill performance parameters such as rotary torque, hydraulic downpressure, rotary speed and penetration rate, from several blastholes have been interpreted to provide details on the variation in bench stratigraphy. The extent and location of drill monitored holes within the test bench is given in Figure 5.2.1.

The response of the monitored drill parameters have been related to the strength characteristics of the bench rock units (Peck et al, 1988). In this fashion a series of north-south trending stratigraphic cross sections, defining the relative position and extent of hard and soft horizons in the test bench have been drafted. Note that drill monitored data is available

only for the two blasts evaluated during the August 1988 monitoring period - EZ#3 and EM#1. The geology of blast EM#4, excavated during the October 1988 field period, has been established through extensive gamma logging of several blastholes in the pattern, Figure 5.2.2.

Figure 5.2.3 is an example of a typical geologic cross section, which illustrates the positions of hard and soft lithologies, based on drill monitoring, along EZ#3 blast row of holes EZ1370 to EZ1378. For present purposes, sandstones and/or siltstones are taken as hard bands and coal, mudstones and coaly-mudstones are considered soft bands. Similar sections drafted for the remainder of the test bench area, provide the basis for the composite of bench geology depicted in Figure 5.2.4. The remaining geologic sections, drafted for each blast, are presented in Appendix C.

Figure 5.2.4 reveals the presence of a massive sandstone unit in the eastern portion of the test bench area. This gives way towards the west, to a more variable lithology characterized by the presence of several weaker coaly mudstone units.

A similar "composite" cross section has been drafted, based on gamma logs for the bench area within blast EM#4, Figure 5.2.5. The section reveals the massive sandstone to be continuous to the north of blast EZ#3, and dominating the eastern portion of blast EM#4. The western portion of blast EM#4 was also characterized by a more variable, weaker, lithologic sequence.

5.3 STRUCTURAL GEOLOGY OF BENCH 2240.

The area of bench 2240 studied is a portion of a conformable sedimentary sequence situated within the axial plane of the Fording syncline, which plunges gently towards azimuth 345°. The mean attitude of bedding on the property is 17°/060° (dip / dip

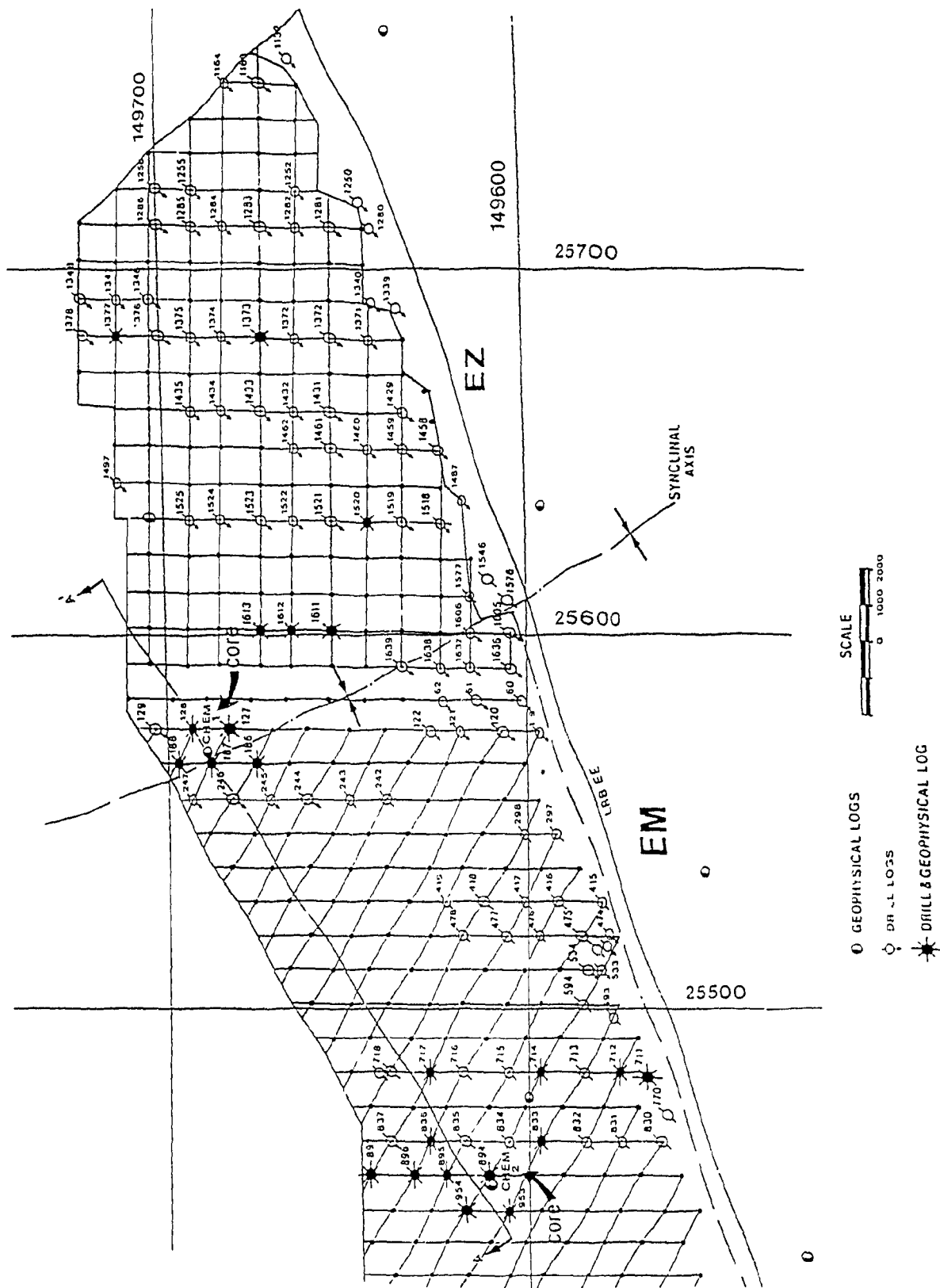


Figure 5.2.1 - Location of drill monitored, gamma logged and cored blast holes.

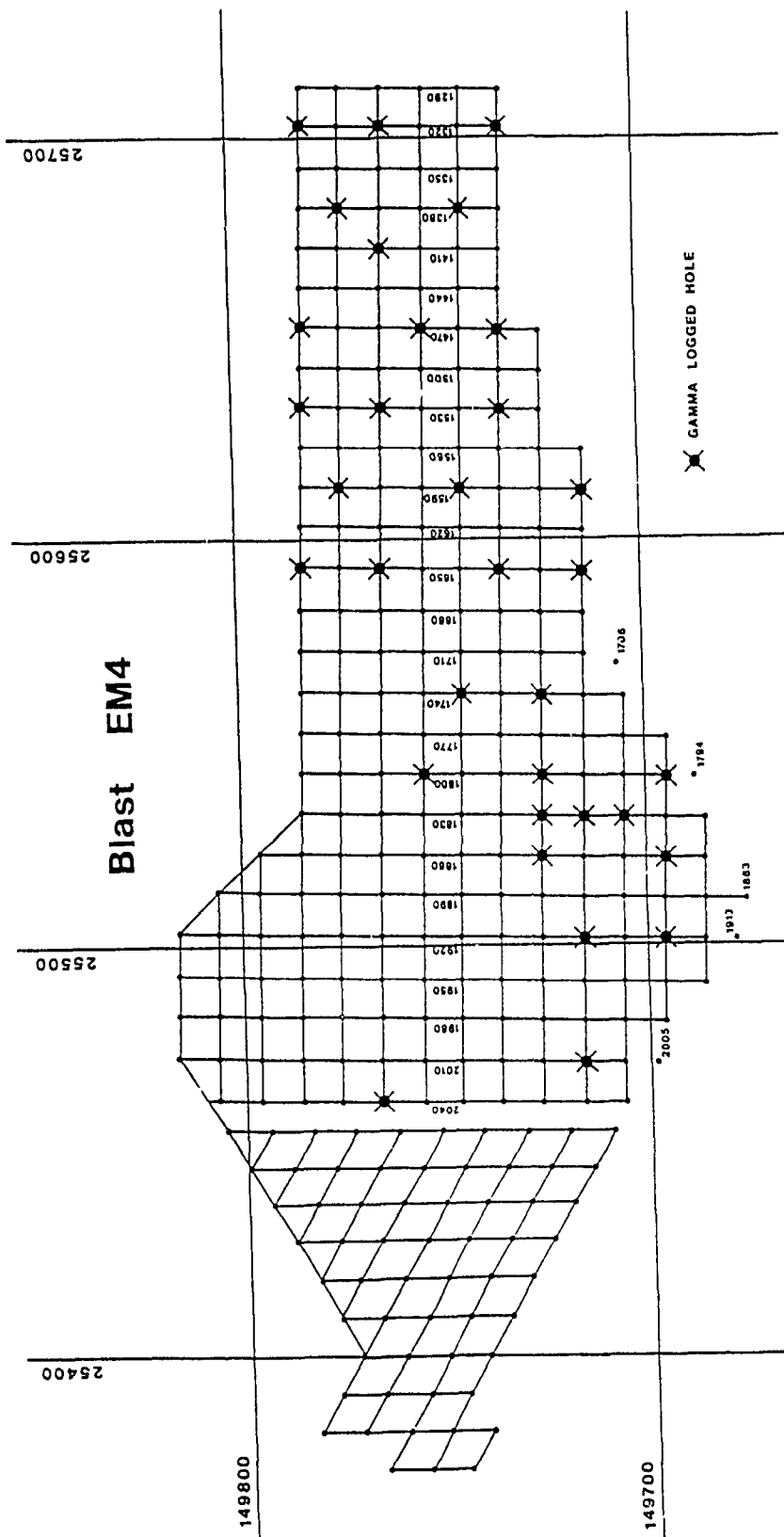


Figure 5.2.2 - Blast EM#4 - showing location of gamma logged blastholes.

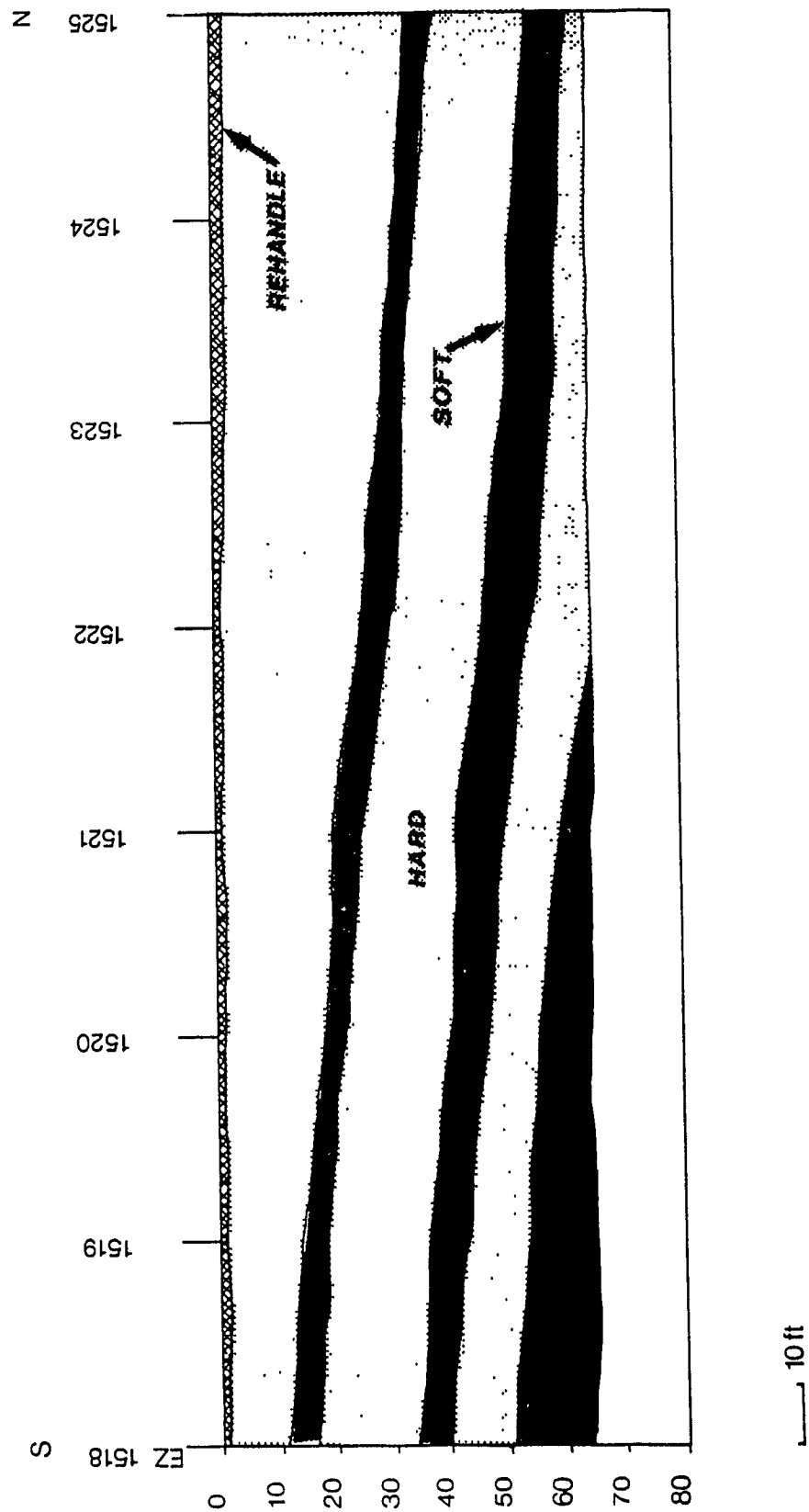


Figure 5.2.3 - Example geologic cross section, identification of hard and soft bands by drill monitoring.

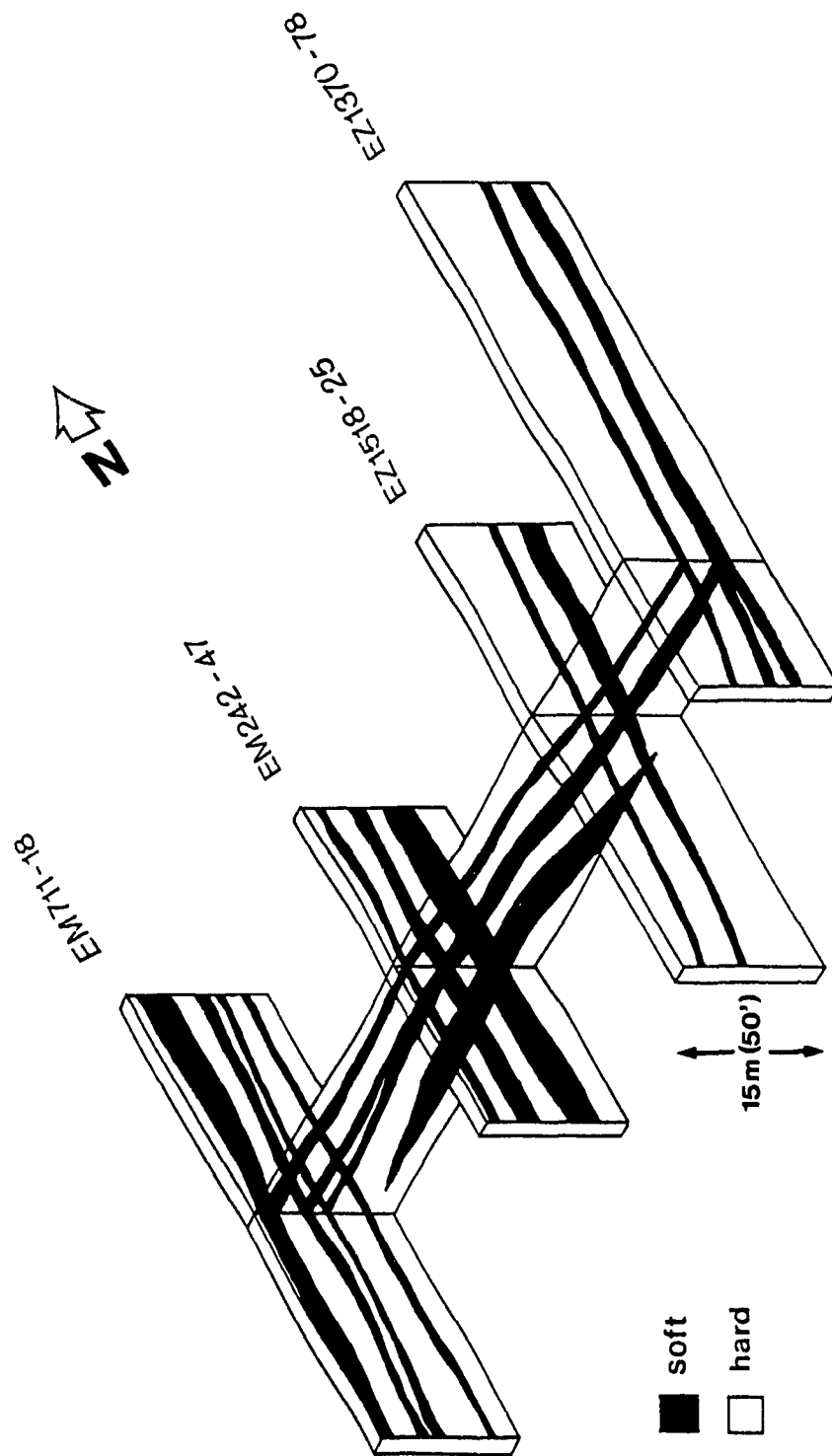


Figure 5.2.4 - Panel diagram of Eagle Mountain 2240 bench geology - showing location of hard and soft bands.

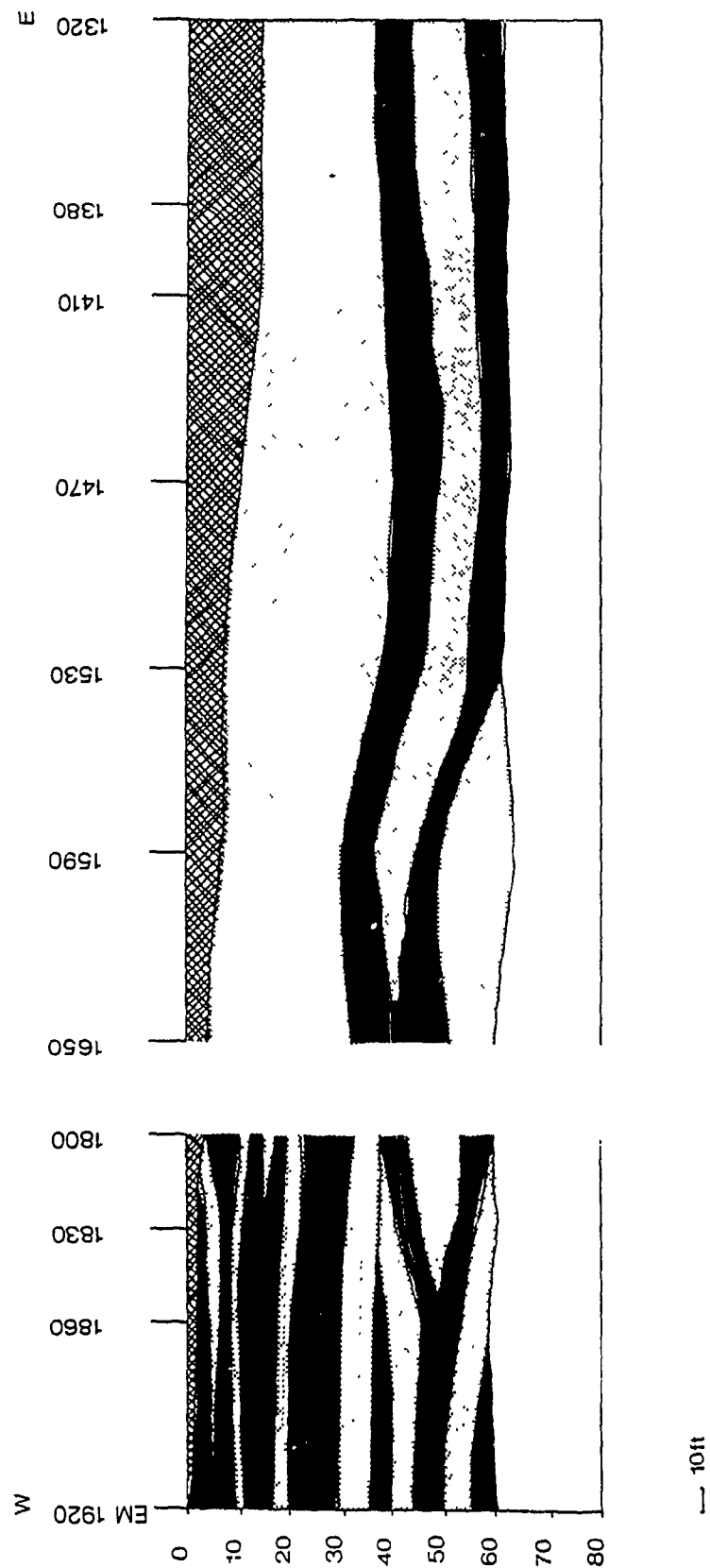


Figure 5.2.5 - Location of hard and soft bands in blast EM#4, defined by gamma logs.

direction). However, for practical purposes bedding in the test bench, within the axial plane of the syncline can be regarded as essentially horizontal and striking approximately north north-west.

Local structure of the test bench is a product of bedding and jointing within the rock mass. Structural mapping of the entire test bench area revealed the presence of two joint sets, S-1 and S-2, with mean orientations of $84^{\circ}/270^{\circ}$ and $81^{\circ}/190^{\circ}$ respectively. These measurements compare favourably with those recorded by Golder in 1983, S-1 = $76^{\circ}/270^{\circ}$ and S-2 = $84^{\circ}/140^{\circ}$.

The average joint spacings for each of the joint sets are dependant on waste rock type. The coarser grained more competent sandstones exhibit larger mean spacings than the finer grained siltstones and mudstone units. The spacings in joint set S-2 are greater than those of the S-1 set. The following measurements were recorded by Golder in 1978 (Golder, 78):

<u>Rock Type</u>	<u>Mean Spacing</u>	
	<u>Set S-1</u>	<u>Set S-2</u>
Sandstone	8 to 12"	1 to 2'
Siltstones & Mudstones	4 to 8"	8 to 12"

The joint spacings recorded by Golder are not entirely representative of those observed in the test bench area, in which joint spacings as great as 2 to 3 meters were observed in the massive sandstone unit.

As noted by Golder (Golder, 78), all joint sets were observed to terminate against bedding discontinuities. Joint surfaces were clean except in weathered zones in which iron staining and/or calcite infilling was observed.

5.4 PHYSICAL PROPERTIES OF BENCH 2240 ROCK UNITS.

The strength characteristics of the bench waste units have been determined at McGill University from laboratory testing of core retrieved from the two cored holes in blast EM#1 - CHEM1 and CHEM2 (Figure 5.2.1). Laboratory tests have determined the unconfined compressive strengths, tangent modulus and point load strengths of the various units in the test bench. Prior work by Golder and associates in 1978, has been used for values of density, basic friction angle, and Poisson ratio for the various waste units. A summary of the above values as determined by Golder and McGill University is presented in table 5.1.

TABLE 5.1
ROCK STRENGTH PARAMETERS.

PROPERTY	ROCK	McGILL (1988)	COLDER (1978)
Unconfined comp. strength. (MPa)	sdst	125.08 (14) (72.09-186.79)	149.34 (8) (99.00-204.00)
	slst	115.00 (11) (61.76-160.10)	117.00 (5) (83.00-156.00)
	mdst	65.30 (3) (20.7-76.80)	58.93 (7) (26.00-99.00)
	coal	-	9.30 (3) (3.45-20.80)
Modulus (MPa)	sdst	18802.09 (14) (11978-25380)	31930.00 (3) (29600-31000)
	slst	15027.81 (11) (9785-21227)	393000.00 (2) (36500-50300)
	mdst	11010.72 (3) (8743-11935)	39300.00 (1) -
	coal	- (tangent mod)	3100.00 (1) (secant mod)
Density	sdst	-	2.59 (3) (2.44-2.67)
	slst	-	2.62 (3) (2.53-2.71)
	mdst	-	2.77 (3) (2.60-2.81)
	coal	-	1.32 (3) (1.19-1.39)
Basic Friction Angle (degrees)	sdst	-	28.6 (7) (26-30)
	slst	-	31.5 (8) (23.5-39)
	mdst	-	29.6 (5) (26-33)
	coal	-	22.9 (5) (16-32)
Point Load Str. (MPa)	sdst	103.35 (2) (96.3-110.4)	140.04 (62.1-230.5)
	slst	113.37 (4) (91.8-135.6)	108.86 (5) (40.0-177.0)
	mdst	31.86 (2) (20.1-43.6)	63.65 (4) (110.0-193.0)
	coal	-	21.00 (1) (11.0-67.0)
Poisson's Ratio	sdst	-	0.08 - 0.10
	slst	-	0.34
	mdst	-	0.13
	coal	-	0.14 - 0.38

5.5 BLAST DESIGNS - EZ#3, EM#1 and EM#4.

The following subsections report upon the design specifics of each of the three blast studied during the course of the research. Note that all holes in the three blasts were drilled by the same drill, a B.E. 60R with a 12 1/4 inch tricone bit. Additionally, all blasts employed period 10 NONEL down hole delay lines and detonators (340 ms delays), only the surface delays have varied from one blast to another.

5.5.1 Blast EZ#3.

The design details of blast EZ#3 are depicted in Figure 5.5.1. The entire blast was drilled in a square pattern with burdens equal to spacing equal to 10 meters. The blast tie-in and firing sequence is depicted in Figure 5.5.2, which shows the blast was fired in a square "V" type initiation sequence, with the ratio of effective spacing to burden equal to 2.0. The front half of the blast used inter row delays of 50 ms, the later rows were fired with an extended delay interval of 75 ms to see if the potential benefits of extended delays could be detected in shovel performance. Theoretically, the longer delays should help to keep the muck loose at the back of the blast, where digging can often be tight as a result of inadequate heave of final burdens. The blast was fired on June 29th 1988. Much of blast EZ#3 was characterized by the presence of the massive sandstone unit depicted in Figure 5.2.4.

Regrettably, the first half of blast EZ#3 was inadvertently excavated in advance of shovel monitoring. It was not possible therefore, to relate shovel performance to anticipated changes in muckpile diggability due to extended inter-row delay intervals.

Drill monitoring of blasts EZ#3 and EM#1 was completed on July 7th, just before the start of the mine's planned one month summer shut period. Excavation of the blast was not scheduled to

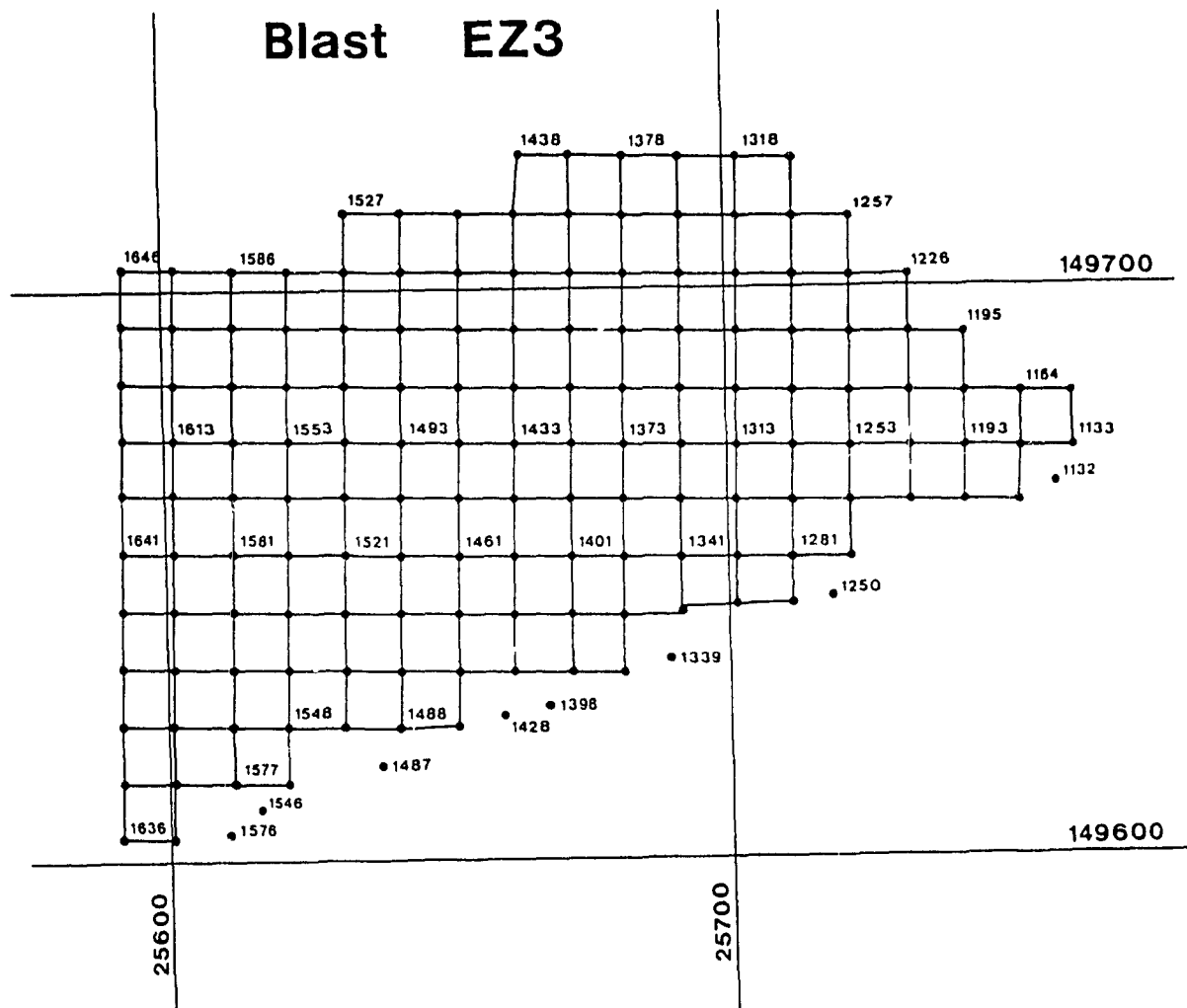
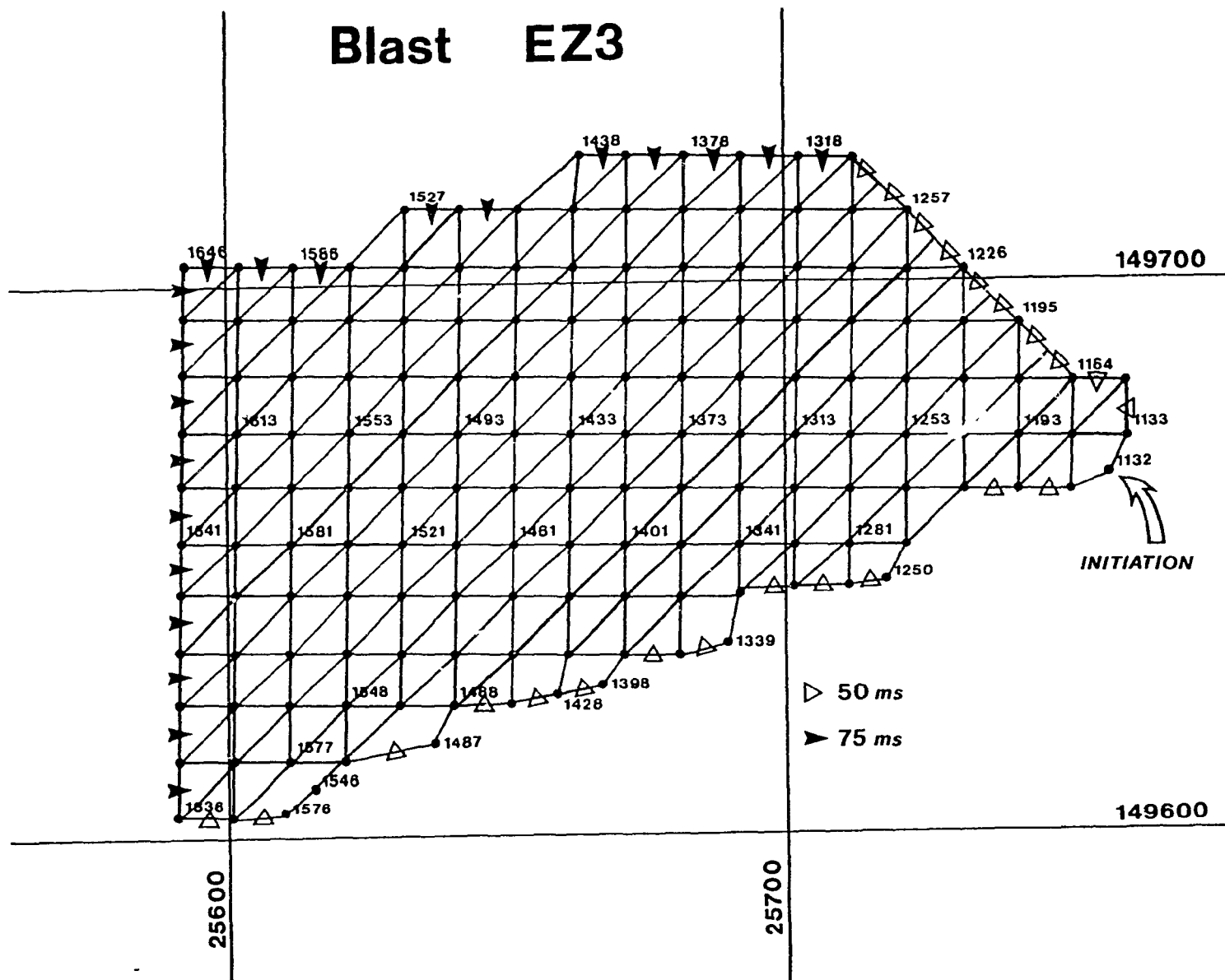


Figure 5.5.1 - Blast EZ#3 design.

Figure 5.5.2 - Blast EZ#3 tie-in and firing sequence.



begin until operations recommenced on August 14th. Production requirements however, necessitated ongoing operation in another mine pit (Taylor Pit) during the shut down. When work in Taylor neared completion before the end of shut down, and shovel operators became available, mine production elected to begin work on Eagle Mountain ahead of schedule. They were unaware of the integrated study and no one in mine engineering, available during this period, was aware of the research project's intent with regard to this blast. The loss of much of this blast meant little data would be available upon which to base an assessment of shovel performance while digging muck from a square blast pattern. It was elected therefore, to conduct a second bench study based solely on a square blast pattern Blast EM#4.

Loading instructions.

Loading instructions for each blast have been considered, and a relative strength rating has been applied to each blasthole row in each of the blasts studied. The strength rating considers the amount of each type of explosive in each hole, multiplies this volume by it's associated relative bulk strength and sums the total strength values for each explosive type in the hole. The strength ratings for each hole in the row are averaged to arrive at an average strength rating in total energy units for the blasthole row.

Figure 5.5.3 is a plot of the westward variation of the distribution of explosive energy within blast EZ#3. Note the significant increase in explosive energy towards the back of the blast, to contend with the massive sandstone unit in this area.

Figure 5.5.4 is a panel diagram which illustrates the distribution of explosive types (varying strengths) throughout the blast. Note, due to space / scale limitations, not every blast hole row is depicted in the panel diagram. A panel is included only when a significant change in loading instructions from one row to the next occurred. Each panel represents

therefore, the characteristic loading instructions for a couple of rows on either side of the panel shown.

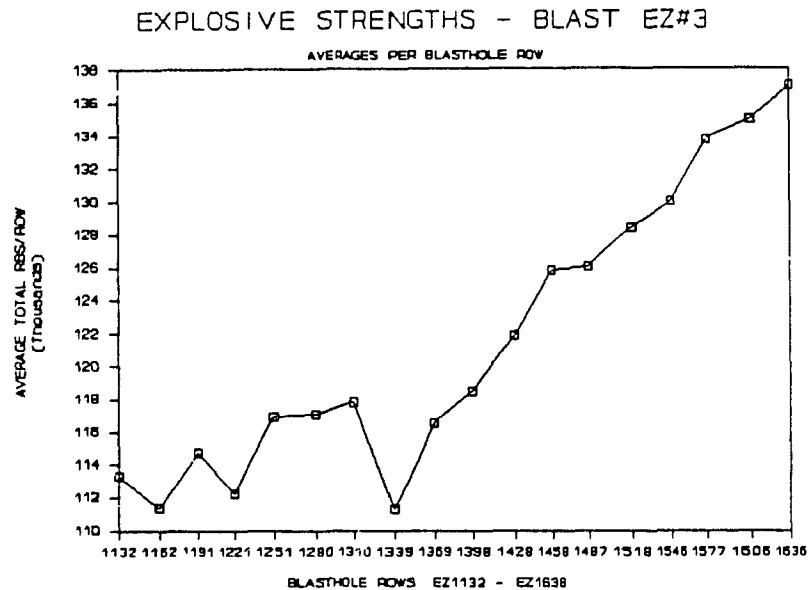


Figure 5.5.3 - Distribution of explosive energy - Blast EZ#3.

5.5.2 Blast EM#1

The design specifications for blast EM#1 are given in Figure 5.5.5. This blast was drilled in an equilateral triangular pattern in which spacing equals 10.7 meters and burden is equal to 9.3 meters. The tie-in and firing sequence is given in Figure 5.5.6, which shows a staggered "V1" initiation sequence was employed ($S_e:B_e = 3.5$). The inter-row delays were held constant in this blast at 50 ms.

The geology of blast EM#1 is characterized by a transition from the massive sandstone unit that predominated much of EZ#3, to a more variable sequence of weaker ground. Here, thinner sandstones units along with weaker mudstone and siltstone units are inter-spaced by a series of very friable coaly mudstone bands up to two feet in thickness. The frequency of these coaly bands increase towards the west of the pattern.

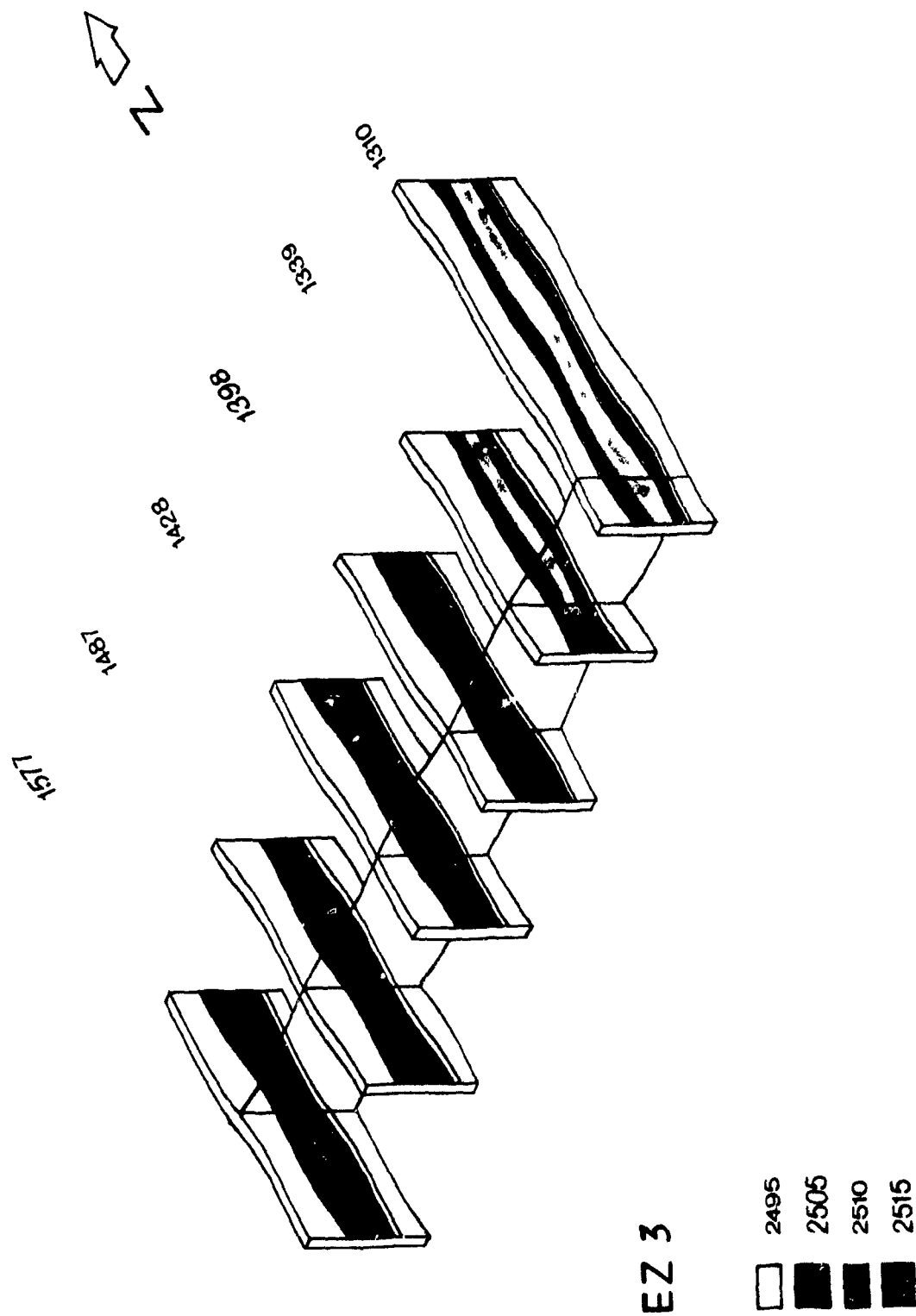


Figure 5.5.4 - Panel diagram of explosives loading Blast EZ#3.

Figure 5.5.5 - Blast EM#1 design.

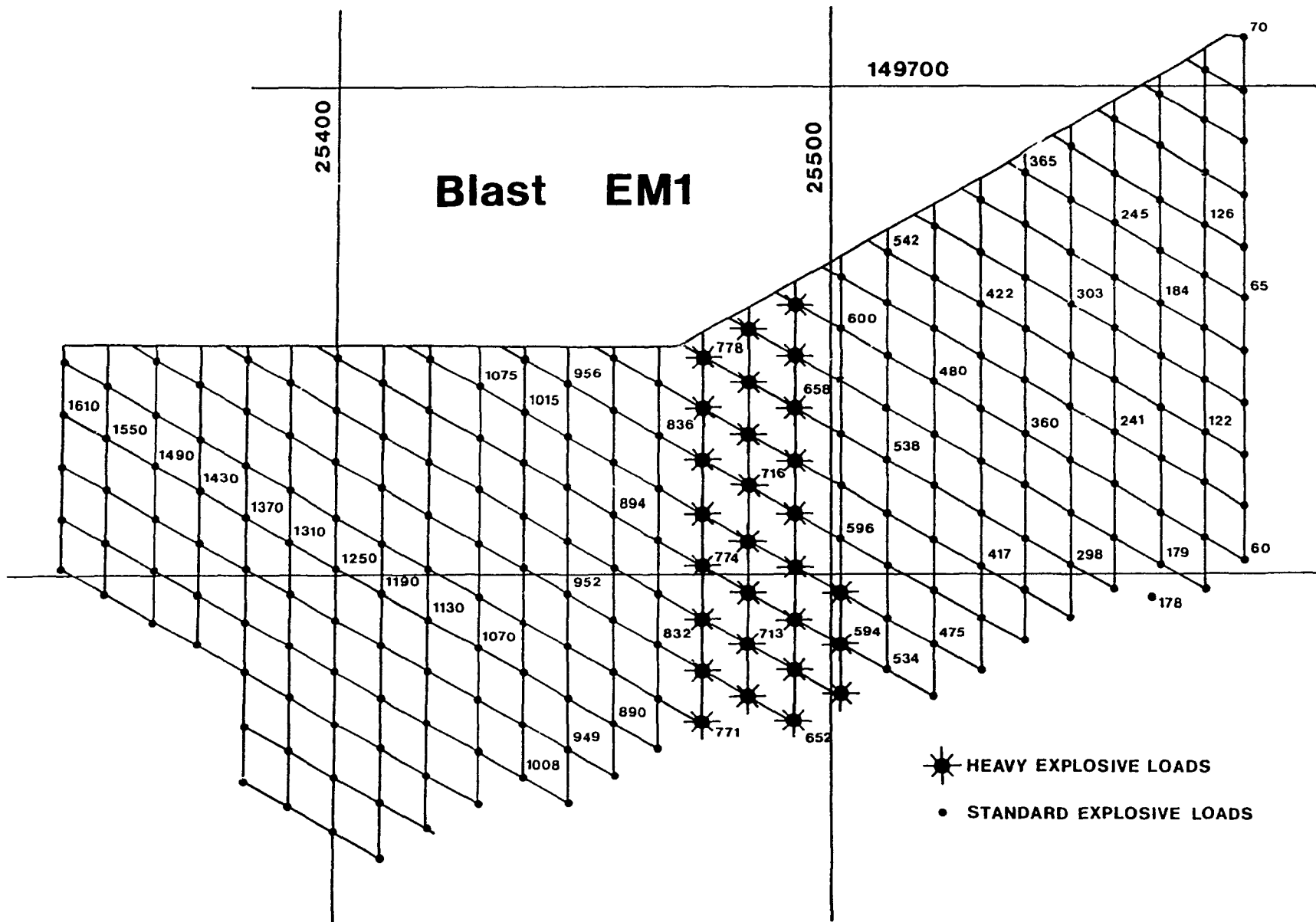
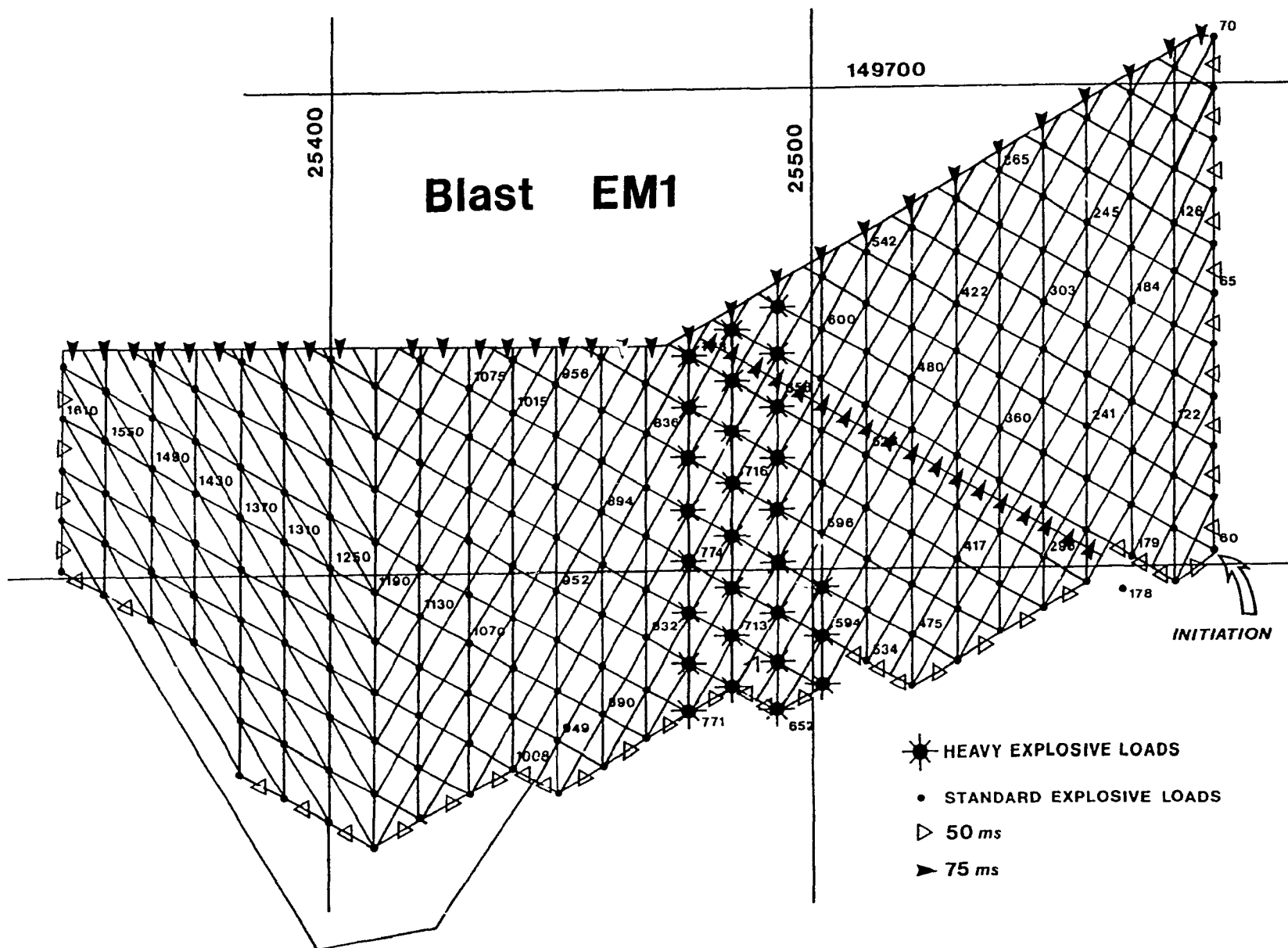


Figure 5.5.6 - Blast EM#1 tie-in and firing sequence.



Loading instructions.

It was decided to vary explosive energy in this blast, to assess the influence of a significant increase of explosive strength on the breakage process and hence, diggability. To this end a section of the blast was intentionally loaded with explosive columns of uncommonly high strength. As indicated on

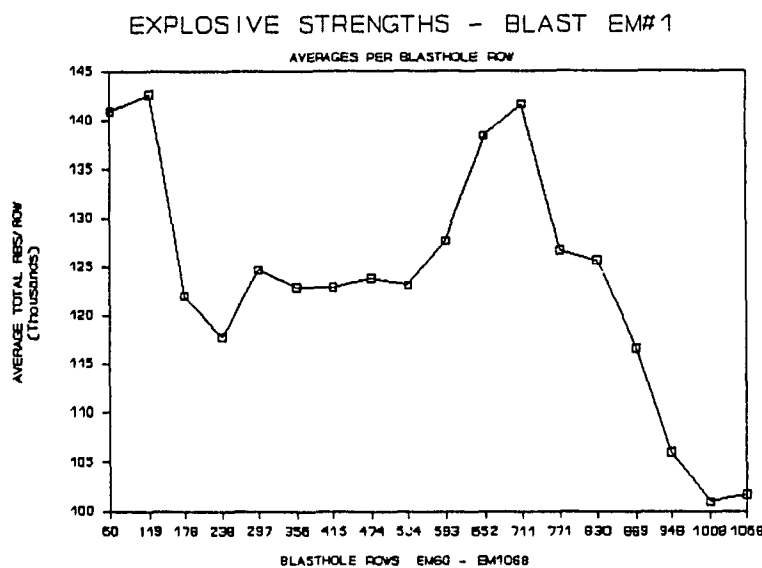


Figure 5.5.7 - Distribution of explosive energy - Blast EM#1.

Figure 5.5.5, 28 holes, in three and a half rows, were loaded with SuperAN 2515 (RBS = 130) in single explosive columns of (on average) 38 ft each, ie. holes filled to 62 feet with standard 24 ft collars.

Figure 5.5.7 illustrates the westward distribution of explosive energy through the blast. The first two rows of the blast continued the loading strategy adopted in blast EZ#3 to contend with the massive sandstone unit. This was a choked blast, as it was fired in advance of the complete excavation of blast EZ#3. This explains the use of such high strength explosives for the first two rows of the blast. The front of blast EZ#3 shot to a free face, hence, the low to average explosive strengths for the first half of the blast. The massive sandstone is no longer

present beyond the first two blasthole rows of blast EM#1. The more variable/weaker geology previously described in this chapter dominates, and explosive strengths are correspondingly reduced. The abrupt increase in explosive energy seen towards the end of Figure 5.5.7 is associated with the intentional heavy loading of blastholes described above. Figure 5.5.8 documents the distribution of explosive types throughout the blast.

5.5.3 Blast EM#4.

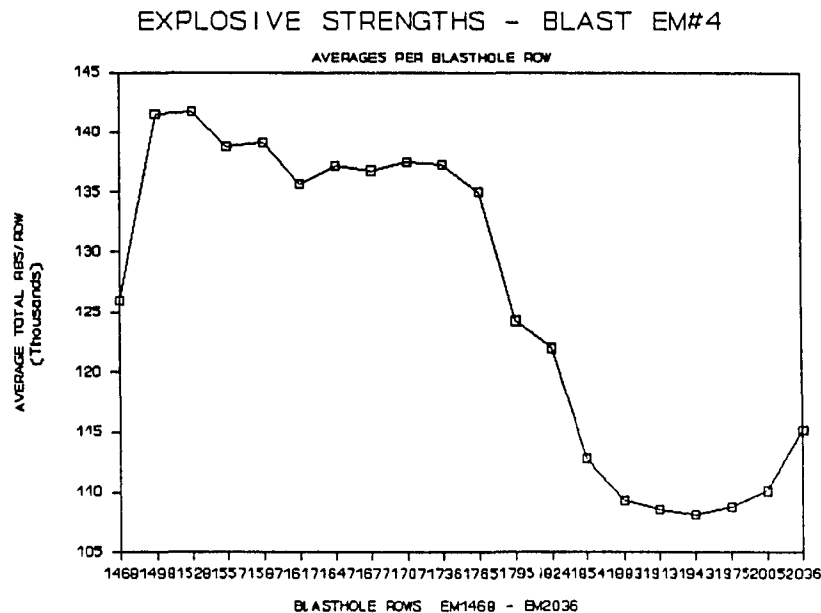


Figure 5.5.10 - Distribution of explosive energy - Blast EM#4.

During the second field period; October 8th to 29th, 1988, blast EM#4 was excavated. This blast employed a square drill pattern with burdens and spacings equal to 10 m (Figure 5.2.2). The tie-in and firing sequence for this blast is depicted in Figure 5.5.9. It is important to note the location of blast EM#4 relative to EZ#3 and EM#1. Referring again to Figure 5.1.1, it is evident that the geology (structure and stratigraphy) that characterized blasts EZ#3 and EM#1 will be continual into the bench region to the north bounded by blast EM#4. This meant the performance of a square blast pattern could be assessed in both the massive sandstone unit that was prevalent through much of EZ#3 and also in the more variable, weaker geology found in EM#1. In other words, by studying the bench area north of EM#1 the

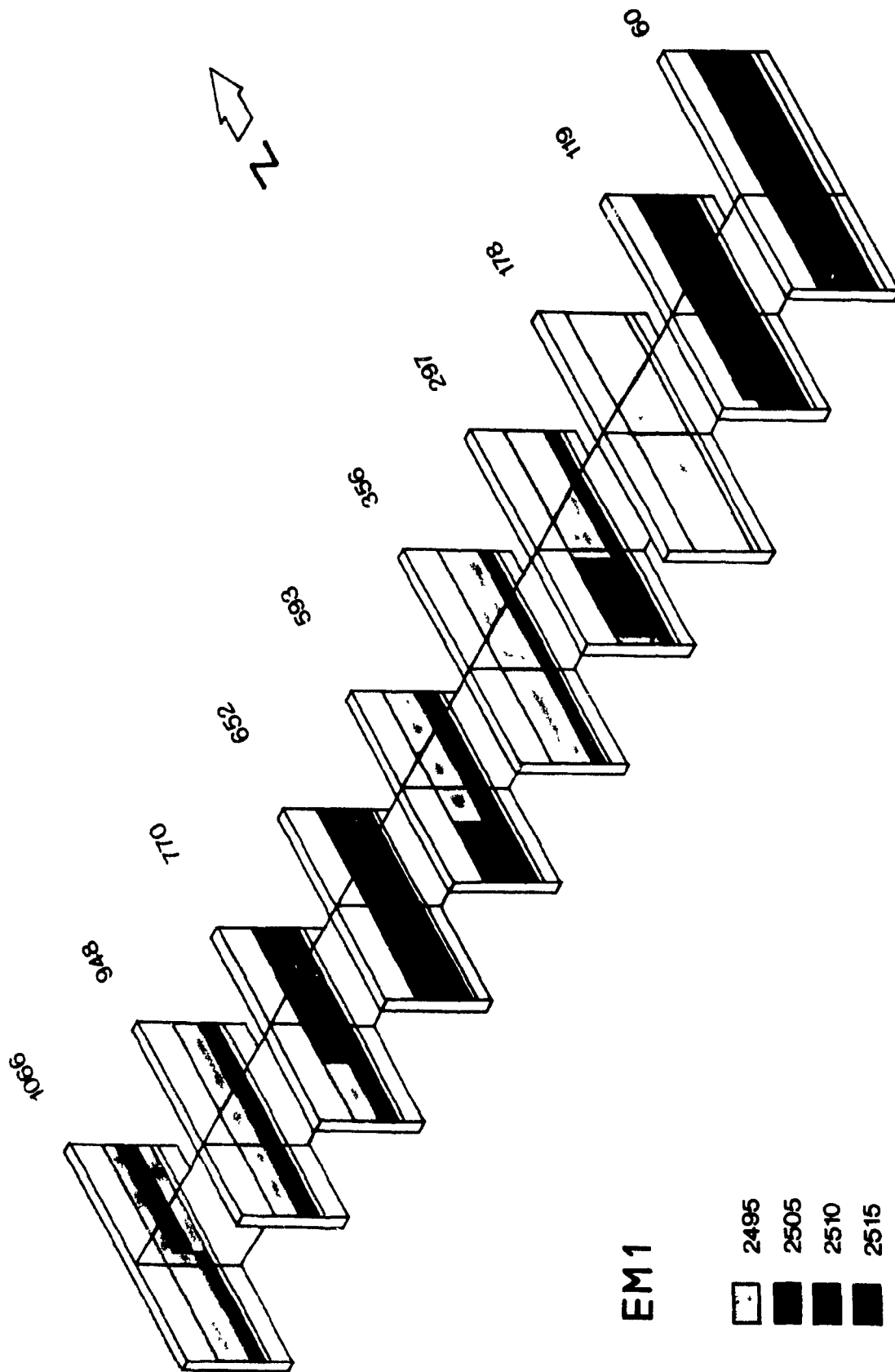


Figure 5.5.8 - Panel diagram of explosives loading - Blast EM#1.

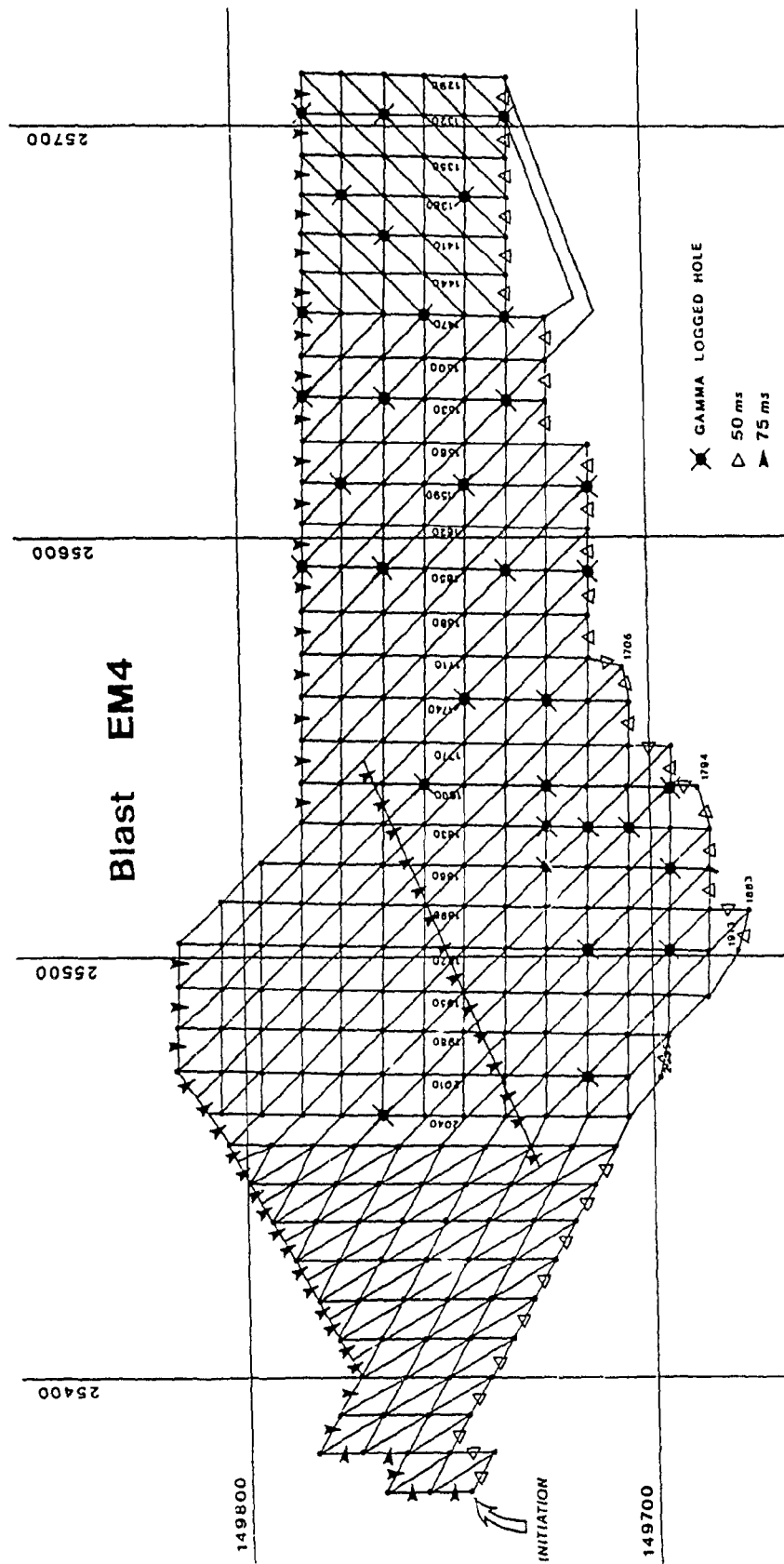


Figure 5.5.9 - Blast EM#4 tie-in and firing sequence.

relative effectiveness of both square and equilateral patterns, were studied in identical geologic environments.

Loading instructions.

Figure 5.5.10 and 5.5.11 reflect the same loading practices described for blast EZ#3 and EM#1: High strength explosives used within areas influenced by massive sandstone, and lower strength explosives towards the west of the pattern, in weaker waste units.

5.6 SUMMARY.

This chapter has described the designs of individual blasts executed within the mine bench selected for the integrated study. Performance monitored data from an instrumented blasthole drill has been effectively used in conjunction with gamma logging and core samples taken from the bench, to accurately define variation in bench geology.

The blast designs described in this chapter were intentionally varied in order to generate the widest possible spectrum of digging conditions. It was not the aim of the current research to advance the state of knowledge surrounding the principles of rock fragmentation by blasting. In this regard, it has been more important to assess the results of each blast in terms of how muckpile properties such as fragmentation size distribution and looseness, impacted on shovel performance - not necessarily how the observed changes in muck characteristics resulted through the varied blast designs employed.

The relation between blast design elements and post blast conditions, is too complex to assess within the scope of the current research. Typically, several trials with each blast design would be required to draw reliable conclusions regarding their relative efficiencies.

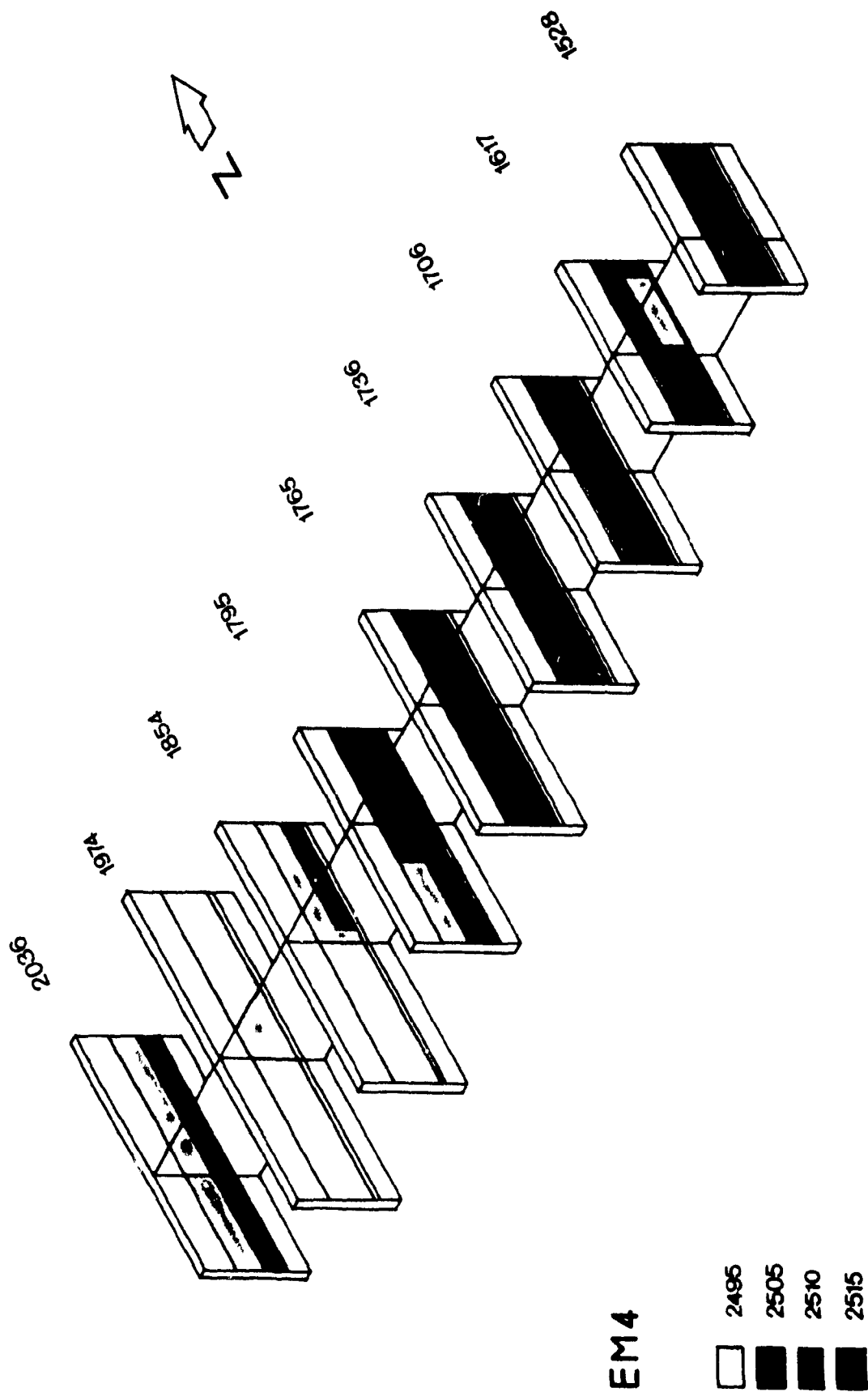


Figure 5.5.11 - Panel diagram of explosives loading - Blast EM#4.

The opportunity still existed however, to attempt a correlation of muckpile conditions (namely fragmentation), and measures of shovel performance (digging effort) with elements of blast design.

Appendix D communicates a brief, simplified summary of the principles of rock fragmentation by blasting, as they pertain to what would theoretically be expected to consequent from the blast designs employed in the current research. Further to this, aspects of the influence of rockmass characteristics on fragmentation will be touched upon, in relation to the transition from massive to jointed rock units identified in the test bench area.

6.0 POST BLAST EVALUATION.

6.1 INTRODUCTION.

It was required that some quantified measure of muckpile conditions be available, against which to compare the monitored performance of the shovel as it excavated each of the test blasts.

The preceding chapter addressed those aspects of a blast that are known to impart a significant influence over blast results (efficiency). The present chapter will briefly discuss several methods commonly employed in surface mining, to rate or assess the efficiency of blasting; post blast evaluation techniques. Such techniques usually focus on fragmentation, as it affects several phases of the mining operation, and hence, overall mining costs.

The post blast assessment technique adopted for the current research is the photographic method, in which photographs of muckpile fragmentation have been processed by a semi-automated image analyzing system, to calculate daily fragmentation size distributions. A detailed explanation of the technique, and the results obtained, follows a review of other methods considered.

6.2 METHODS OF BLAST EVALUATION CONSIDERED.

6.2.1 High Speed Photography.

High speed photography provides a relatively simple and direct diagnostic method of assessing blast performance, design and/or for confirming results predicted by various computer blasting models (Chiappetta, 83, Chung, 75, Lang, 72, Winzer, 79 and Floyd 87). The method employs a camera operating at a rate of between 500 and 1000 frames per second, capable of filming short lived blast events. At such speeds it is possible to record and analyze

the influence of; initiation sequence, delay times and stemming effectiveness on blast efficiency. In addition, aspects such as the nature of muckpile formation (Harries, 1987), measurements of burden velocities and displacements, time of initial face movement and the delay intervals for optimal displacements of burdens can be determined.

The proper filming of a blast involves placing the camera at a location approximately 200 meters from the blast, protected from fly rock, and viewing the bench face at an angle of about 30 degrees. An adequate number of large visible targets should be suspended from the crest of the bench such that their positions define critical portions of the bench ie. the toe, lower and mid sections.

Recent work by Harries (1987), which describes portions of ICI's SABREX blasting model (Tidman et al., 1987 and Jorgenson et al. 1987) has demonstrated that heave velocities of face fragmentation can infer realistic measures of muckpile looseness. The swell (regarded as a measure of looseness) of the muckpile is proportional to the displacement of the fragments from their original positions in the bank, to their final positions in the muckpile. Figure 6.2.1 and 6.2.2 illustrate plots of muckpile profile with respect to original bench profiles, as calculated by the blast model, and post blast surveys of actual muckpile profiles. Note that blasts employing Heavy ANFO, Figure 6.2.2 (with associated higher heave velocities m s^{-1}) resulted in better heave (displacement of burdens) than trials involving regular ANFO; Figure 6.2.1. While the model offers meaningful insights on the dynamics of muckpile formation and looseness, it suffers a few limitations which would have impacted significantly on the current research should it have been applied: First no method has been determined that would permit realistic measures of swell inside the muckpile i.e. beyond the

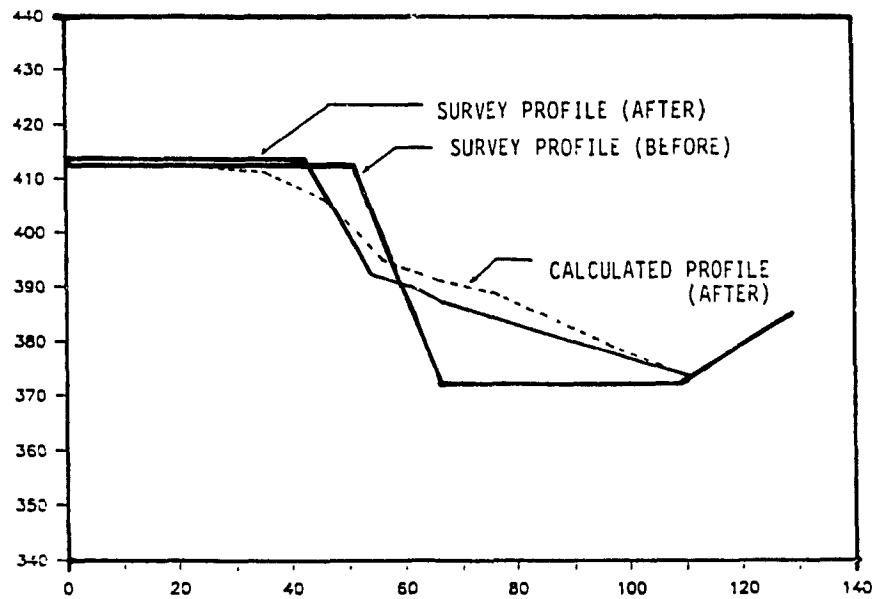


Figure 6.2.1 - Results of blast with ANFO (Harries, 1987)

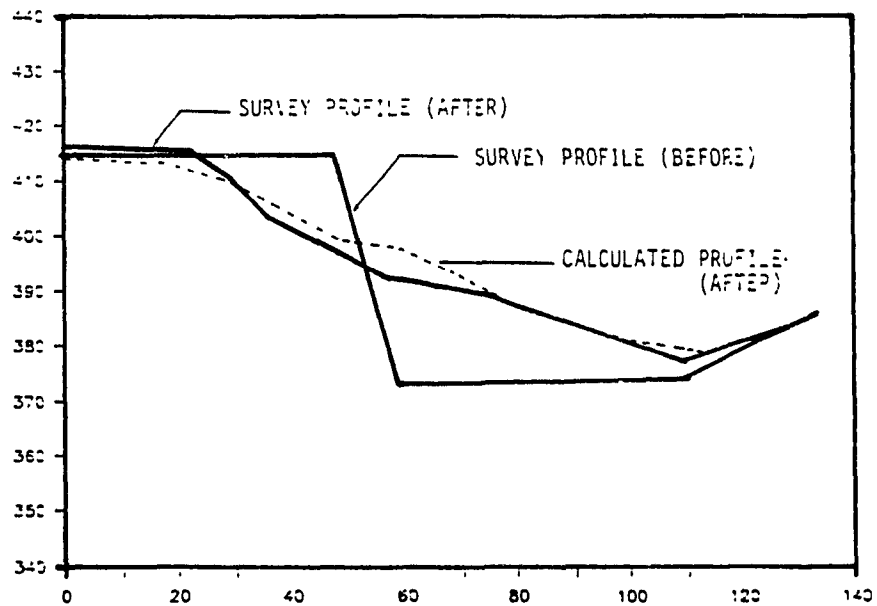


Figure 6.2.2 - Results of blast with Heavy ANFO (Harries, 1987)

surface profile generated by the first few burdens of the blast. Secondly, the model cannot predict heave or looseness when the blast is choked, e.g. blast EM#1 fired into blast EZ#3 prior to it's complete excavation (chapter 6).

Clearly, high speed photography can offer some insights into the behaviour of a rock mass during explosive detonation. The data would have contributed to the efforts of the current research. However, the following constraints denied its application: First, the mine could not guarantee the date on which the blasts for the drill monitored bench would occur. Production requirements and scheduling often invoke changes in short term planning, a consequence of which is the delay or advancement of scheduled blasts (on occasion by as much as three weeks in either direction). The required equipment (high speed camera) could only be obtained with ample notice specifying the exact period during which the equipment is desired. Secondly, the only higher ground from which to film the blast afforded a poor perspective. The free face of the bench, for all blasts considered, would not have been visible and hence, the conclusions drawn from the filming of the blast would have been limited.

6.2.2 Boulder Counts and/or Secondary Blasting.

It is possible to infer the relative success of different blasts in operations that experience oversized material in their blasts. Oversized material is generally classified as blocks from a blast that can not be handled by mine equipment such as loaders, hauling vehicles, crushers etc, without secondary blasting. In such cases records of the volume and frequency with which oversized material occur can be related to different blasts to gage their effectiveness. The method is crude but simple to apply.

In the course of the current research no occasion was found to apply the method because the frequency of truly oversized

material encountered was negligible.

6.2.3 Crusher Delays.

Monitoring the frequency of delays at a crusher has been cited by Chiappetta (1983), as a valid indirect means of measuring blast effectiveness since delays are often a result of dealing with oversized fragments. In addition, steel plate wear within, and power consumption by the crusher have also been cited as functions of fragmentation coarseness. This method was not used because the blasted material was all waste, sent directly to spoil piles.

6.2.4 Loading Rates - Time Studies.

Application of this method assumes that the rate at which loading equipment operates is related to the size of the fragments it is handling. According to Chiappetta (1983) other factors such as the presence of misfires, oversize and poor toe conditions will be reflected in the evaluation. The method is regarded as a reliable indicator of fragmentation when it is conscientiously applied and the effects of different operators are taken into account.

On the other hand, results of an investigation by Grant et al., 1983, to monitor fragmentation in an open stope operation (Mount Isa Mine, Australia), indicated that the monitored dig times of LHDs showed little correlation with observed fragmentation size distributions. Note however, that fragmentation was assessed on a visual basis only and cycle times were monitored on a very limited basis. The results therefore cannot be regarded as conclusive, or for that matter, indicative of the behaviour of shovels in a surface environment. This study will be addressed in more detail in section 7.2 which addresses prior diggability investigations.

The preliminary investigations conducted at the onset of the current research focused on cycle time studies. A complete

discussion of the method, its application, limitations and results was presented in chapter 5. The method suffers from several limitations. It was applied to gain a relative indication of blast effectiveness from one mine environment (differentiated on the basis of blast design and bench geology) to another. The method, used alone, did not permit determining what factors (ie fragmentation, muckpile shape or looseness etc.) were responsible for influencing a blast's associated load (dig) times.

6.2.5 Visual Assessment.

This is the most readily and frequently applied method of blast evaluation, unfortunately it is also the least accurate. It simply involves visually inspecting the surface of a muckpile after a blast, at which time a subjective assessment of fragmentation is made, based on a comparison of the results observed in previous blasts. Only large differences are distinguishable with this method. Rarely however, do most blasts result in large differences from one to another, unless something has gone very wrong. This means less dramatic differences, indiscernible to the "mind" of the observer go unnoticed, when in fact they may be indicative of significant differences between results of separate blasts.

6.2.6 Summary.

The preceding methods described are for the most part indirect means of assessing blast fragmentation. The only truly accurate means of determining fragmentation size distributions is to sieve the entire muckpile. This is of course seldom practicable. Many researchers agree however that the best practical means of obtaining a quantified estimate of true size distribution is with the photographic method. The method, as has been applied in the current research, involves photographing the muckpile at regular intervals during its excavation, and employing computer digitization to calculate the size distributions from the photographs.

6.3 PHOTOGRAPHIC MEASUREMENT OF BLAST FRAGMENTATION.

Several investigations have been undertaken to ascertain the credibility of the photographic method. The method employs various procedures as outlined in Noren (1974), Reid (1976), Carlsson (1983), Maerz and Franklin (1986, 1987) Chiappetta (1983), Sheikh and Chung (1987) and Nie and Rustan (1987), to photograph muckpiles and determine fragmentation size distributions through either manual, or semi to fully automated measurements (computer digitization) of fragments in the photographs.

Sources of error associated with the photographic method fall into one of two categories; those involved in the actual photographic survey of the muckpile ie. sampling error, and those attributed to the subsequent measurements conducted upon the photographs ie. the digitizing process.

One of the basic assumptions of the photographic method is that the surface fragmentation of a muckpile is a valid representation of the whole muckpile. Work by Noren (1974) and Winzer (1983) suggest that surface fragmentation is in fact a reasonable reflection of the muckpile fragmentation as a whole. A second fundamental assumption is that the void spaces between all blocks are filled with fines. According to Nie (1987), this results in fines being overestimated in the surface layer of a muckpile. On the other hand if fines are excluded, the method becomes biased towards larger fraction.

In employing the technique, it is important to use a camera fitted with a normal or telephoto lens, in order to minimize distortion of blocks in the muckpile due to their distance from one another. A telephoto lens is best suited to the process since it has the effect of placing all elements in a photograph effectively within the same plane. Additionally, the photographs should be taken as near to normal to the surface of the muckpile

as possible in order to minimize the effects of distortion.

Finally, when digitizing the photographs for computer calculation of size distribution, errors are introduced by the inability to accurately follow individual block contours with a digitizing pen.

It must be noted, apart from potential sources of error associated with the photographic approach, a significant limitation of the method is the inability to assess variations in fragment shape. As will be explained in the following section, the approach requires that all fragment areas be converted to areas of equivalent circles, and thence, by assuming a uniform height or "Z" dimension for each fragment, to spherical volumes of equivalent diameters. Apart from the errors introduced by assuming uniform block heights in determination of area distributions, the implications on ratings of diggability are more significant. It can be assumed, that since different shaped fragments will assume different packing densities (Dallavalee, 1948), the resistance offered to excavation of equal size distributions but of different shaped fragments will be distinct.

Within the confines of the present study, variations in fragment shape have not been significant, as the rock mass structure within the test bench area was fairly uniform, yielding consistently orthogonally shaped blocks. The potential influence of variations in fragment shapes on muckpile diggability, will be addressed in greater detail in **section 8.3.1; Relationship between and diggability index - DI.**

The preceding has touched only upon the basic application and some limitations, of the technique of determining fragmentation size distributions from muckpile photographs. It must be appreciated that the use of this technique in the current research has not been to determine "precise true" size

distributions for each muckpile considered, nor was it to advance the application of the method. Its use was intended solely to obtain, on a daily basis, an objective and reasonably accurate means of comparing the "relative" fragmentation size distribution being handled by the shovel from one blast environment to another.

This meant that the photographic survey of fragmentation need only be over the narrow range of the bench actually being excavated by the shovel. Since during the course of a typical days monitoring the shovel remained in one location, this meant only photographing an area of the muckpile equal to the approximate swing radius of the shovel; 75 feet across.

To this end, an innovative means of muckpile photography has been developed and a semi-automated image analyzer-processor has been acquired to subsequently process the photographic data. The following section will explain in detail the approach followed, which minimizes several of the errors associated with a more traditional application of the technique.

6.3.1 Application of the Photographic Method.

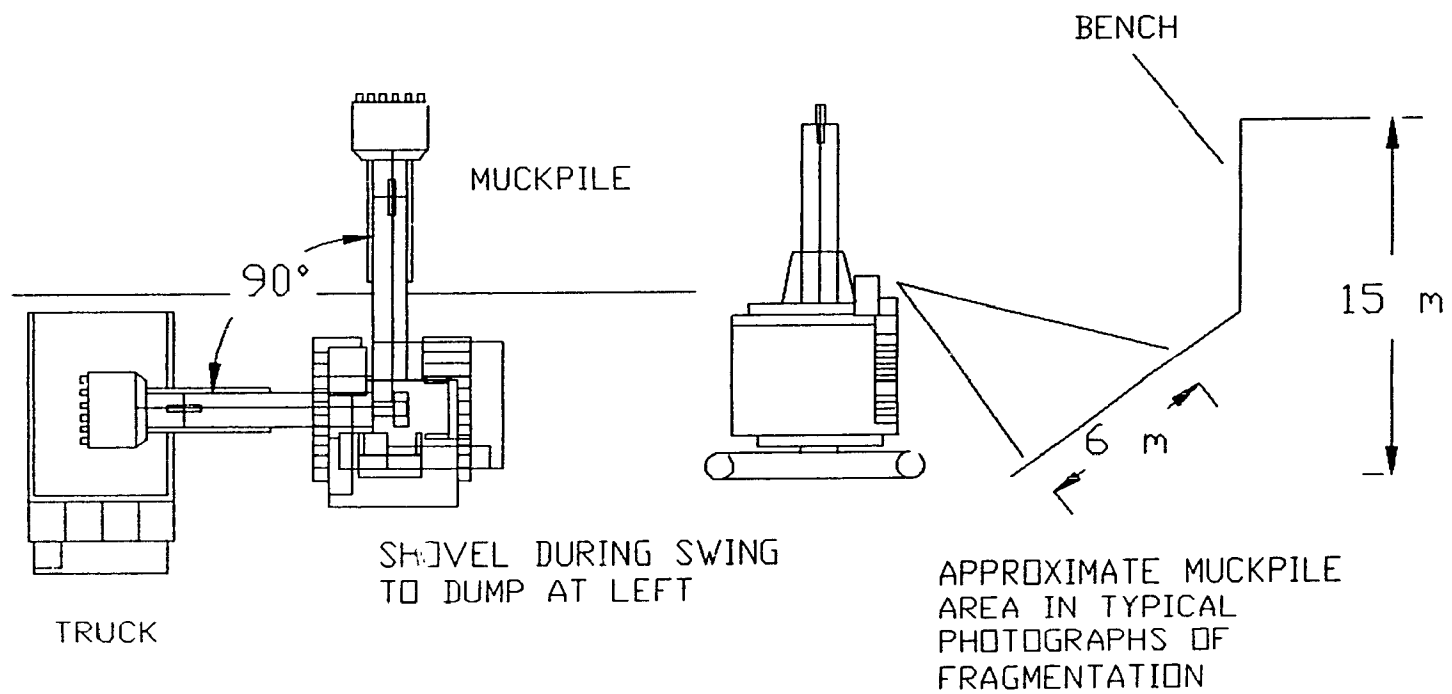
The responses of the monitored shovel operating parameters must be relatable to some quantifiable characteristic of the muckpile. The only aspect of a muckpile that can be quantified without disrupting the production cycle proved to be the fragment size distribution. Provided identical daily procedures are followed and that a large enough sample is investigated, an examination of fragmentation should yield the least subjective and most quantifiable assessment of muckpile conditions.

An approach was developed to dynamically determine fragmentation size distributions on a continual basis during monitoring periods, without disrupting production. The shovel was boarded daily for four hour monitoring periods, with upwards of 30 one meter stakes cut from 1" X 1/2" wood strips. These

dispensable stakes were thrown from the walkway external to the operators cab, on to the muckpile to provide scale for the photographic survey of muckpile fragmentation. The stakes were thrown when the shovel swung to dump a load into a truck on the operators left. In this position the photographer is directly above the toe of the muckpile and facing the bench, perpendicular to the muckpile surface Figure (6.3.1). Furthermore since the shovel remains an equal distance out from the toe of the muckpile, all photographs are taken from the same, and as near to an optimal perspective possible. These conditions created close to an ideal situation under which to conduct a photographic assessment of muckpile fragmentation. A normal lens was used since there was little time during the dump cycle to contend with adjusting and focusing a telephoto lens. This has meant that all photographs comprise the same surface area of the muckpile; approximately 6m x 8m. Given the large area encompassed by each photograph, it is felt that the entire muckpile area of concern, has been photographically surveyed for each day of monitoring. The current photographic approach minimizes sampling error by including between 800 - 1500 blocks per photograph.

During a typical study the shovel would remain working the same portion of the muckpile for the entire four hour period. Thus, the series of photographs taken during the daily studies, will in essence represent a cross-section through that volume of the muckpile excavated during the study period. In this sense, the photographic record taken as a whole, is not solely representative of the muckpiles "surface" characteristics at one moment in time.

Figure 6.3.1 - Method of muckpile photography from shovel.



6.3.2 Image Analysis of Fragmentation Photographs.

The fragmentation size distributions for each photograph have been determined through the use of a semi-automated image analysis system which employed software recently developed at the University of Waterloo; WIEP (Waterloo Image Enhancement Program), by Franklin and Maerz (1986, 1987). This software, was donated (by Franklin and Maerz) to the department of mining engineering at McGill University for the purposes of the current research.

Hardware requirements for use of the WIEP software included the following, which were purchased and installed/connected to a COMPAQ 386 computer:

1. One Imaging Technology PC VISION PLUS
512-3-U-AT Image processing board.
2. One ELECTROHOME high resolution
RGB analog color monitor.
3. One COHU CCD 4800-1500-0000 solid state
monochrome video camera.

The procedure followed for image analysis of the photographic records of muckpile fragmentation was as follows:

Of over 175 photographs of muckpile fragmentation taken during the course the research, 64 were selected for comprehensive analysis. The selection process was based upon: a) The quality of photographs; clarity, lighting etc. b) The orientation of the scale stake in each photograph (i.e. it should be as plane to the muckpile surface as possible). c) The number of photographs selected from each day was based upon whether significant changes in fragmentation conditions had occurred (i.e. if conditions remained constant, two or three photographs

were sufficient to document the daily average fragmentation size distribution). These 64 photographs were then developed in an 8" x 10" format.

The Waterloo image analysis system was not designed with the capability to automatically detect the boundary of each fragment in the photographs of muckpile fragmentation. For this reason, each photograph was overlain by a sheet of clear acetate and a permanent ink marker was used to manually trace the outline of the fragments within the central most portion of each photograph (e.g. leaving an average border of about 1.5" to the edge of the photograph). In this fashion only wholly exposed fragments were considered in the analysis. Once tracing is complete, the length of the scale stake in the photograph, is recorded on the periphery of the tracing in the form of a line segment. The acetate tracing was then photocopied on paper to generate a high contrast copy. Figure 6.3.2 is an example of the fragment outlines traced from a photograph typical of good (fine) fragmentation.

These manually derived block outlines were then subjected to the image analysis system for determination of actual fragmentation size distributions. The tracing of block outlines is placed below the video camera and the image is digitized by the system, and displayed on the high resolution RGB monitor. Cross hairs then appear on the display monitor which are manoeuvred by the analyst from one end to the other, of the line segment defining the scale for the tracing. The scale is

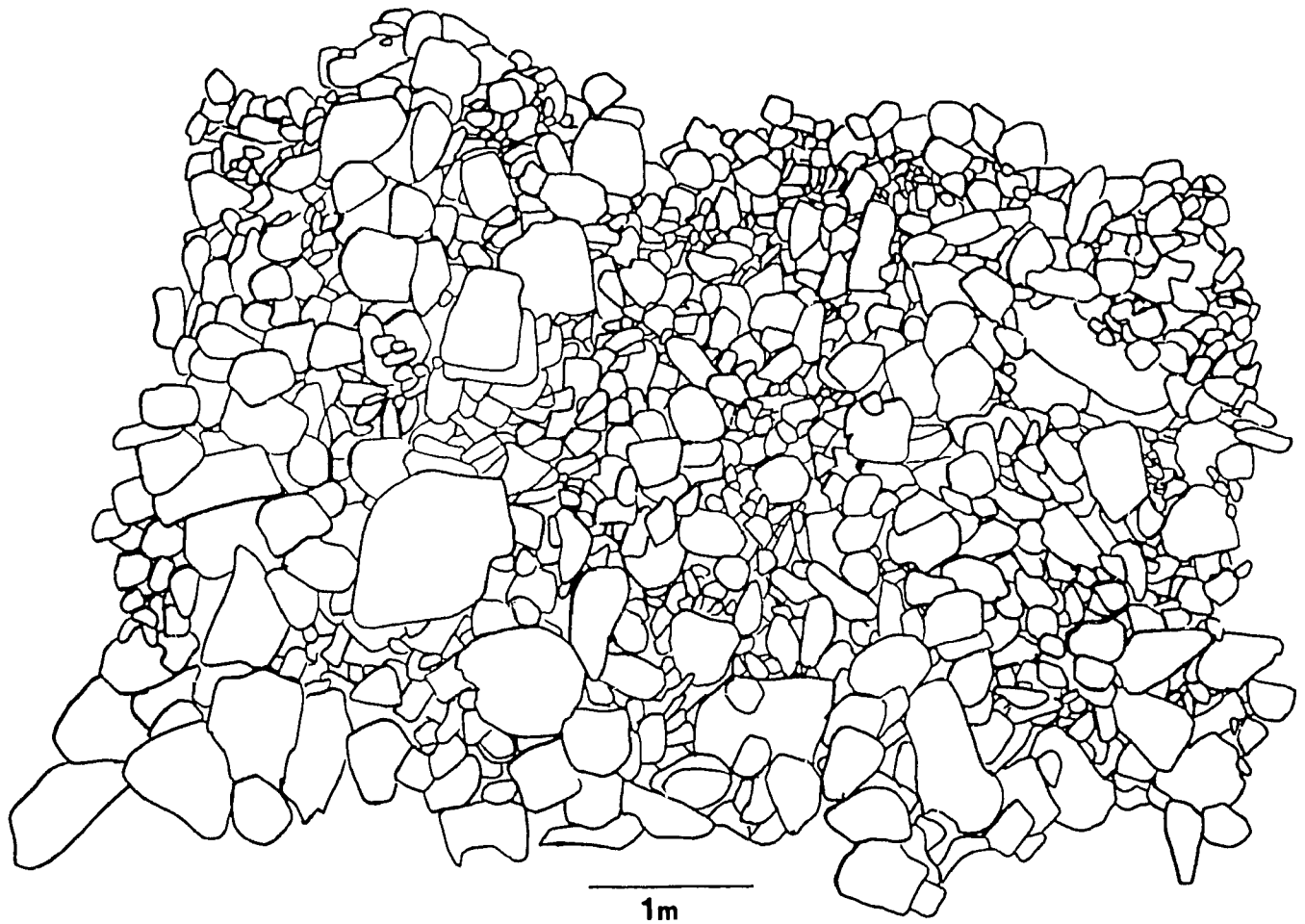


Figure 6.3.2 - Manually traced fragment block outlines.

thus recorded by the system as equivalent to the number of pixels traversed by the cross hairs along the line segment. The system then executes an edge detection algorithm which identifies the boundary of each block. The number of pixels within each defined block is then calculated, and converted to an actual area measure of the fragment, based on the line segment scale in pixels.

The results of the image analysis obtained from Figure 6.3.2 are presented in Figures 6.3.3, 6.3.4 and 6.3.5. Figure 6.3.3 is the fragmentation (block) size profile distribution, given as the diameter of an equivalent circle. The area of each block is calculated, from which the area of an equivalent circle is determined, the diameter of the circle is then expressed as the

equivalent block size. Figure 6.3.4 is the true block size distribution, in which the frequency distribution of equivalent spheres are expressed. In this manner, raw block areas are transformed into volumes (spheres). Figure 6.3.5 is a graph of the cumulative weight percent for the particular photograph, and assumes an average material density of 2700 kg/cu.m. For comparative purposes, Figures 6.3.7, 6.3.8 and 6.3.9 have been determined from the tracing depicted in Figure 6.3.6; illustrative of coarse fragmentation.

Note the following values are also documented in each of the frequency distribution plots:

min = minimum fragment size in photograph (tracing).
max = maximum " " " " "
n = number of blocks identified in the tracing.
x = mean fragment size.
s = standard deviation.

Included in the cumulative weight percent passing graphs (Figures 6.3.5 and 6.3.9) are the following statistics:

CU = Coefficient of uniformity.
CS = Coefficient of sorting.

Table 6.3.1 below documents the results of the fragmentation analysis of all muckpile photographs. Indicated in Table 6.3.1 are the number of photographs analyzed per day of the shovel monitoring studies conducted in August and October 1988. The statistics (D_{10} - D_{90}) represent the average size

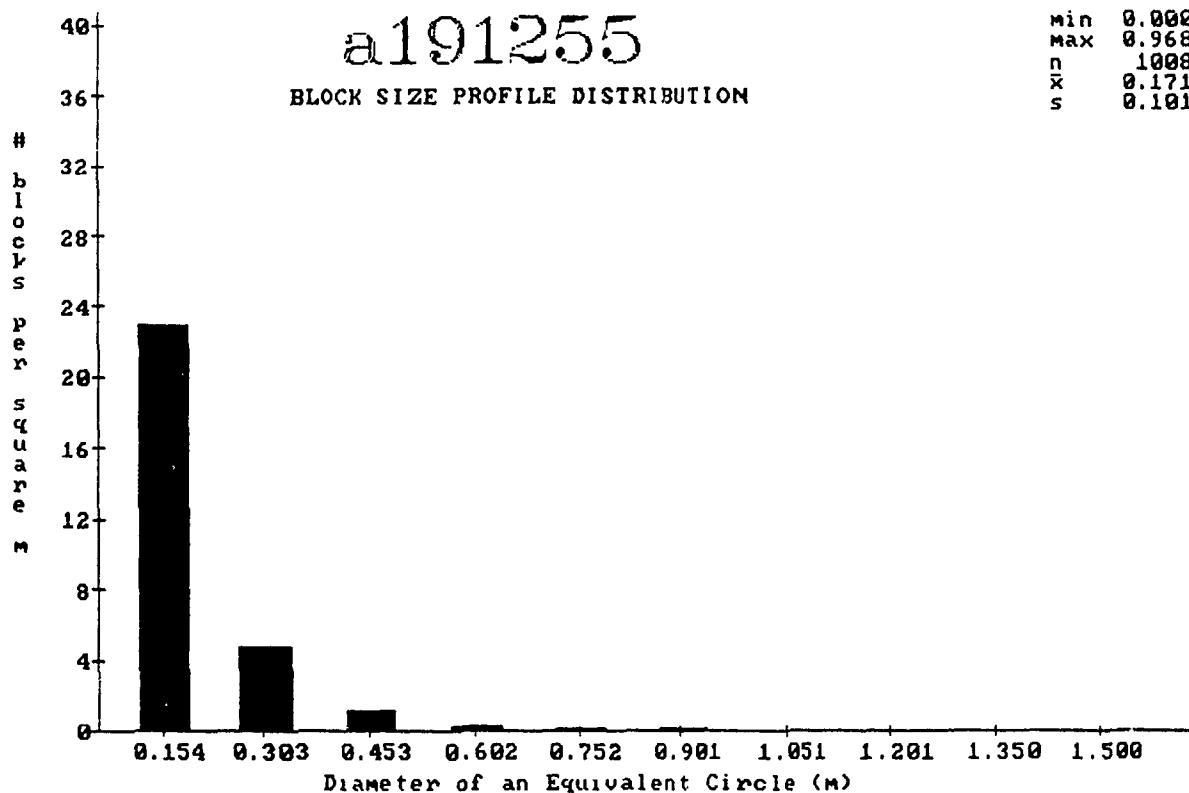


Figure 6.3.3 - Fragment areas determined from Figure 6.3.2.

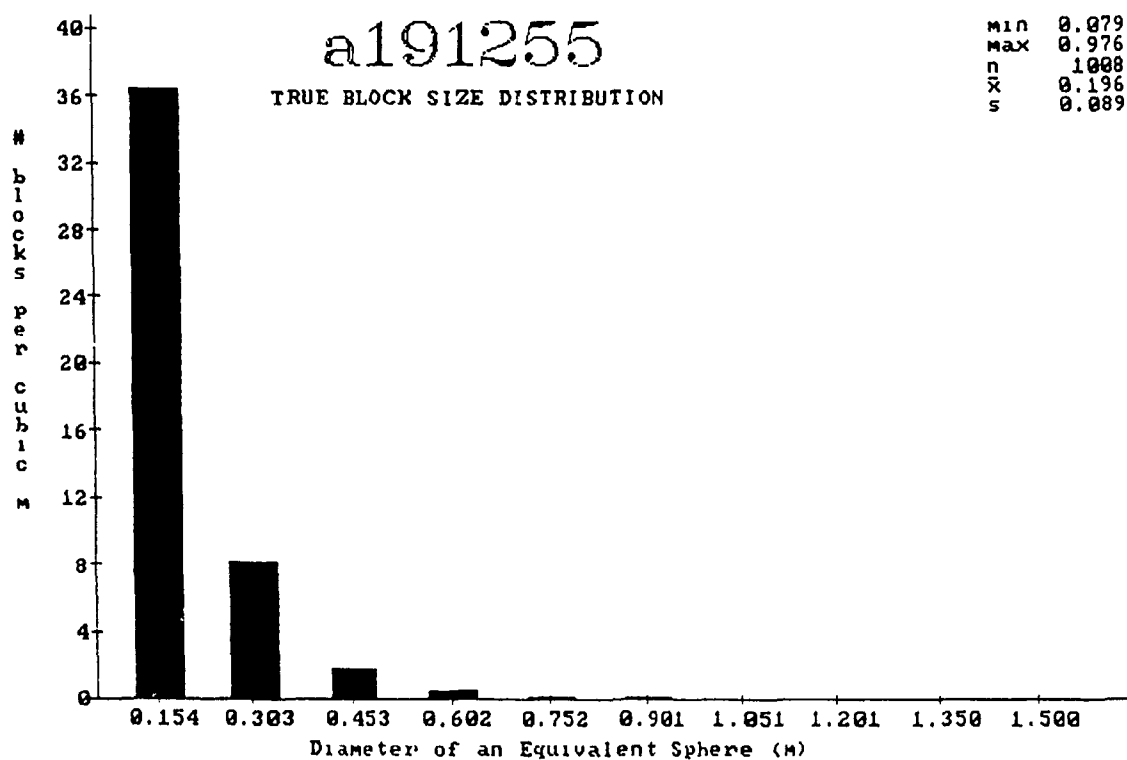


Figure 6.3.4 - Fragment volumes determined from Figure 6.3.2.

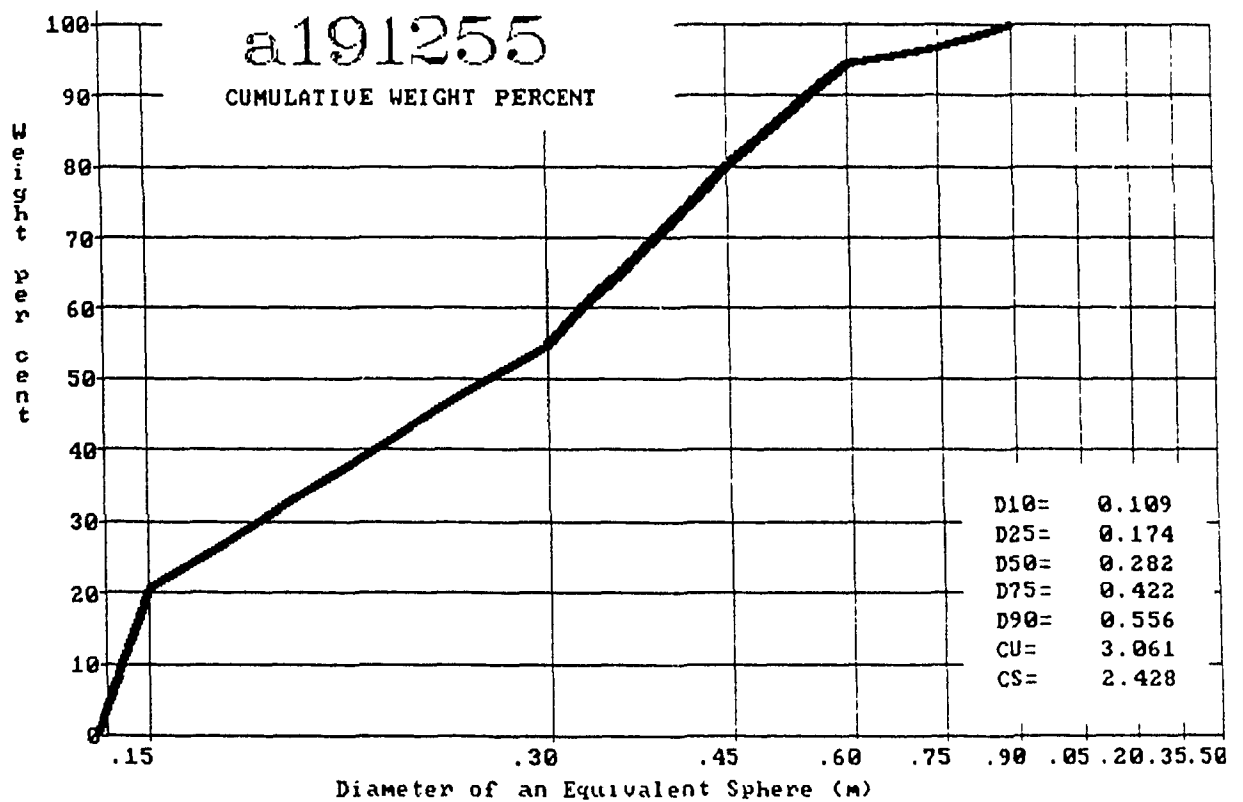


Figure 6.3.5 - Cumulative weight percent for Figure 6.3.2.

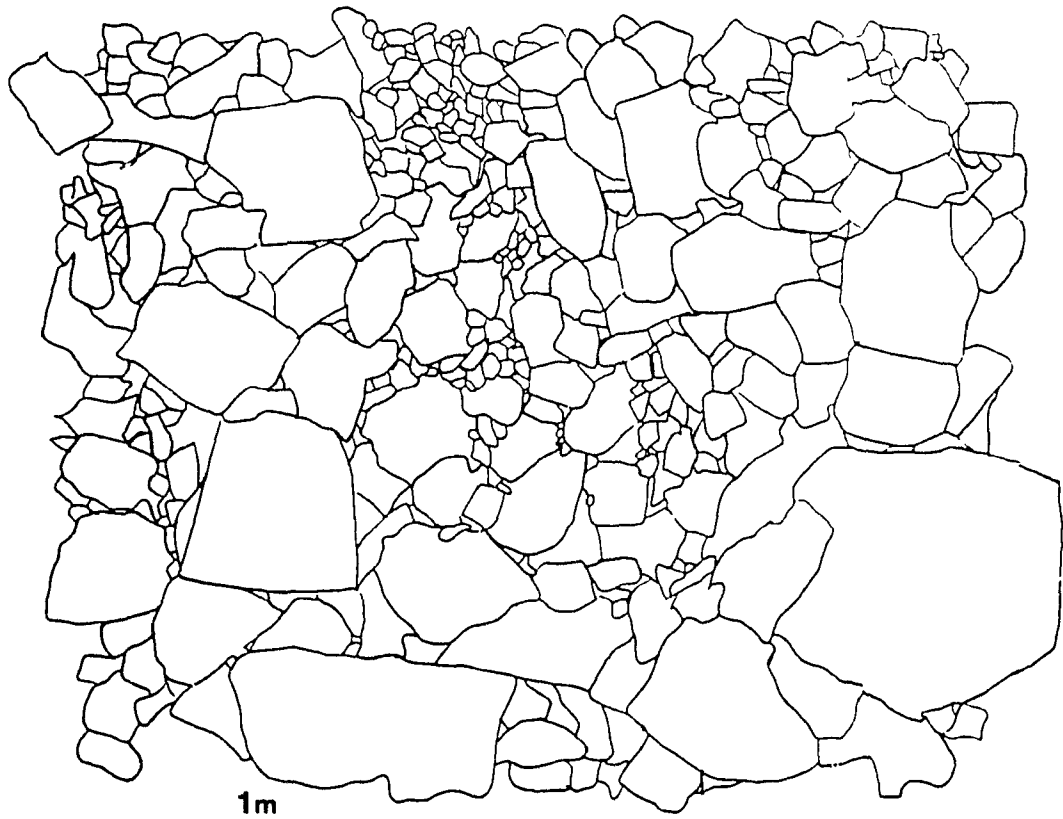


Figure 6.3.6 - Tracing of fragment outlines for coarse fragmentation.

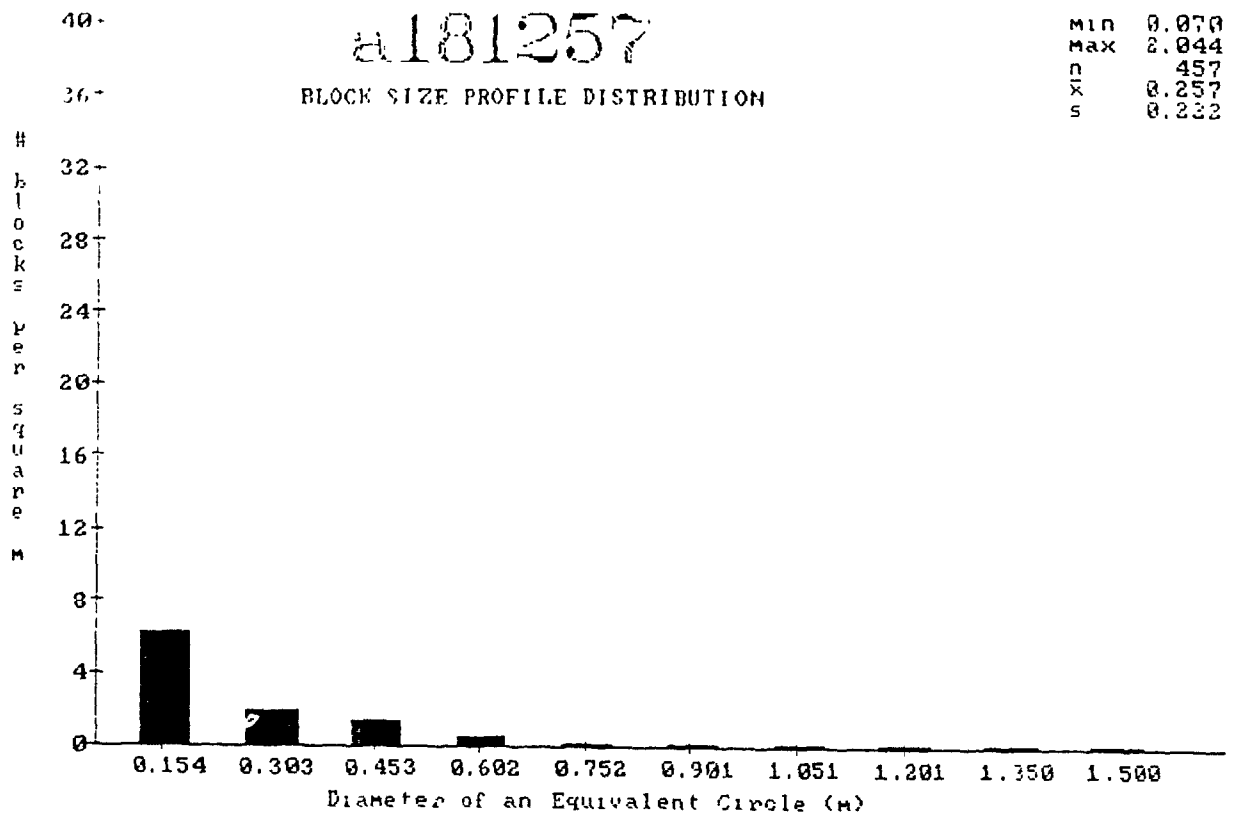


Figure 6.3.7 - Fragment areas determined from Figure 6.3.6.

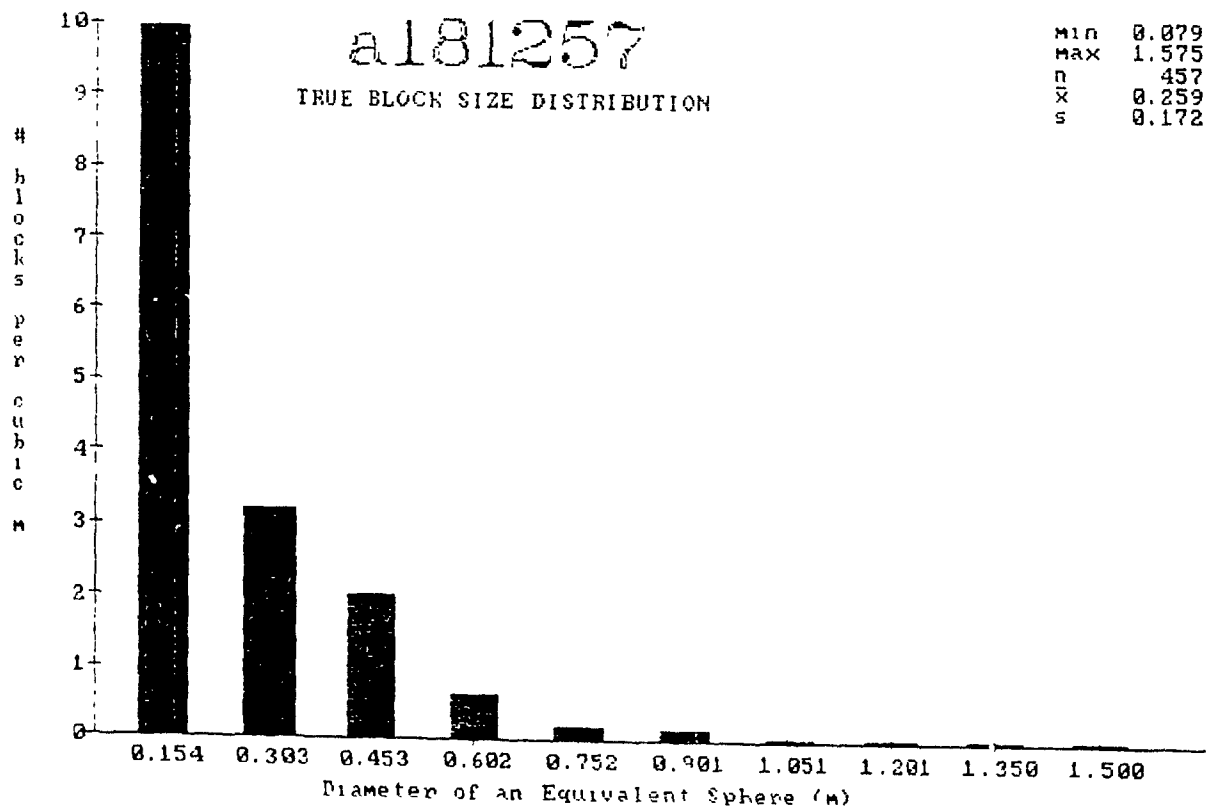


Figure 6.3.8 - Fragment volumes determined from Figure 6.3.6.

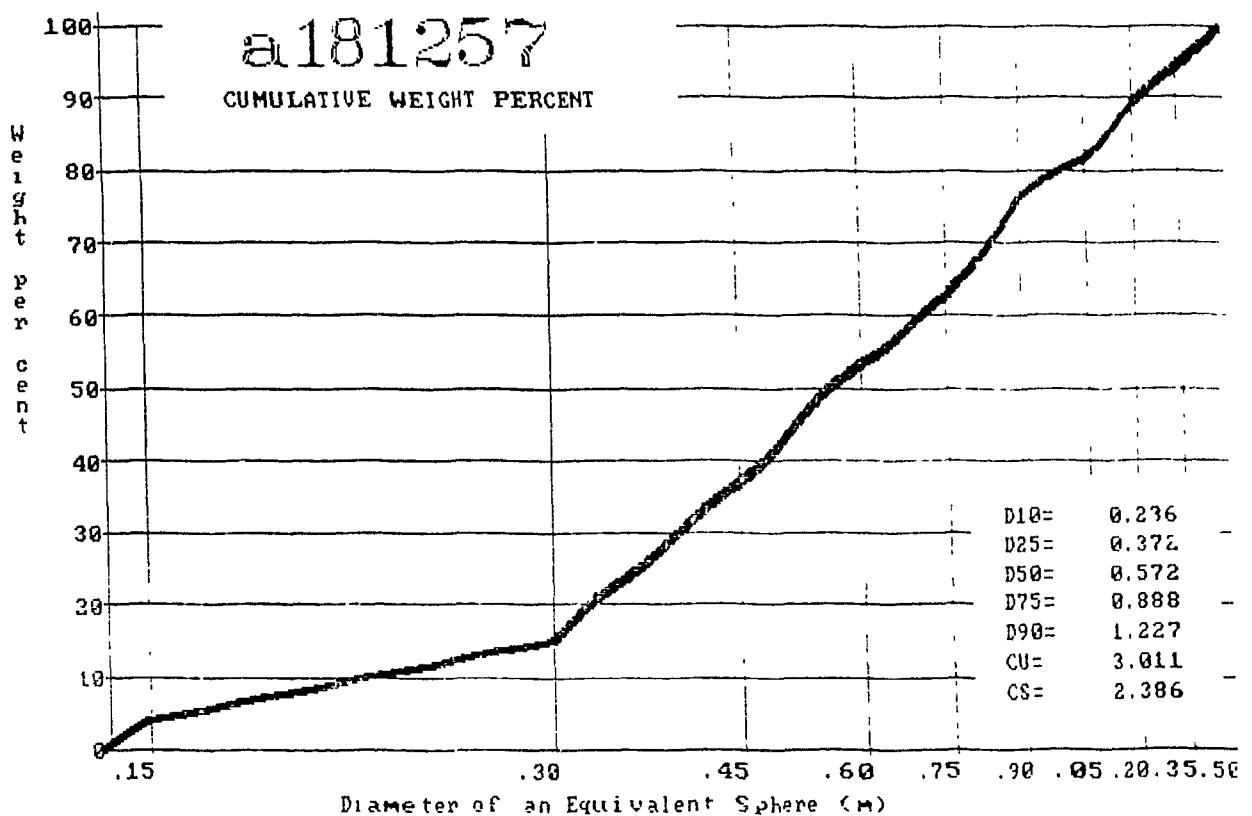


Figure 6.3.9 - Cumulative weight percent for Figure 6.3.6.

distributions for each day. These have been determined by amalgamating the files in the WIEP program that contain the block areas calculated from the individual photographs for each day. The final file containing the block areas for all photographs for a particular day was then processed as per normal sized files, and the statistics presented in Table 6.3.1 were generated. This procedure required modification of the WIEP software program since it was not designed to process data on more than 1500 blocks at a time. The concept of combining results from individual photographs to arrive at a composite or averaged set of measures, had not occurred to the systems designers at Waterloo. The suggestion that this facility would be of benefit, especially to the current research, was welcomed, and modifications were made to the original software by the systems designer (Mr. N. Maerz) at the University of Waterloo.

The averaged cumulative weight percent passing plots for each day of shovel monitoring are presented in **Appendix E**.

The results of the complete fragmentation analysis are depicted graphically in Figures 6.3.10 and 6.3.11 which display the average fragment size (D_{10} , D_{25} , D_{50} , D_{75} and D_{90}) for each day of shovel monitoring. Figure 6.3.10 represents the fragmentation results obtained from blasts EZ#3 and EM#1, monitored during August 15th to 25th 1988. Figure 6.3.11 shows the fragmentation observed during the October 8th to 29th 1988 monitoring of blast EM#4. Note that the lack of data on the graph between the 14th and 26th corresponds to a major repair of all four of #15 shovel's swing transmissions. For ease of reference, the approximate location of the shovel in the blast pattern during each day of monitoring, has been indicated by citing the blast hole to which it was closest.

Table 6.3.1 - Daily measures of muckpile fragmentation (metres)

Date	No. of Photos.	D_{10}	D_{25}	D_{50}	D_{75}	D_{90}
August 15th	3	.237	.357	.521	.747	1.371
August 17th	3	.201	.290	.453	.656	.939
August 18th	4	.252	.388	.618	.889	1.242
August 19th	3	.152	.199	.277	.402	.530
August 20th	3	.134	.182	.262	.378	.485
August 21st	3	.199	.313	.515	.889	1.265
August 22nd	4	.190	.276	.422	.605	.876
August 23rd	5	.175	.245	.371	.528	.769
August 24th	3	.140	.189	.272	.420	.541
August 25th	3	.165	.231	.257	.534	.727
October 8th	4	.169	.245	.422	.644	.859
October 9th	4	.160	.222	.332	.482	.652
October 11th	3	.184	.279	.450	.750	1.022
October 12th	3	.178	.258	.394	.584	.838
October 14th	4	.181	.279	.446	.702	.948
October 26th	2	.113	.166	.255	.446	.970
October 27th	4	.187	.305	.502	.847	1.345
October 28th	2	.125	.181	.273	.428	.695
October 29th	4	.145	.209	.311	.445	.588

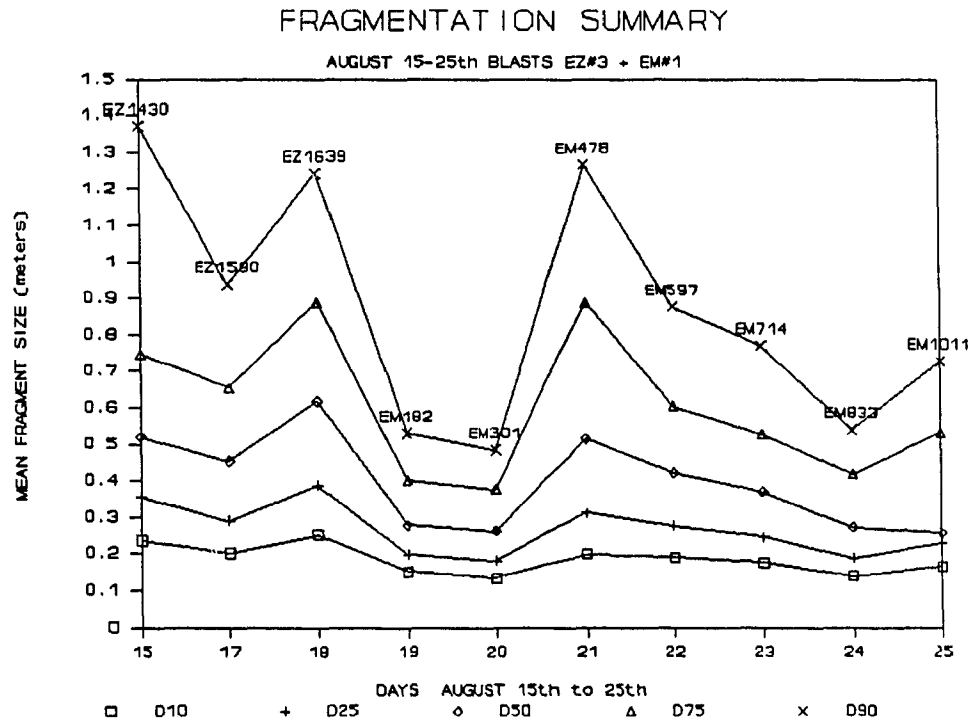


Figure 6.3.10 - Fragmentation summary, blasts EZ#3 and EM#1.

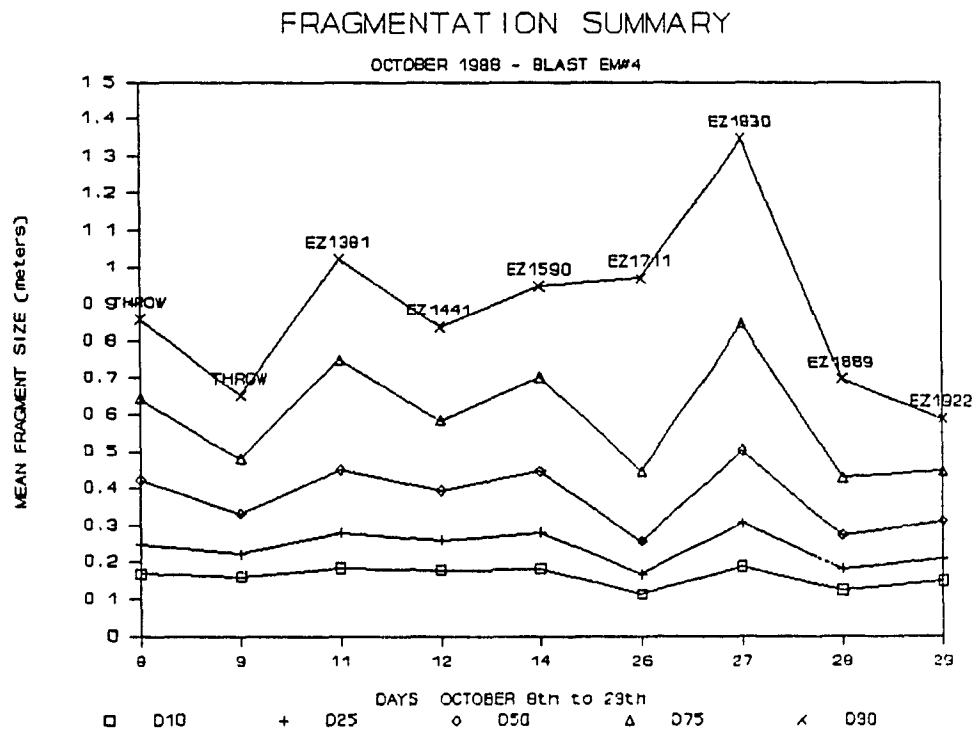


Figure 6.3.11 - Fragmentation summary, blast EM#4.

The data on explosive strength distribution within each blast, presented in section 5.5 has been related to the measured fragmentation for each day. Figure 6.3.12 is a plot of the mean daily fragmentation D_{50} and the composite relative bulk strength of the explosive column, averaged for the blasthole row, at which fragmentation distributions have been determined. Figure 6.3.12 suggests that there exists no direct relationship between the amount of explosive energy used at a particular location in the blasts and the ensuing fragmentation. This was not wholly unexpected, given the several blast variables (chapter 7) involved (other than explosive energy) that determine resultant fragmentation.

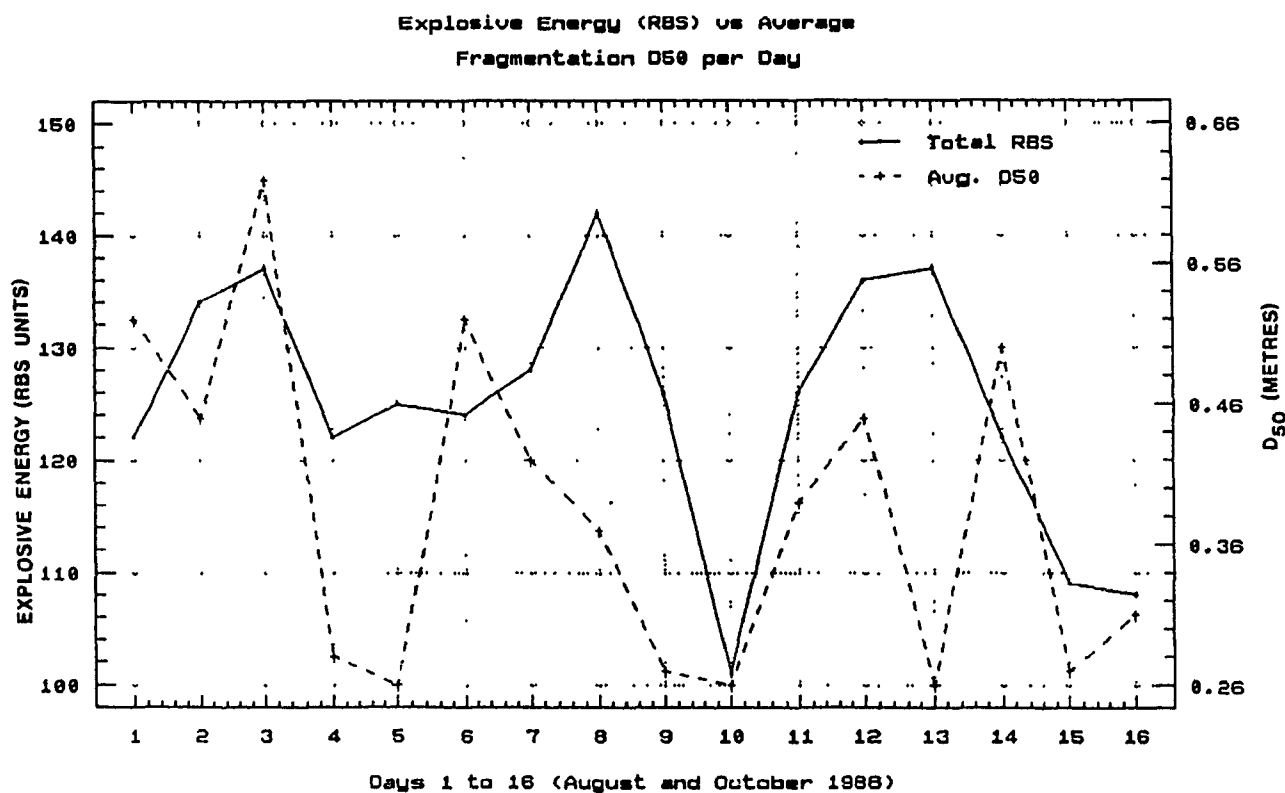


Figure 6.3.12 - Plots of mean daily fragmentation D_{50} and explosive energy (RBS).

6.4 SUMMARY.

A methodology was evolved which has permitted a comprehensive assessment of muckpile fragmentation - on a daily basis, throughout the shovel monitoring studies. In excess of 60,000 fragments have been considered during the course of the investigation. This data has provided a quantified aspect of muckpile conditions against which to compare shovel performance during each day of the study. Prior investigation on shovel performance monitoring have relied on subjective, visual assessments of muckpile fragmentation by operating personnel. These prior research efforts are now addressed in detail, in the following chapter.

7.0 PRIOR DIGGABILITY AND SHOVEL INSTRUMENTATION STUDIES.

It is appropriate, at the beginning of this section, to provide some clarification of terms such as "diggability" and "diggability studies". A certain ambiguity arises from the application of these terms to more recent studies, the natures of which, bear little resemblance to the earlier investigations that coined the original terminology. Essentially, the terminology is being applied to research that falls into two unique categories. Early investigations (Franklin, 1971, Kirsten, 1982, Muftuoglu, 1983 and Scoble 1984) applied the term to an assessment of the physical characteristics of the in situ or intact rock mass, and how they related to the selection of appropriate excavating equipment. The same terminology is currently being applied to more recent research in which the objective is to determine the actual performance of an excavator as it interacts with the broken rock mass (Williamson, 1983, Grant, 1983, Mol, 87 and Hendricks, 1988, 1989). These more recent investigations have focused on determining the effectiveness of ground preparation (ie. blasting) for the excavation process. Clearly, referring to both as "diggability studies" is inappropriate, since one only predicts digging conditions, typically in environments requiring little ground preparation. Quite opposite, one involves digging in an environment requiring extensive ground preparation - shovel instrumentation studies. A discussion of the prior research in each category follows.

7.1 GEOTECHNICALLY BASED PREDICTIONS OF DIGGABILITY FOR EQUIPMENT SELECTION PURPOSES.

Research by Franklin et al. (1971), proposed a rock quality classification system based on measurements of fracture (discontinuity) spacings and point load strengths. The system derived a diggability index for use in evaluating the amount of

ground preparation required, from which appropriate equipment and methods of excavation could be selected.

A comprehensive classification system for excavations in natural materials was proposed by Kirsten in 1982. The system is based entirely on the following fundamental properties of the in situ material, from which an "excavatability index" is calculated:

1. Strength of parent material	7. Block shape
2. In situ density	8. Block orientation
3. Degree of weathering	
4. Seismic velocity	9. Joint roughness
	10. Joint gouge
5. Block size	11. Joint separation
6. Shape of excavation relative to excavating equipment.	

The above grouping of physical properties have their effects considered and amalgamated into the following four main parameters:

1. Mass strength number.
2. block size number.
3. Relative ground structure number.
4. Joint strength number.

The excavatability index is equivalent to:

$$a_1 \times a_2 \times a_3 \times a_4$$

where a_1 etc. refers to the numerical ratings of the characteristics denoted above.

The classification system based on the excavatability index is again intended as an aid in determining / selecting appropriate equipment and excavating methods.

Work of a similar nature was undertaken by Muftuoglu and Scoble (1983), in their assessment of diggability in surface coal mines. Data on the physical characteristics of several mine benches was used to characterize the rock mass. Analysis of fundamental rock properties such as intact strength, weathering and joint and bedding spacing were used to define a diggability index for working mine benches. This was then related to digging and ground preparation requirements experienced by the operations. Again, the aim of the investigation was to match appropriate excavating equipment and ground preparation techniques with the observed/calculated rock mass characteristics.

A later study by the same authors (Scoble et al., 1984) focused again on the derivation of a diggability index for surface mine equipment selection. The investigation advanced their previous work in that it conducted field monitoring of equipment during the excavation process. The accrued equipment performance data suggested good correlation between the diggability index, suggestive of the amount of ground preparation required and excavator performance. This work is unique from the other work reported in this section in that it involved actual instrumentation of an hydraulic shovel (CAT 245 - 4 cubic meter bucket). Hydraulic pressures were recorded at the main hydraulic cylinders (ie. stick, bucket and boom) as it dug and were subsequently correlated with amount of ground preparation, block sizes, loading rates etc. In this regard the research can be regarded as transitional, between prior purely geotechnical investigations of rock mass characteristics and how they may influence equipment performance, and the more recent investigations (to be reported on in the next section), which have involved microprocessor based instrumentation of certain excavator performance parameters, to investigate their responses to varied characteristics of a fragmented rock mass.

The remainder of the work reported upon in this section is significant in that it provides methods by which observed and/or recorded rock mass characteristics, can be compared and ultimately correlated with shovel performance data.

7.2 ASSESSMENTS OF MUCKPILE DIGGABILITY IN RELATION TO BLAST EFFECTIVENESS THROUGH SHOVEL INSTRUMENTATION.

The instrumentation and monitoring of electric mining shovels represents a fairly recent area of research. Work by McDonnell Douglas Electronics (Keller, 1982), contracted by the United States Bureau of Mines (USBM), is credited with the earliest complete instrumentation of an electric mining shovel. This involved the development of a microprocessor based monitoring system to improve the utilization of electric mining shovels in the coal industry. The system, designed to provide maintenance alerts to the operator, and mine management with a data base on shovel production and maintenance, was deployed on a Bucyrus Erie 295B shovel used in overburden removal at a western American coal mine.

This work was concerned only with the development of shovel monitoring technology, not the manner in which monitored shovel responses could be related to variations in muckpile diggability.

The remainder of this section will report on the work to date involving the use of micro-processor based shovel instrumentation, to assess blast efficiency through measures of muckpile diggability. Prior investigations in this area have been instrumental in determining the scope and direction of the current research, which aims to advance the technology and state of knowledge obtained through these preliminary studies. The amount of previous work of direct relevance to the current research is limited to two projects undertaken in Australia by Williamson et al., 1983 and later by Mol et al., in 1987.

The author credits Williamson (1983) with the first research in the area of measuring blast efficiency through the performance monitoring of an electric mining shovel. The research involved instrumenting and recording the following shovel operating parameters:

- Crowd armature voltage.
- Crowd armature current.
- Swing armature voltage.
- Hoist brake relay.
- Crowd propel transfer relay.
- Dipper trip relay.

Originally, the hoist motor was instrumented to provide data on hoist armature voltage and current. However, the instrumented shovel models - P&H 1900 and 2100 are Magnetorque models, which employ a clutched eddy current power transferal system between the hoist and crowd motors. This means power is supplied between crowd and hoist differentially depending on power requirements of the respective motors ie. more power when digging is harder. Hoist responses were therefore not solely a product of digging, since power is not supplied uniformly at all times, and the effects of crowd motor power draws can not be eliminated. For these reasons hoist motor responses could not be considered in the analysis and their instrumentation was not continued. Recall that the current research involves the instrumentation of a P&H 2800XP, an Electrotorque controlled model with separate power and controls for hoist and crowd permitting their instrumentation and interpretation as valid indicators of digging conditions.

Productivity variables that are assumed to reflect diggability were recorded for later comparison against a diggability index, determined through an analysis of the instrumented shovel parameters. The production variables included, among others;

1. Total monitoring time
2. Actual operating time.
3. Truck waiting and spotting times.
4. Number of trucks loaded per monitored hour.
5. Number of trucks loaded per hour of actual operation.
6. Bucket fill factor as inferred by the ratio of minimum bucket loads per truck observed under operating conditions to the average number observed.
7. Dig utilization as defined by the ratio of the number of buckets hauled to total number of digging actions.

In addition, a subjective assessment of fragmentation was provided by the shovel operator for each monitored period.

The diggability index to which the above observations were to be related was derived by analyzing only the digging portions of the crowd voltage traces. An example of the motor traces obtained is given in Figure 7.2.1, according to the authors (Williamson, et al.) the top set of traces represent easy digging conditions, while the lower set are indicative of difficult digging.

The diggability index for each dig cycle is calculated based on the following equation:

$$\text{Diggability index} = \Sigma |\delta V|/A$$

where δV is change in voltage

$$\text{and } A \text{ is area under the signal trace} = \int_{t_1}^{t_2} V \, dt.$$

The calculated values of the voltage based index were found to vary between 1 and 10 and concentrating between 3 and 7. Values above 5 were deemed indicative of hard digging requiring excessive use of crowd force.

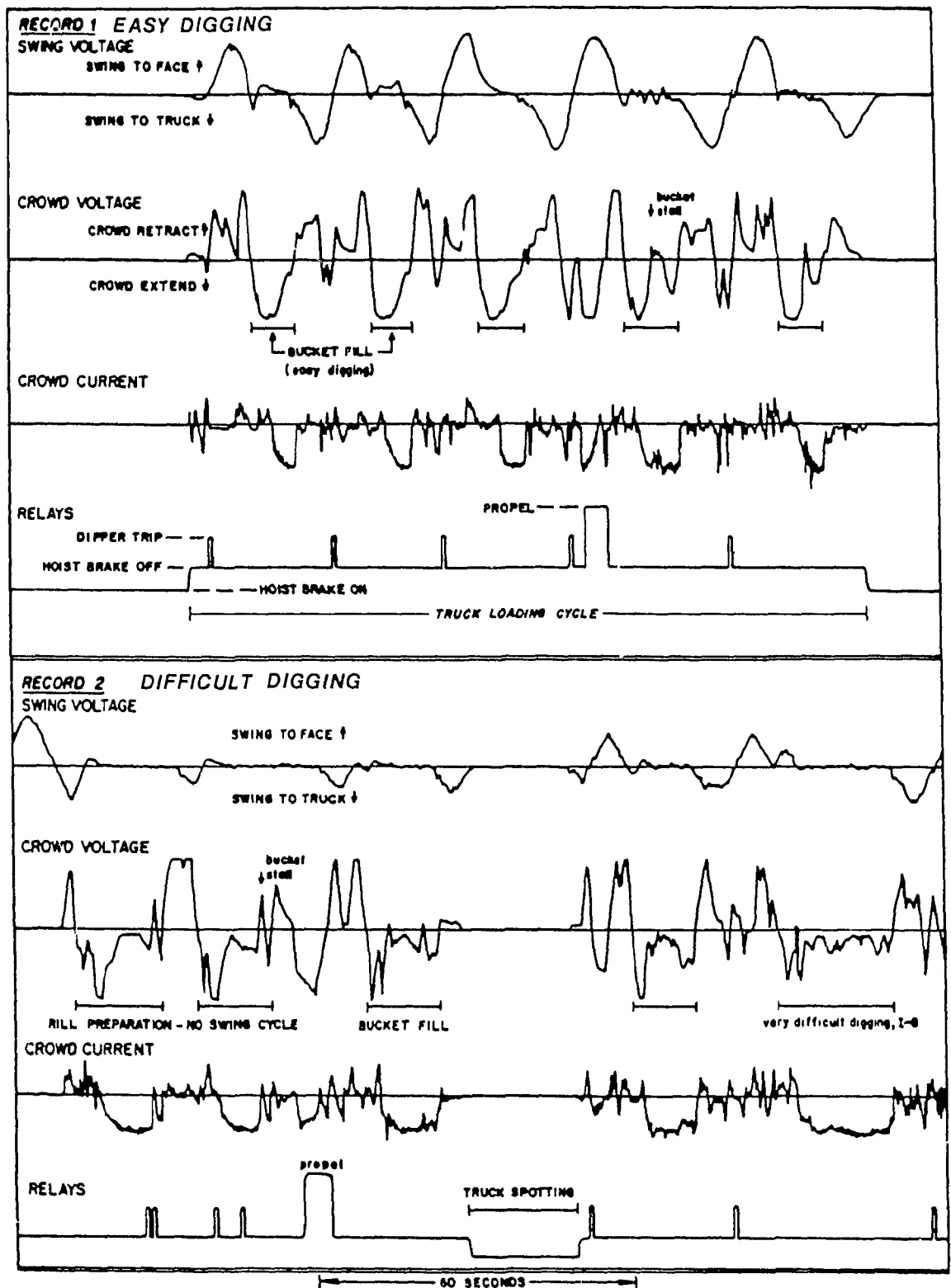


Figure 7.2.1 - Examples traces of monitored shovel performance (after Williamson et al, 1983).

The results of the study are presented in Table 7.2.1, which summarizes and illustrates the manner in which the various production variables, (observations of fragmentation, shovel and rock type), were correlated with calculated diggability indices for each of seven studies. Some of the principal conclusions that were drawn from the study include:

- The dig utilization ratio (number of buckets hauled to total number filled) is inferred to provide a good indication of muckpile looseness and face profile. The lower the ratio, the greater the amount of muckpile preparation required before loading. It is claimed that a low ratio coupled with a high diggability index is indicative of a tight muckpile.
- Higher diggability indices were associated with blasts in ore. It is suggested this is a consequence of lower explosive volumes as compared with blasts in waste, where a higher lump-fines ratio can be tolerated, resulting in less blocky ground.

The work undertaken by Williamson has indeed proven of benefit, several insights were gained into the potentials for micro-processor based shovel instrumentation strategies in the future. Further research by Williamson has however not been reported on since 1983, the technology appears to have remained in a state of infancy, not sufficiently refined to adequately validate the application of monitored shovel parameters to the issues of evaluating blast efficiencies. The work is considered to justify extension in order to account for deficiencies in the following areas:

1. The extent of monitored parameters is not sufficient, from a scientific stand point, to adequately define or characterize the behaviour of the shovel during the various components of the loading and dig cycles. The only instrumented parameter used in the determination of the diggability index was crowd voltage. It has been demonstrated in the literature by shovel manufactures

(Chapter 2), that the design of a cable shovel is such that the work and hence, productivity, is accomplished primarily through the hoisting action of the dipper. Crowd is important in that it assists the hoist action by maintaining a suitable depth of dipper penetration into the face or muckpile. Monitoring crowd alone is a misrepresentation of the true forces exerted by the dipper on the muckpile. Furthermore, it is erroneous to consider motor velocities (voltage) without due consideration to the generated torques or motor effort (current).

2. It is essential that the position of the dipper in the bank during the dig cycle be defined. It was demonstrated through an analysis of the preliminary data from the current research, that motor responses and dipper position (ie depth of penetration in the bank as a function of crowd) are intimately related. The amount of work the shovel motors accomplish, or the amount of material handled, on a pass of set distance through the bank is obviously a function of how far the dipper penetrates into the bank. It is possible therefore to record motor traces indicative of easy digging solely by maintaining shallow penetration in the muckpile. Clearly, motor performance data without dipper position data, is of limited "scientific" worth.

3. No objective and quantifiable measurements of muckpile characteristics have been made, only subjective assessments of fragmentation have been attempted. As reported on in section 6.2.5 of this document, this method is very inaccurate and can do more harm to an experiment than good. If monitored shovel parameters are to be confirmed as indicative of digging conditions, some aspect of the muckpile needs to be quantified and shown to correlate with the monitored shovel performance data.

TABLE 7.2.1

Production Variables and Diggability Indices
for a Series of Monitoring Trials. (Williamson 83).

Trial number	1	2	3	4	5	6	7
Blast type	waste	waste	waste	ore	ore	ore	waste
Shovel model	1900	1900	1900	1900	2100	2100	2100
Max. haul rate (trucks/hour)	13	13	13	15	19	19	15
Achieved rate (trucks/hour)	11.3	8.7	10.7	11.3	12.7	7.1	13.6
Load rate (trucks/hour)	14.5	12.7	16.2	17.2	17.5	9.4	16.5
Time utilization (%)	92.6	77.3	75.0	67.6	75.1	94.4	82.0
Actual operating time (%)	78.3	68.4	65.9	65.8	72.6	75.6	82.0
Truck waiting / spotting time(%)	7.4	22.7	25.0	32.4	24.9	5.6	18.0
Downtime (%)	14.3	8.8	9.1	1.8	2.5	18.8	0.0
Repositioning time (%)	3.2	6.9	2.2	1.0	1.8	4.9	N.A.
Average bucket fill factor	0.85	0.85	0.89	0.83	0.79	0.67	1.0
Average bucket fill time (sec)	10.0	9.8	9.8	11.9	12.3	11.9	10.1
Dig utilization ratio	0.88	0.82	0.82	0.98	0.97	0.62	0.96
Diggability index mean	4.3	4.7	3.7	5.6	4.3	5.6	3.0
std. dev.	1.0	3.0	1.0	1.4	1.2	1.8	0.8

COMMENTS FROM PRODUCTION PERSONNEL:

- Trial
1. Bucket malfunction slowing production, relatively easy digging with a few floaters
 2. Moderately good fragmentation but low muckpile on a narrow bench and vertical face being finalised.
 3. Good fragmentation but poor scheduling
 4. High grade well fragmented but with hard toe, rilling easily.
 5. Moderately easy digging, hard toe, slow operator.
 6. High grade ore moderately difficult digging, cleaning up at limit of blast.
 7. Very fine fragmentation, high swell factor.

4. The effects of both the blast design and local bench geology should be accounted for in a more detailed manner. Williamson has not attempted to relate the performance of the shovel with the actual design aspects of separate blasts. Studies have been conducted in blasts for both waste and ore, and associated differences in performance in each blast type have been reported. However, if changes were instituted in either of these blast types would changes be observed in shovel performance?

5. There were all together too many variables in the study to confirm wether or not the shovel is responding primarily to muckpile conditions. Of the seven trials reported on, none were under equivalent conditions. The following significant differences were reported from one trial to another; bucket malfunctions during study, digging a low muckpile on a narrow bench, poor scheduling, hard toes, slow operator, cleaning the limit of a blast, etc. Furthermore, different model shovels, equipped with different size and designs of dippers were employed. It was observed during the course of the current research, that even two identical model shovels can have performance standards unique to each.

In summary, the study although considered to be a good first attempt at the problem, has failed to provide control over the several variables that could easily confound attempts to correlate shovel performance solely with muckpile conditions as an assumed function of blast efficiency. The current research, has provided the detail and experimental control required to assess the accuracy of post blast evaluations based on monitored shovel performance parameters. The philosophy of the current research has been one machine, one environment, one isolated change in blast design parameter at a time, one set of operators whose performance relative to one another is known, and at least one quantifiable aspect of muckpile conditions against which to relate monitored records of shovel performance.

Work reported on by Mol et al., 1987, employed shovel monitoring techniques similar to those employed by Williamson et al. in 1983. The work was undertaken as part of a larger project to investigate rock fragmentation by drilling and blasting in both coal and metal mines in Australia. Three monitoring trials were carried out on P&H model 2300 electric mining shovels (an Electrotorque Controlled model) used in overburden removal at an open cut coal mine. Again, the only shovel operating parameters monitored were:

1. Crowd voltage.
2. Crowd current.
3. Hoist voltage.
4. Hoist current.
5. Swing voltage.
6. Dipper trip relay.
7. Crowd/propel transfer relay.

Figure 7.2.2 illustrates the recorded traces of these monitored signals. The study strove to develop diggability indices based in part on the analysis of the crowd motor data. Again as per Williamson, the crowd motor response (signal) is given primary consideration and only during the dig portion of the cycle. The component of the diggability index based on motor performance data is determined in the following manner after Mol:

- a). If the dig time is less than a lower set value (e.g. 15 seconds), the ratio of the total vertical crowd voltage signal length to the area under this signal is used to derive the index.
- b). If the dig time is between the lower and upper set values (e.g. greater than 15 but less than 25 seconds), the total vertical signal lengths, per second, of crowd voltage and current, or the approach (a) above, whichever is greater, is

used to determine the index.

c). If the dig time is greater than the upper set value, an index value is calculated based on the dig time and this is then compared with the other two obtained by the above procedures; the greatest is chosen as the index for that cycle.

The above criteria indicate that the digging index is viewed as a function of both dig time and dig effort. Where dig time is shortest the index is solely a product of crowd motor voltage (RPM). If an intermediary time is recorded then both crowd speed and torque are considered. Finally if a very long dig time is recorded then the index is based only on dig time.

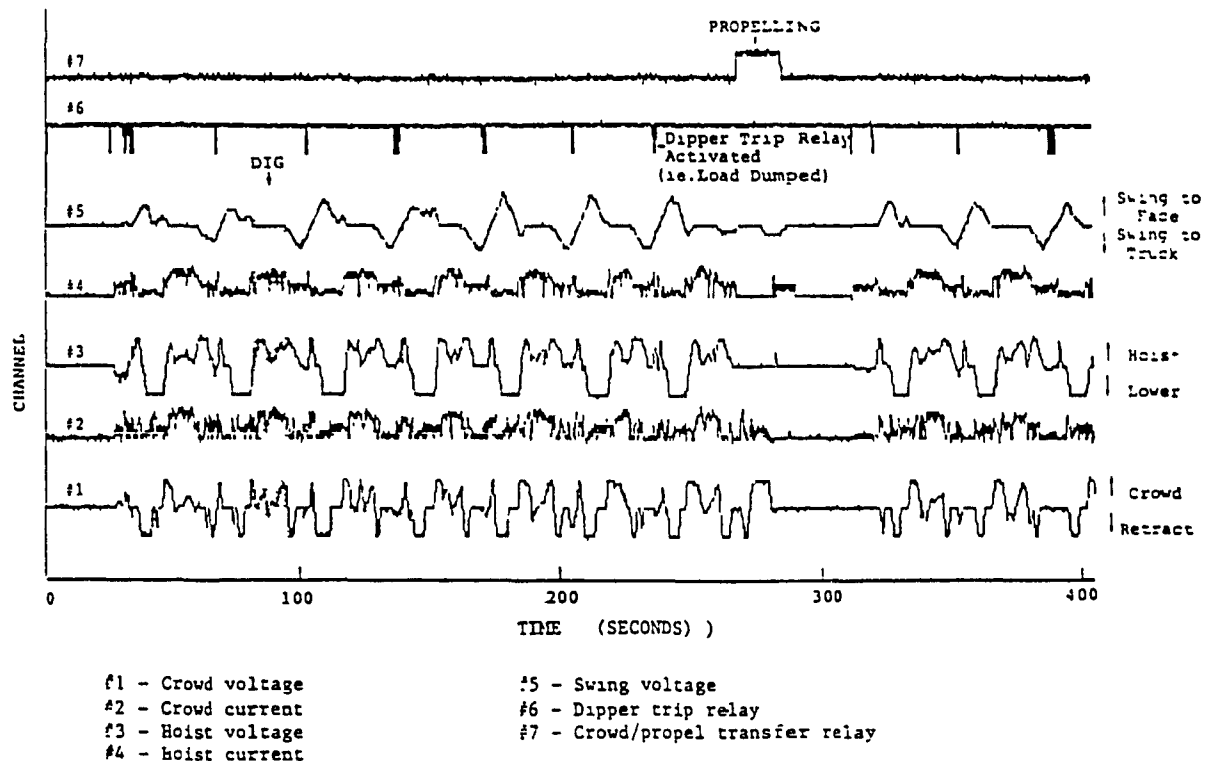


Figure 7.2.2 - Example traces of monitored shovel performance parameters (after Mol et al, 1987).

In addition, the diggability index determined through an analysis of crowd motor data is considered in conjunction with an estimate of face preparation effort, based on an analysis of hoist and crowd motor current traces (Figure 7.2.3). Face preparation is regarded as all digging activity not associated with actual loading of material. The effort involved, is determined from the total vertical signal lengths, per second of crowd and hoist currents. The value lies between 0 and 10, representative of digging conditions outlined by the authors as per the table below.

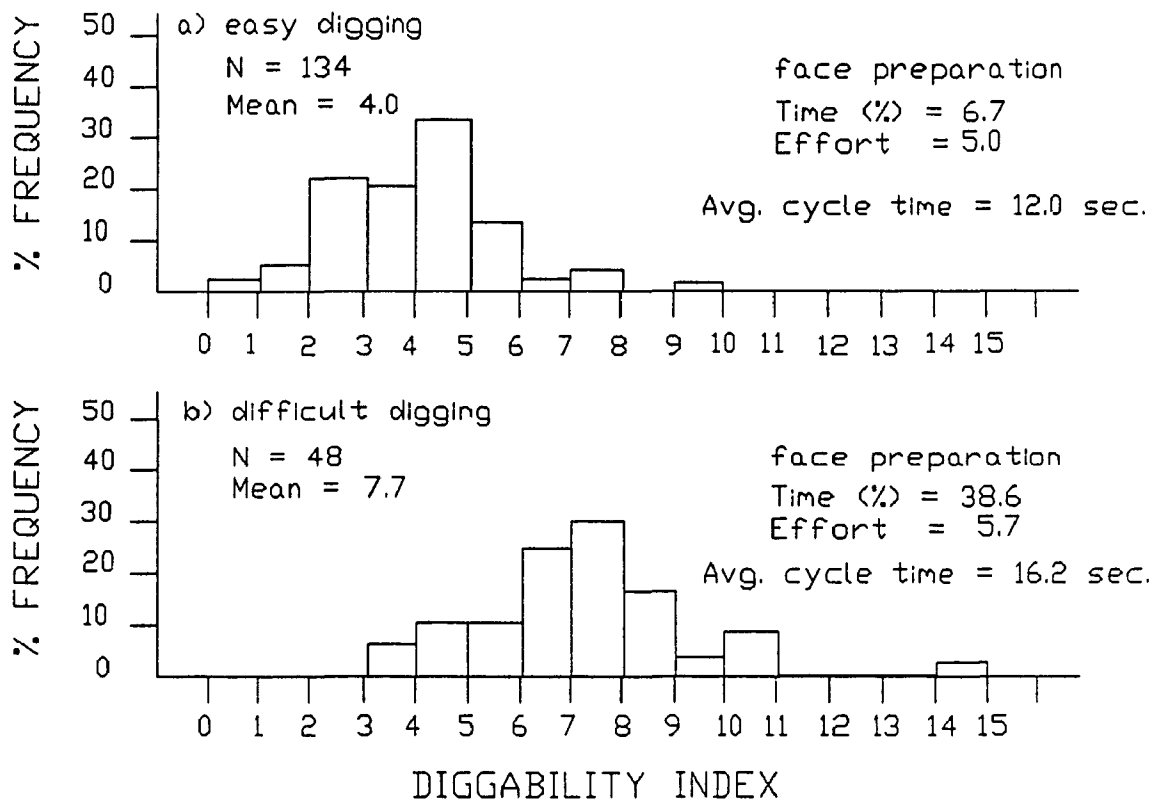


Figure 7.2.3 - Frequency distributions of diggability index values and face preparation for a) easy and b) difficult digging (After Mol et al., 1987).

**CLASSIFICATION OF DIGGABILITY INDEX VALUES
TO REPRESENT DIFFERENT DIGGING CONDITIONS.**

<u>INDEX VALUE</u>	<u>DIGGING CONDITION</u>
less than 1.0	extremely easy
1.0 - 2.0	very easy
2.0 - 4.0	easy
4.0 - 6.0	normal
6.0 - 8.0	difficult
8.0 - 9.0	very difficult
over 9.0	extremely difficult

The work by Mol et al., mirrors several of the deficiencies associated with the preliminary work by Williamson. Most importantly, it fails to incorporate valid traces of hoist motor performance in the determination of the actual diggability index. Hoist data is only considered during non-loading activities, but clearly every loading action is to a large part a function of the hoist action. Again no quantified assessment of muckpile conditions were reported and no attempt was made to relate shovel performance to blast design criteria.

In addition to the two investigations reported on in this section two other studies of a similar nature are worthy of mention; works by Scheck et al., 1982 and Grant et al., 1983.

In 1982 a project sponsored by the United States Bureau Mines was undertaken by Scheck et al., to develop technology suited to automated blast hole logging and design through drill performance monitoring. An analysis of drill monitored performance parameters (from a B.E. 60R) was used to improve blast designs in a mid-western surface coal operation. It was intended to assess the effectiveness of the new blast designs by monitoring the performance of the operations dragline as it excavated the blasted overburden. The dragline was equipped with a monitor to measure energy consumption, cycle times and other factors that may be related to fragmentation of the overburden. Unfortunately, a failure of the drill monitoring unit delayed the integration of monitored data from both pieces of equipment and an "integrated study" was never realised.

The work by Grant et al., 1983 is of relevance in that it involved the instrumentation and monitoring of an LHD as part of a larger project to evaluate open stope blast performance at Mount Isa Mines, Australia. The study employed a visual assessment of average rill particle size and the monitoring of LHD cycle times. The data was then analyzed with the intent of

establishing a relationship between loading rate and fragmentation size. Examples of the performance data are illustrated in Figure 7.2.4. Attempts to correlate this data with observed average fragment sizes were unsuccessful as evidenced by the plot in Figure 7.2.5. Loading rates were only significantly affected when the LHD encountered an oversized boulder.

Load-Haul-Dump Cycle and Digging Times

STOPE	CALCULATED FROM WEEKLY AVERAGES					
	TRAMMING DISTANCE (ONE WAY) (Metres)		LHD CYCLE TIME (ROUND TRIP) (Minutes)		LHD MUCKING TIME (TIME SPENT) (Seconds)	
	AVE	STD DEV	AVE	STD DEV	AVE	STD DEV
<u>H710</u> 3 Drawpoints Tipping into I71 Orepasses	62	12	1.53	0.32	16.4	1.85
<u>Q561</u> 3 Drawpoints Tipping into Q55N Orepass	52	18	0.99	0.20	10.9	3.4
<u>N461</u> 2 Drawpoints Tipping into O46 & N46 Orepasses	135	31	1.91	0.44	9.9	2.2
<u>O480</u> 5 Drawpoints Tipping into O48, P48 & Q48 Orepasses	79	27	1.34	0.32	11.2	2.4

Figure 7.2.4 - Data from LHD monitoring (Grant et al, 1983)

Data obtained from the instrumentation of the LHD (Figure 7.2.6) was employed to supplement the above data, and confirm that little correlation exists between fragment size and shape, and the mucking time of an LHD. It was presumed that the operator induced bucket oscillations (depicted in the performance traces of Figure 7.2.6) mask most of the effects fragment size and shape may have on the mucking operation. The

LHD MUCKING PERFORMANCE

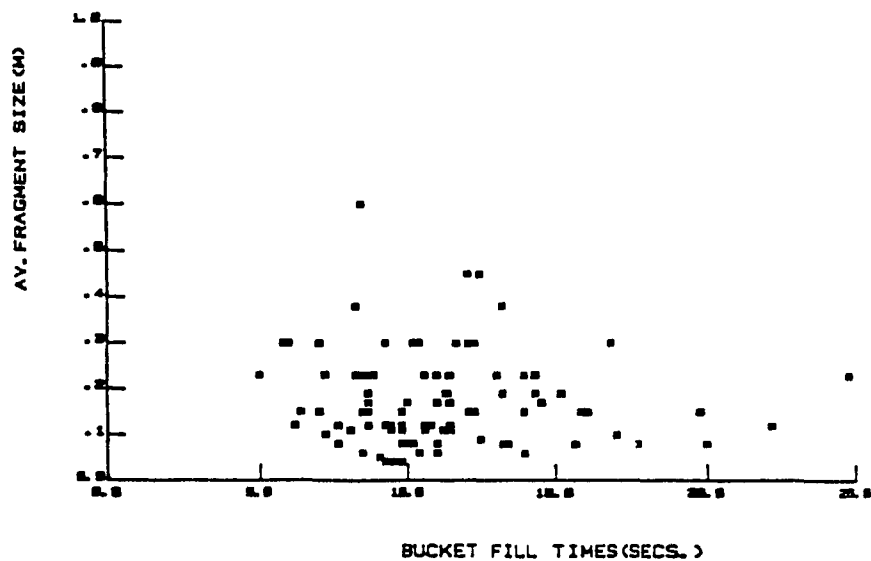


Figure 7.2.5 - Relation between LHD fill time and fragmentation (Grant et al, 1983).

oscillations, act to fluidize the layer of rock in front of the buckets teeth allowing easy penetration into the muck.

It is in no way suggested that the conclusions drawn from a preliminary investigation of LHD performance monitoring, can be extrapolated to infer the behaviour of a mining shovel. The operation of the two pieces of equipment are in very few ways related to one another. The significance of the underground study is that it too strove to establish a means of relating (through micro-processor based instrumentation) the behaviour/performance of an operation's primary loading equipment to the effectiveness of the blasting practice.

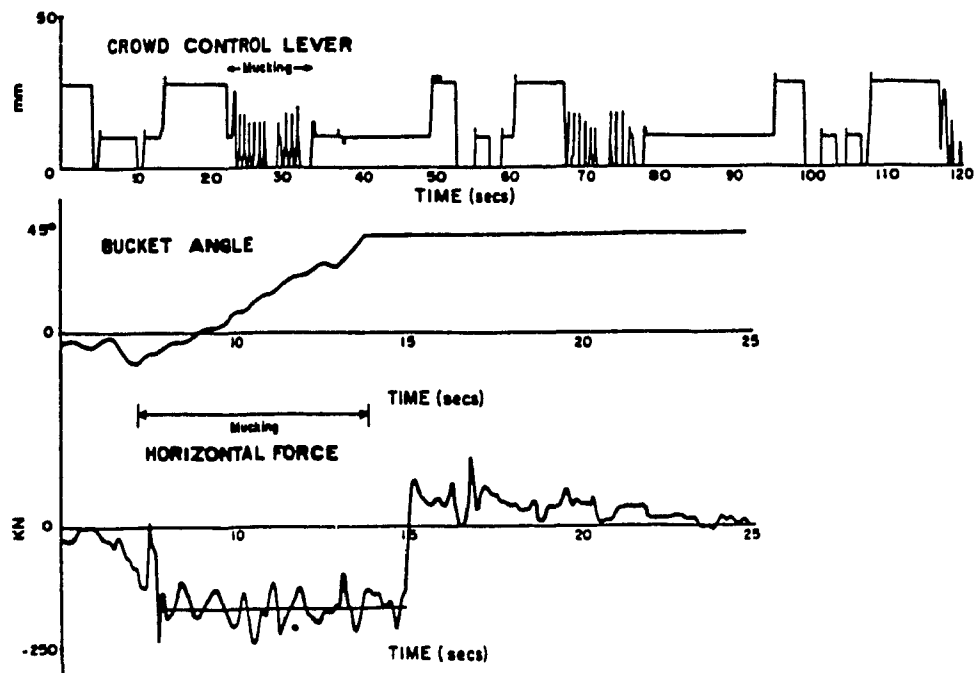


Figure 7.2.6 - Example of monitored LHD performance parameters
(Grant et al, 1983).

7.3 SUMMARY.

It is the present opinion, that the experimental design of prior works involving shovel instrumentation, have suffered from limitations that hindered the development of verifiable relationships between monitored shovel responses and actual variations in muckpile diggability. The current research has sought to alleviate and/or control and account for, several of the experimental variables that characterized prior work (i.e. variations in operating practice, different blasts, geologic environments and different shovel models etc.). A broader scope of shovel monitored data, including dipper trajectory, has been collected from one shovel while excavating in a single mine

environment, in which elements of bench geology and blast design have been extensively explored. The following chapter will address the manner in which the current data has been analyzed towards isolating the above experimental variables in establishing the nature of shovel - muckpile interactions.

6.0 POST BLAST EVALUATION.

6.1 INTRODUCTION.

It was required that some quantified measure of muckpile conditions be available, against which to compare the monitored performance of the shovel as it excavated each of the test blasts.

The preceding chapter addressed those aspects of a blast that are known to impart a significant influence over blast results (efficiency). The present chapter will briefly discuss several methods commonly employed in surface mining, to rate or assess the efficiency of blasting; post blast evaluation techniques. Such techniques usually focus on fragmentation, as it affects several phases of the mining operation, and hence, overall mining costs.

The post blast assessment technique adopted for the current research is the photographic method, in which photographs of muckpile fragmentation have been processed by a semi-automated image analyzing system, to calculate daily fragmentation size distributions. A detailed explanation of the technique, and the results obtained, follows a review of other methods considered.

6.2 METHODS OF BLAST EVALUATION CONSIDERED.

6.2.1 High Speed Photography.

High speed photography provides a relatively simple and direct diagnostic method of assessing blast performance, design and/or for confirming results predicted by various computer blasting models (Chiappetta, 83, Chung, 75, Lang, 72, Winzer, 79 and Floyd 87). The method employs a camera operating at a rate of between 500 and 1000 frames per second, capable of filming short lived blast events. At such speeds it is possible to record and analyze

Once shovel operation began, five sources of data could be collected during the four hour monitoring period aboard the shovel;

1. General Electric sensor data (position and motor data).
2. General Electric production data (cycle times, weights etc.).
3. Polycorder data (hoist and crowd motor data).
4. Video tape records of the shovel activity during the monitored periods.
5. Photographic records of fragmentation and structure of bench face.

As explained in section 3.3, operating the G.E. monitor in a capacity to collect sensor data prohibits the collection of all other production data. The most significant consequence of this is that no data on the weights of suspended dipper loads is recorded, for analysis in conjunction with the recorded traces of shovel performance parameters. In order to compensate for this, it was elected to occasionally run the Polycorder while the G.E. monitor was in a production data gathering mode. In this manner, the motor traces recorded by the Polycorder could be related with the records of suspended weights for each recorded cycle. An analysis of this data has helped to establish the extent of correlation between the weight of the material handled per pass of the dipper and the associated motor performance traces for the dig cycle. Appendix F lists all of the monitored studies undertaken during each day of the study, in each of the blasts evaluated. Also indicated in Appendix F are the times at which the individual studies started, as well as the instrumentation employed in each study.

The video-camera played a critical role in that it provided a means of "reliving" the entire monitored period during data analysis. No written record can do justice to the intricacies of shovel operation during a monitored interval. Therefore, in addition to their use in determining shovel cycle times, the video tapes have been of substantial aid in resolving the actions

of the shovel for erroneous cycle traces recorded by both the G.E. monitor and Polycorder.

The video-camera was turned on at the onset of the first study of the day, and was left running until the two hour tape employed was exhausted. Normally, five studies, employing various combinations of the G.E. monitor and Polycorder were completed in this two hour period. In this manner a continuous record of shovel operation was maintained. The video records (a total of 36 hours) have been reviewed during the course of a comprehensive manual cycle time study.

Finally, the last source of data was the photographic record of fragmentation. Photographs were taken when conditions warranted (e.g. when coarse rill from the collared portion of the bench was introduced into the muckpile) during individual monitored studies. This provided a record of the size distribution of the material being handled by the shovel for each set of shovel performance traces. In Appendix F, the last two columns record the number of photographs taken during each study as well as what was photographed, e.g. normal fragmentation or coarse, recently rilled material.

8.3 ANALYSIS OF SHOVEL PERFORMANCE DATA.

This section will address the results and approach used in the interpretation of the collective data from the integrated bench study. Data analysis has focused on gaining an understanding of the information conveyed through the monitored performance traces, recorded by the G.E. monitor and the Polycorder data-logger. Supplemental analysis of the production data associated with the performance traces has been undertaken in an effort to corroborate these findings. Results indicate good correlation exists between the recorded micro-processor based performance parameters and associated shovel productivity. On days characterized by difficult digging conditions, as defined by

an analysis of the shovel performance parameters, productivity was correspondingly lower.

The remainder of this section will discuss the means employed in the analysis of the G.E. sensor data, Polycorder motor performance data, G.E. production data and time studies based on the video-taped records.

8.3.1 General Electric Shovel Sensor Data.

Examples of the shovel performance traces collected by the G.E. monitor, while operating in a sensor data collecting mode, are presented in Figure 8.3.1. Seven signals are presented here, the first five of which were recorded by the General Electric monitor. The sixth and seventh signals; crowd armature voltage and current, were monitored by the Polycorder data-logger. A discussion of the Polycorder related data is presented in a following section. The G.E. monitored signals are all time based, and recorded at a sampling interval of 100 ms, they are from bottom to top of the graph:

1. Hoist rope position (feet).

This signal, recorded by a position transducer on the hoist drum, indicates the length (in feet) of the hoist rope from the shovel point sheeves to the dipper attachment point on the crowd arm.

2. Crowd arm extension (feet).

This signal is recorded by a position transducer on the crowd drive motor. The signal indicates the amount of crowd (feet) applied for each dig cycle.

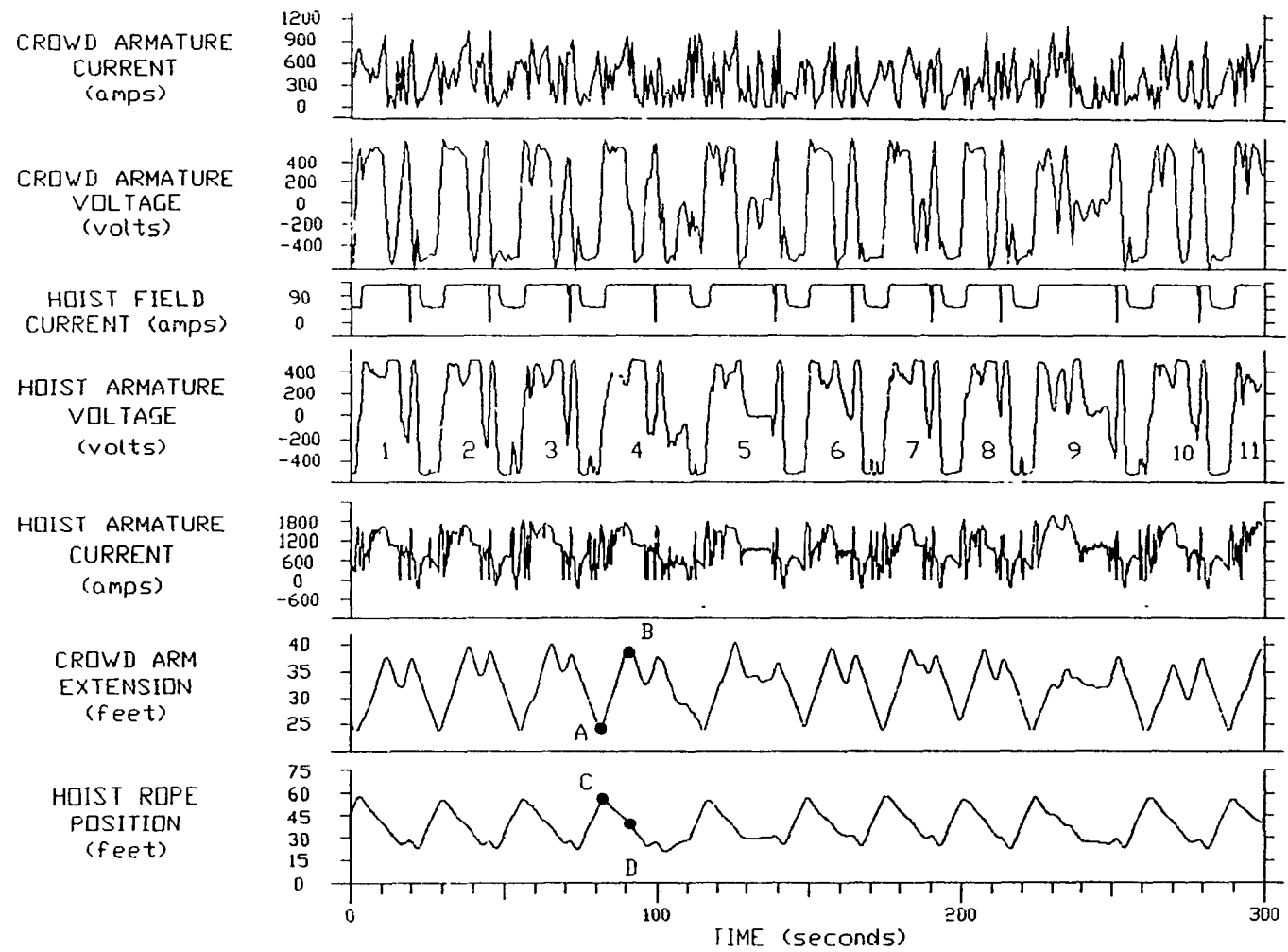
3. Hoist armature current (amps).

4. Hoist armature voltage (volts).

5. Hoist Field Current (amps) and dipper trip relay.

The hoist field current signal is monitored to define the intervals during which the shovel is operating in a weak, as opposed to a strong field mode. Weak mode operation is indicated on the trace of this signal by an abrupt, sustained flat trough.

Figure 8.3.1 - Example of monitored performance parameters recorded in easy digging conditions.



In this mode, the hoist motor is engaged in the controlled gravity lowering of the dipper. Activation of the dipper trip relay is indicated on this signal by the abrupt downward "spikes", seen to occur a few seconds in advance of the onset of weak field mode operation. The activation of the dipper trip relay indicates the dumping of load into a waiting haul truck.

Figure 8.3.1 relates to data recorded from the actions of the shovel for ten complete loading cycles as recorded over a five minute (300 second) period. It is to be noted, that a complete loading cycle refers to the digging, swinging and dumping elements for each bucket load. Individual loading cycles may be considered to begin at the end of the troughs seen in the hoist voltage signal, at which point, voltage rapidly increases as the dipper enters and commences to travel through the muckpile. This marks the beginning of the dig portion of the cycle, during which the dipper is actually loading the fragmented material. For the purposes of the current research, only the digging element of complete loading cycles was analyzed. The duration and nature of the digging event can be accurately determined and isolated from the remaining cycle elements (swing and dump) through an inspection of the first and second signals; hoist rope position and crowd arm extension. In combination these signals define the precise location of the dipper at all times and hence, the trajectory of the dipper through the muckpile during the dig cycle. Referring, for example to cycle number four in Figure 8.3.1, the interval from point "A" to point "B" on the trace of crowd arm extension defines the dig cycle, point "A" represents maximum retraction of the crowd arm and corresponds with point "C" on the hoist rope position trace, which represents the point of maximum extension of the hoist rope. The geometry of the shovel at this point is as illustrated in Figure 8.3.2, for the beginning of the dig cycle i.e. the dipper is at ground level near the toe of the muckpile. The dipper is actually a few feet in front of the muckpile, suspended just behind the shovel boom point. Point "B" defines the point of maximum crowd

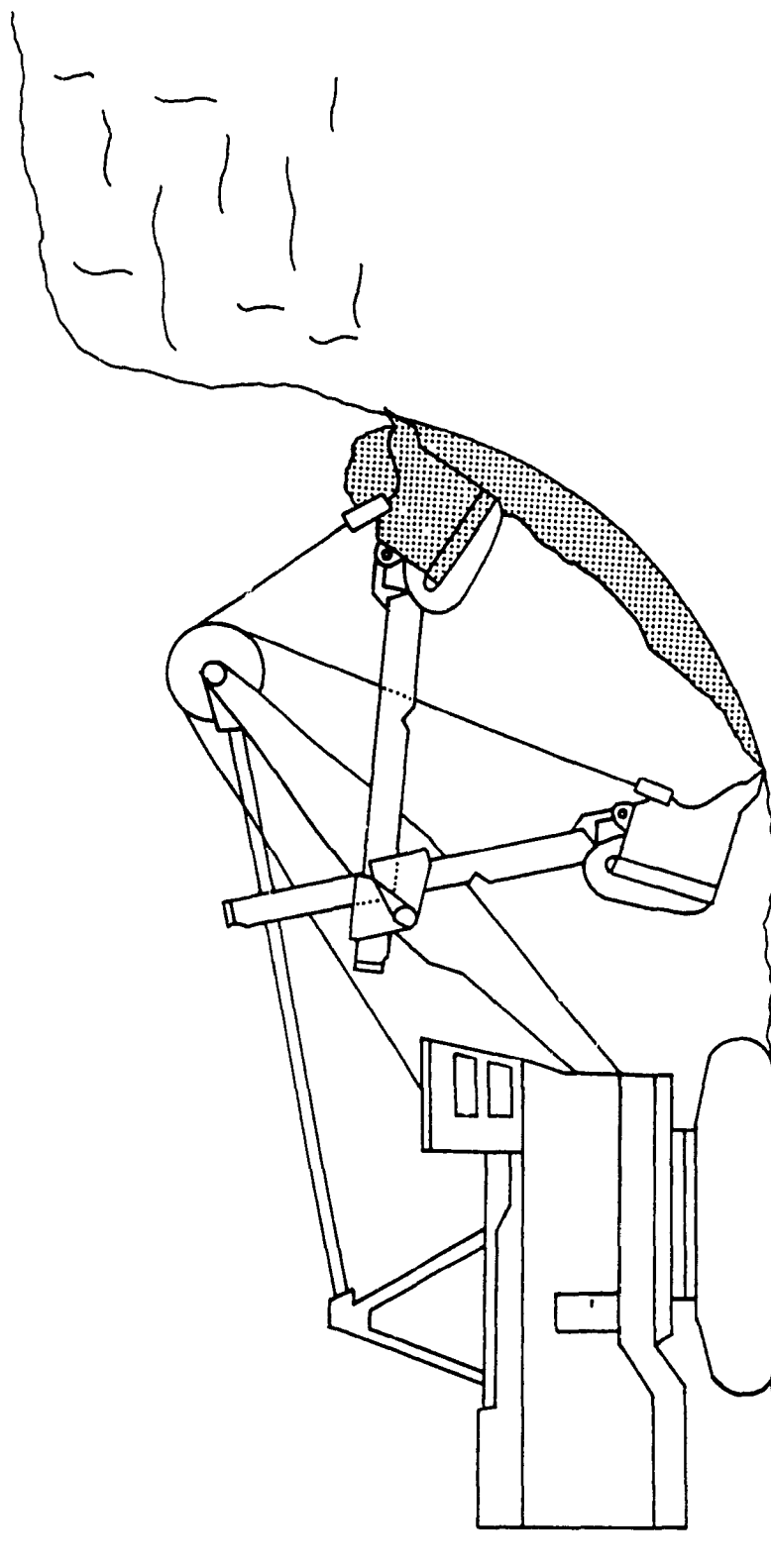


Figure 8.3.2 - Shovel geometry at start and end of dig cycle.

(thrust) into the muckpile and hence the end of the dig cycle, beyond which the crowd arm is retracting i.e. pulling the dipper out of the muckpile synchronous with the onset of swinging to a truck. The geometry of the shovel at this point is again given in Figure 8.3.2. Point "D" on the hoist rope position trace represents the point of maximum hoist rope travel associated with the dig cycle. Hoisting continues beyond a position equivalent to point "B" during the swing cycle in order to sufficiently elevate the dipper to a level which will clear the haulage truck's box. This point is indicated on the hoist rope position trace by a small plateau and is synchronous with the dumping of the load. The small secondary peak on the hoist rope trace for each cycle represents the added hoist required to lift the tripped dipper, i.e. its bottom open and swinging freely, clear of the haul truck's box.

Prior investigations (Williamson et al., 1983 and Mol et al., 1987) have relied on swing voltage traces to define the dig portion of the cycle. When the swing voltage signal is static the shovel is assumed to be engaged in digging activity. Figure 8.3.3 illustrates the application of this approach to sample data from the current research. It is felt that the method of isolating the dig cycle based on hoist rope position is more realistic in that it relates to actual dipper travel through the muckpile. Therefore, swing voltages, although monitored, will not be considered in the present data analysis.

8.3.2 Signal Interpretation - G.E. Data.

Having isolated the digging component of each cycle, it is possible to examine some of the characteristics of the monitored parameters for both easy and difficult digging conditions.

Easy Digging Conditions.

The data of Figure 8.3.1 is illustrative of easy digging conditions. Attention is drawn first to the responses of the hoist motor during the dig cycle. The particulars of hoist motor

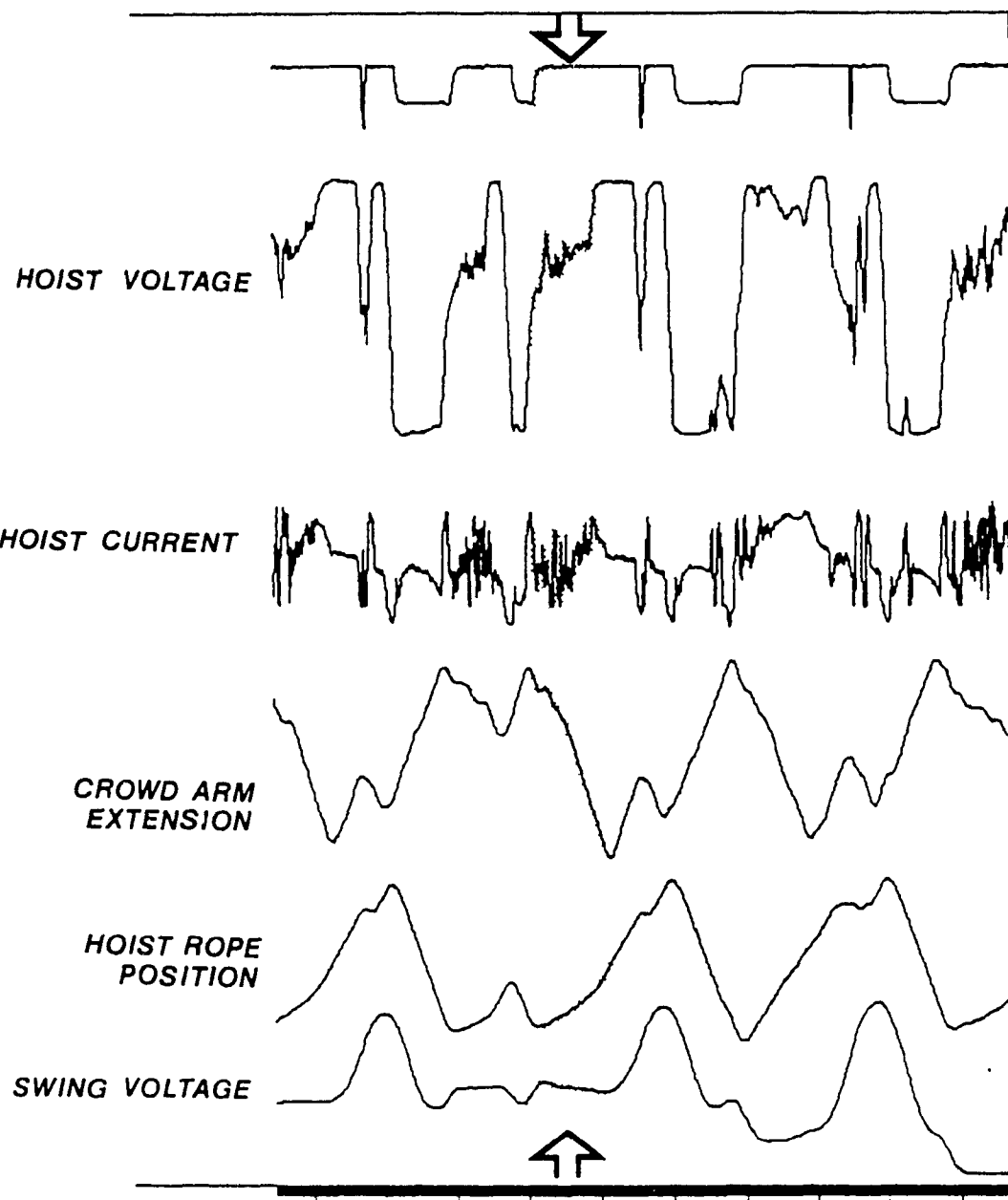


Figure 8.3.3 - Isolation of dig cycle based on swing voltage.

behaviour associated with easy digging conditions are displayed in Figure 8.3.4, in which progressively magnified views of the signals associated with cycle number two in Figure 8.3.1 are shown. In easy digging conditions hoist voltage remains at near

maximum values throughout the dig cycle, representing high hoist motor speeds (hoist drum rpm). Typically, the hoist voltage values for dig cycles recorded in easy digging conditions, will exhibit a "plateau and valley" type trace. The "plateau" that characterizes the first half of the dig cycle results from good, loose digging at the toe of the muckpile. The "valley" that follows, is a consequence of the added effort required to pull the increasingly full dipper up through the muckpile. As Figure 8.3.4 illustrates, the component of the dig cycle associated with the drop in hoist armature voltage (valley) corresponds to the deepest point of dipper penetration. The valley is therefore, representative of the most productive component of the dig cycle, in easy digging conditions. The hoist rope position and hoist armature current traces, have been superimposed on the hoist armature voltage trace in Figure 8.3.5 to corroborate the preceding observations. Note the shallow gradient along the component of the hoist rope position trace that coincides with the valley in the hoist voltage trace.

The hoist current signal represents the torque across the hoist motor armature, or bail pull, as induced by dipper travel through the muckpile. In easy digging conditions this signal displays an initial surge upon dipper advance towards the muckpile (ie. a sudden load on the motor at the onset of hoisting), following which, the signal drops rapidly and builds gradually towards a maximum value at the end of the dig cycle. The surge is not a consequence of initial muckpile penetration. Rather, it results from a return to hoist motion following the controlled gravity lowering of the empty dipper. As mentioned, the dig cycle actually begins with the dipper a few feet in advance of the muckpile's toe. Low hoist current values recorded during the early portions of the dig cycle reflect the favourable characteristics of the muckpile's toe (loose, well fragmented).

Figure 8.3.4 - Particulars of hoist motor response in easy digging conditions.

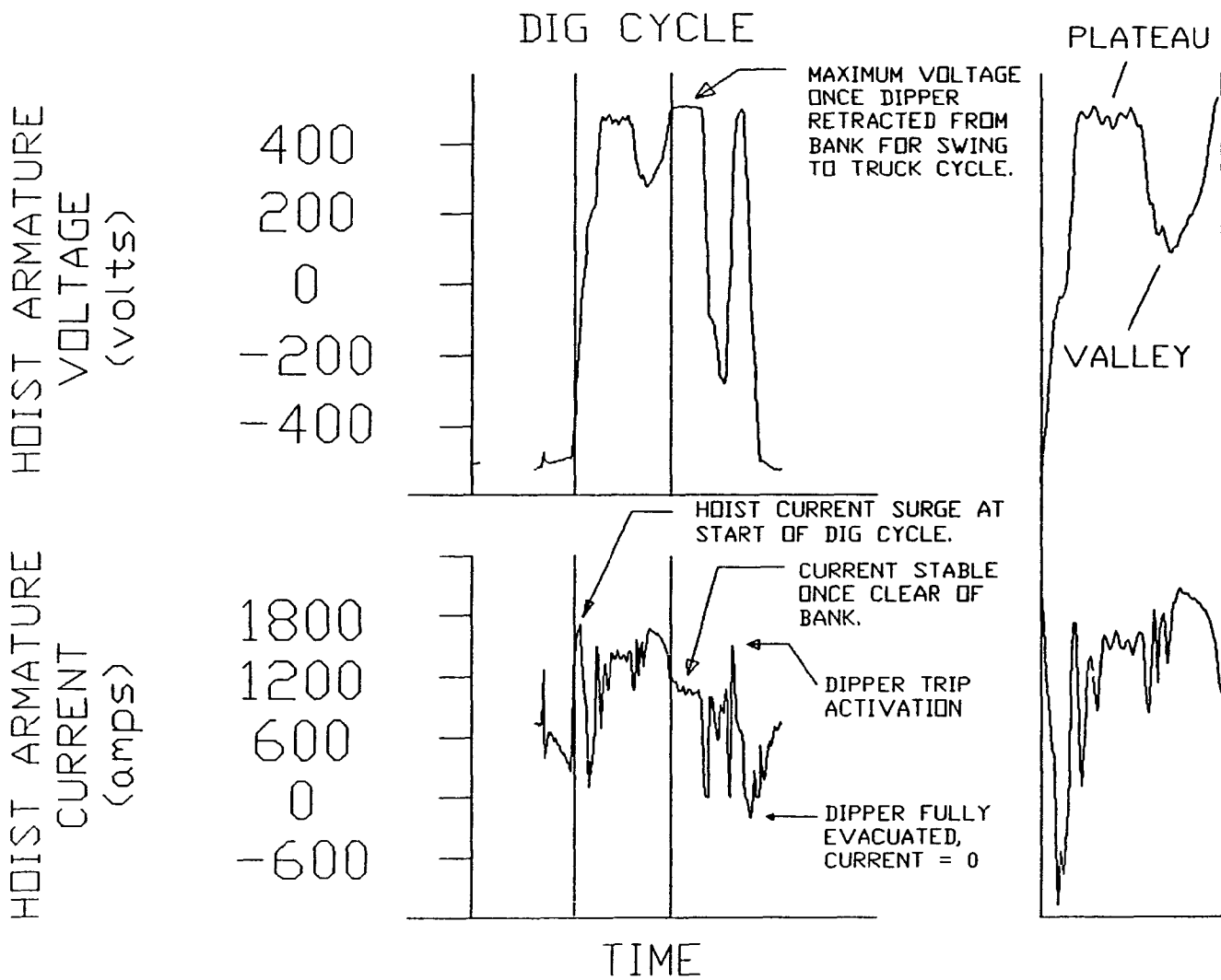
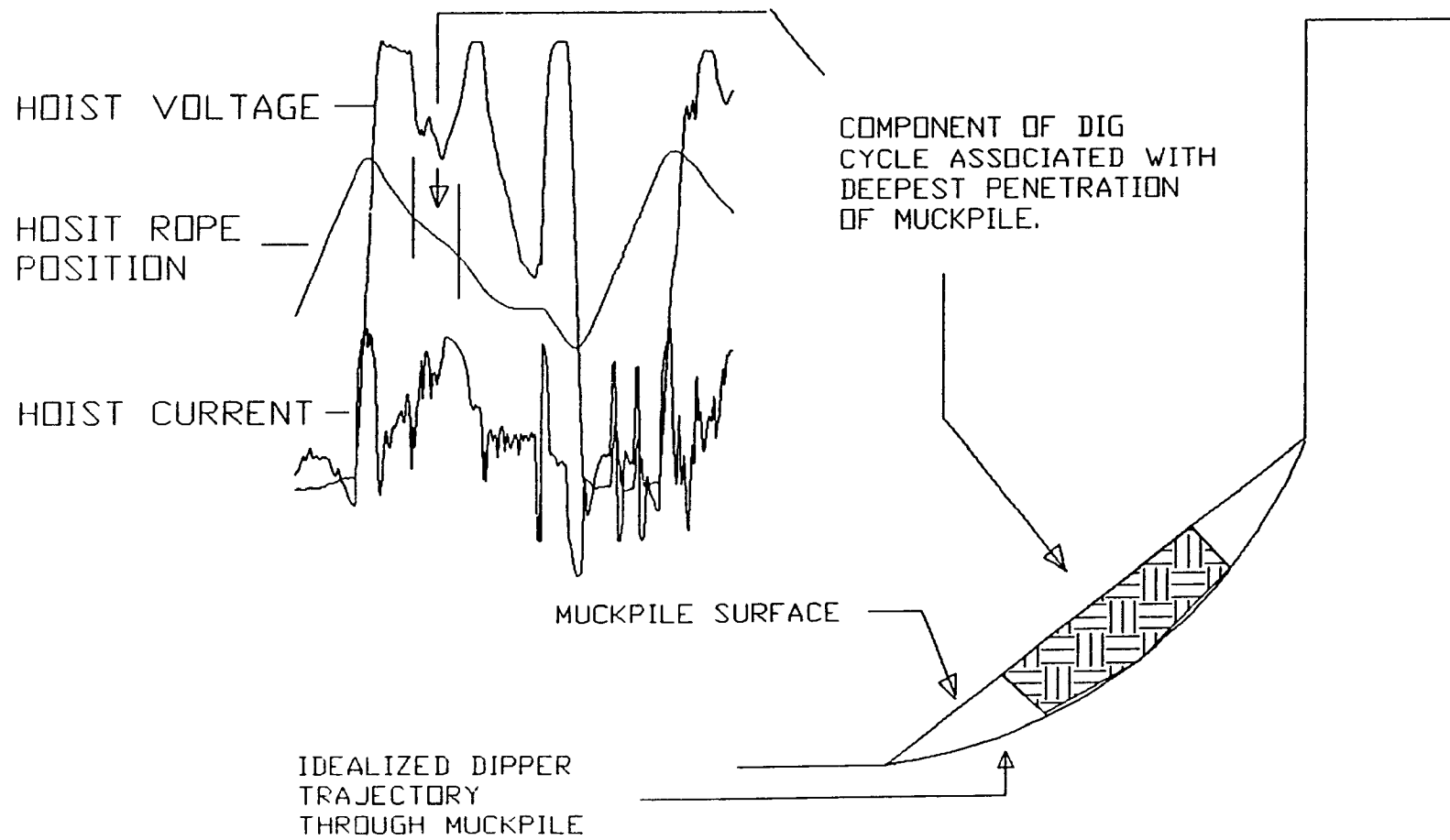


Figure 8.3.5 - Particulars of dig cycle for easy digging.



The two position signals, hoist rope position and crowd arm extension, provide added insight into the response of the shovel to digging conditions. In easy digging, the gradients of these signals during the dig cycle are typically very steep, smooth and continuous. This again reflects the relative ease and speed with which the dipper can negotiate its trajectory through the muckpile.

Difficult Digging Conditions.

Figure 8.3.6 illustrates the responses of the monitored performance parameters in difficult digging conditions. At the time of monitoring, the shovel was excavating a portion of a blast in which poor breakage at the toe had occurred. There are only seven loading cycles depicted in Figure 8.3.6, numbered 1,3,7,9,10,11 and 12. The remaining "apparent cycles", numbered 2,4,5,6 and 8 are episodes of ground preparation required to break or loosen poorly fragmented ground in advance of a genuine loading cycle. Note that the hoist field current signal reveals no dipper trip activation, i.e. dumping of a load, associated with these "apparent cycles". The following observations have been made regarding the responses of the monitored motor parameters in difficult digging conditions.

The values of the hoist voltage signal during the dig cycle are very low, and by comparison with easy digging conditions, considerably more erratic, indicating slow and difficult dipper advance. The hoist current signal displays no stabilization and gradual increase following initial muckpile penetration, as was observed in easy digging conditions. The particulars of hoist motor behaviour associated with difficult digging conditions are displayed in Figure 8.3.7, in which progressively magnified views of the signals associated with "load" cycle number three in Figure 8.3.6 are shown.

Upon closer inspection, the "ragged" hoist current signal during the dig cycle, is seen to result from a series of spikes,

Figure 8.3.6 - Example of monitored performance parameters recorded in difficult digging conditions.

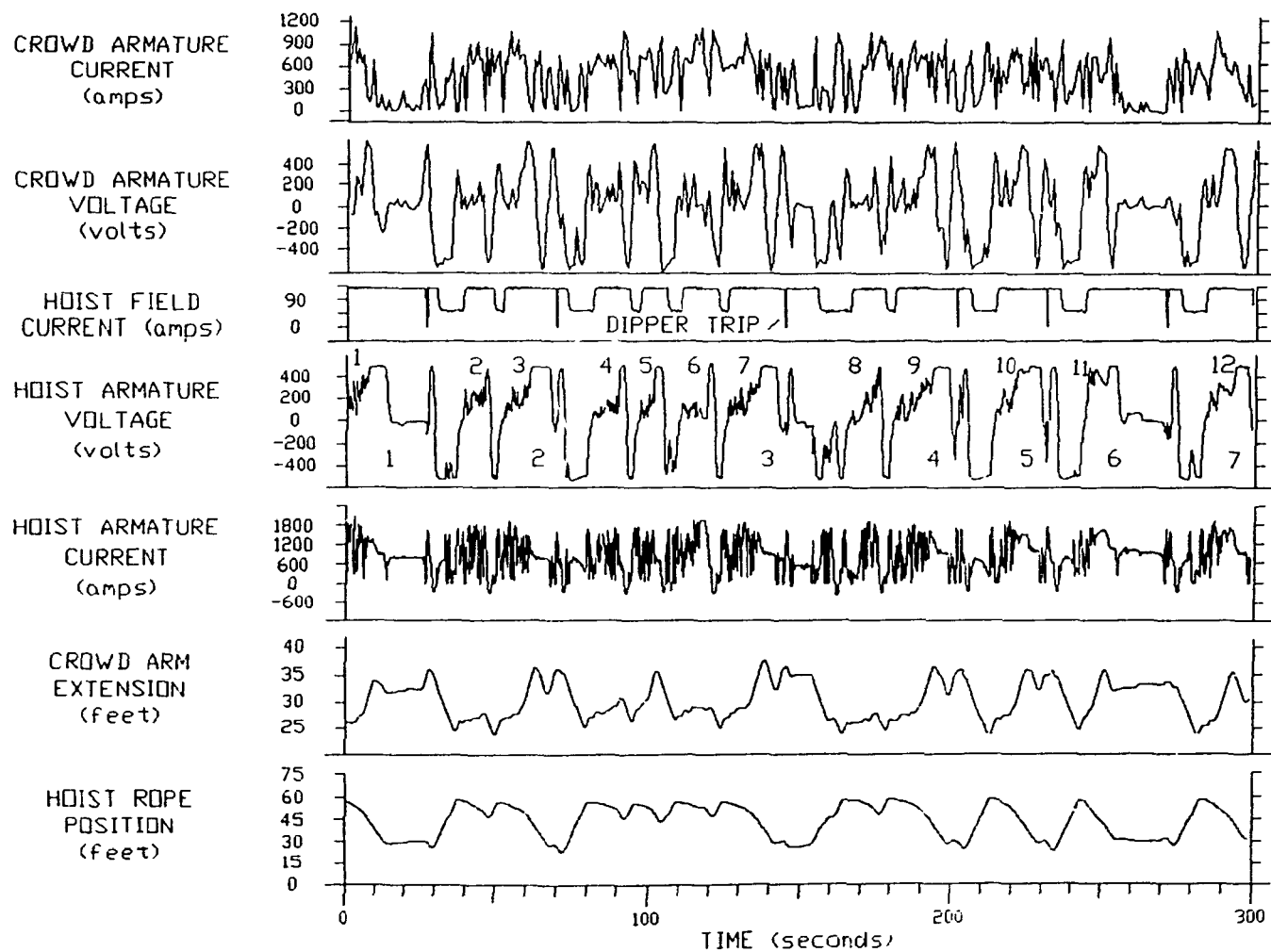
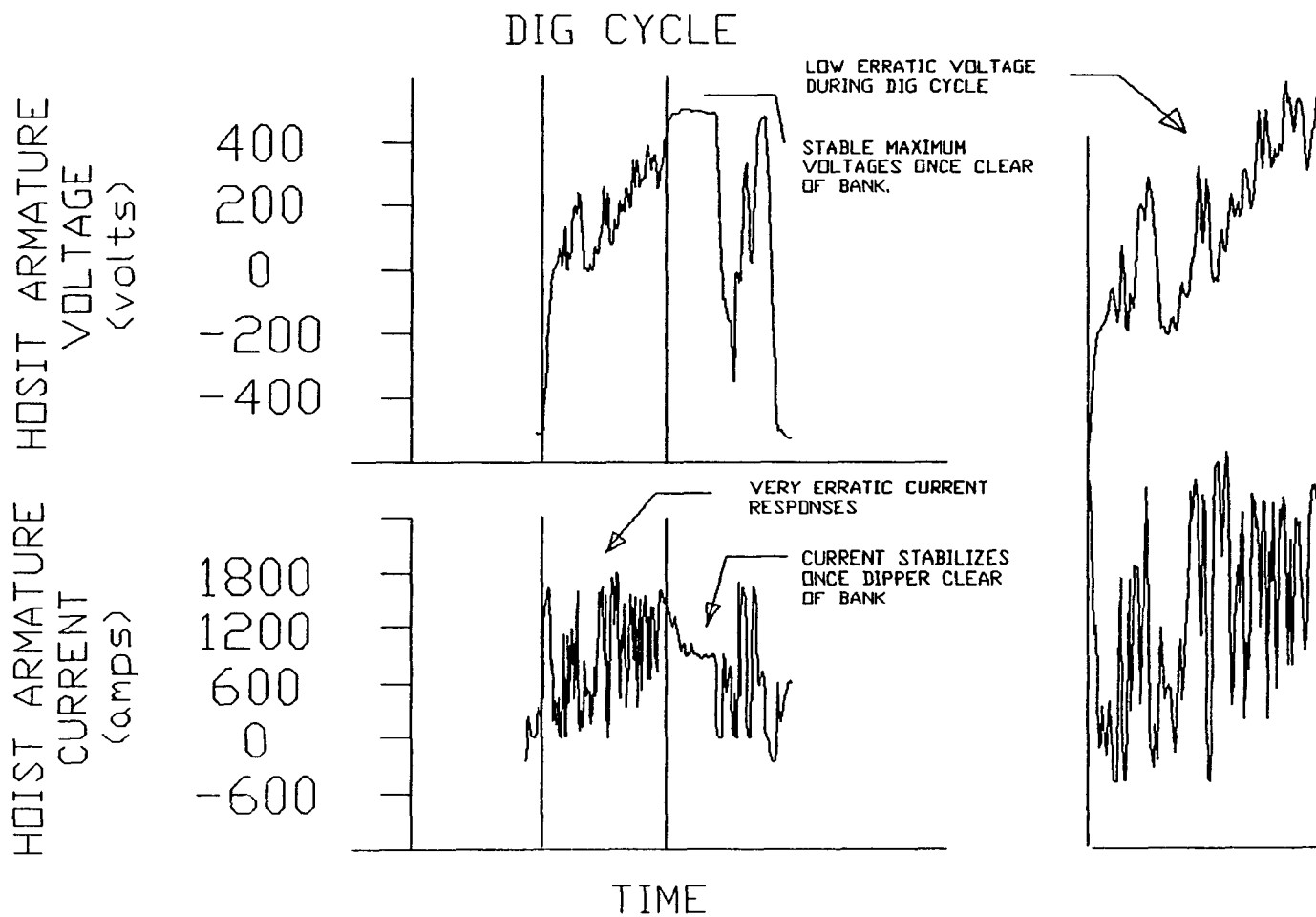


Figure 8.3.7 - Particulars of hoist motor behaviour in difficult digging.



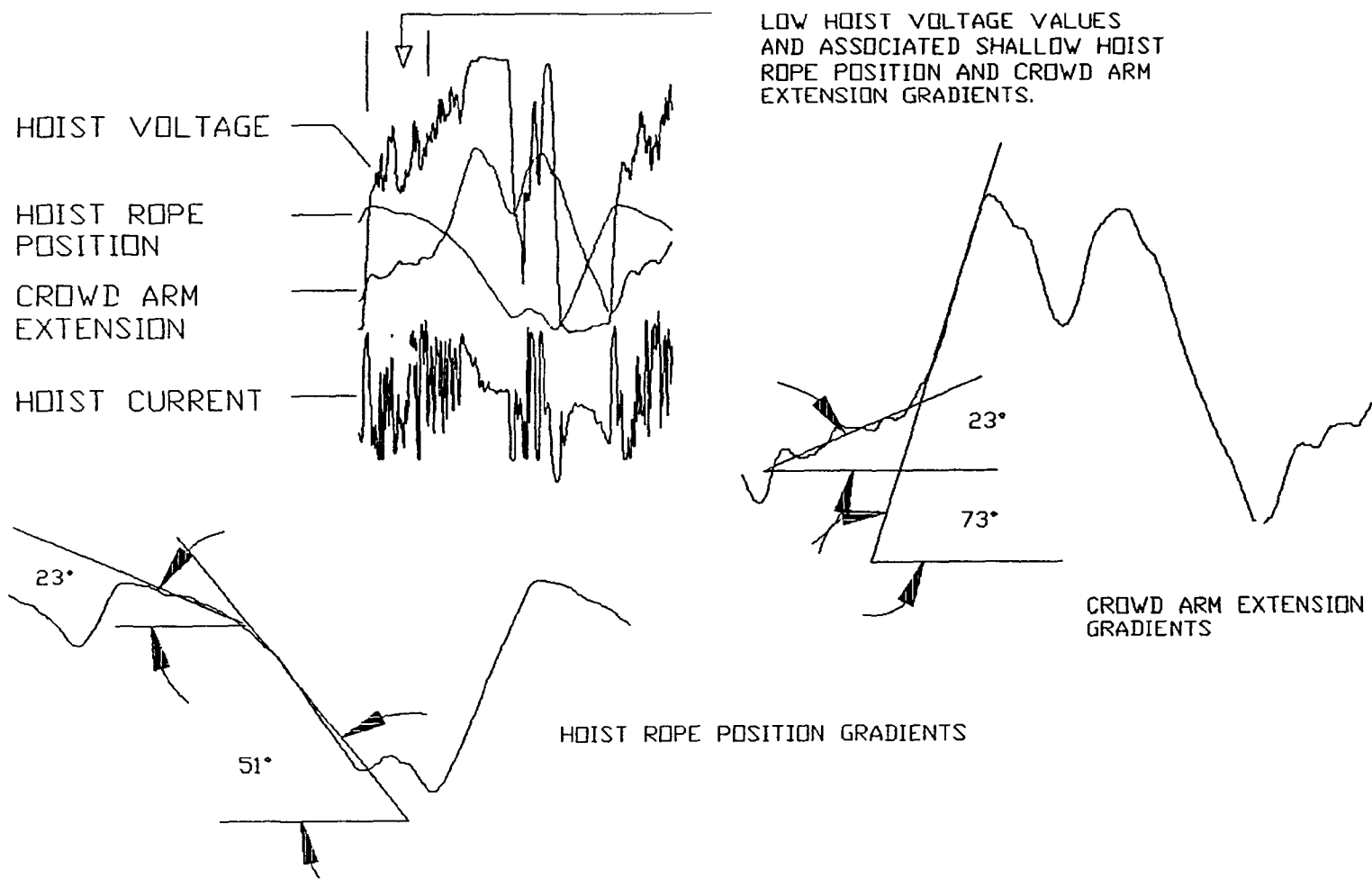
the amplitudes of which range in value from nearly zero amps, to maximum stall current. The spikes in the signal represent transient torque loads and/or overloads of the hoist motor which result from momentary impediments to dipper advance (i.e. the strong interlocking of coarsely fragmented blocks). Once the obstruction is overcome, the hoist current drops to near zero as the torque load across the armature is suddenly released. Conversely, in easy digging conditions resistance to dipper travel is minimal, and the hoist motor current maintains uniform, stable values throughout the duration of the dig cycle. The fashion in which these characteristics of hoist motor response have been employed to derive measures of diggability, is addressed in greater detail in section 8.7.1 - Diggability Equation.

The hoist rope and crowd arm position signals corroborate the above observations. Note the short trajectories defined by these signals for the ground preparation cycles. Most notably, these traces display none of the continuity associated with easy digging conditions. The shape of these traces reflect the difficult digging conditions in the toe area of the muckpile by their shallow gradients during the first half of the dig cycle, see Figure 8.3.8. Once the dipper clears the lower portion of the muckpile, these signals show a distinctive change in gradient or digging effort, reflecting a return to looser, well broken ground towards the upper portion of the muckpile.

Isolation of Remaining Load Cycle Elements: Swing and Dump.

The nature of the current research has considered the dig component of the loading cycle to be of primary concern. It is possible to employ the traces, however, to determine the nature and duration of the remaining three shovel cycle elements; swing to truck, dump and swing back to the muckpile. The manner in which this done is described below, and relates to Figure 8.3.9 which illustrates the isolation of each of the four cycle elements.

Figure 8.3.8 - Relations between hoist and crowd position and hoist motor responses.



Swing to Truck.

Following the dig cycle, the shovel swings to either side to a waiting truck. This interval is indicated in frame B of Figure 8.3.9. Swing is said to commence upon the onset of crowd arm retraction, ie. pulling the loaded dipper out of the muckpile, as evidenced by the crowd arm extension signal. The return to crowd arm extension during the swing cycle is to properly position the dipper at the assumed centre of the waiting truck's box. Note the gradual decrease in hoist motor voltage and relatively static response of the hoist armature current signal during this interval. This is reflected by the shallow gradient of the hoist rope position signal during the swing cycle ie. slow or no hoist motion. The end of the swing to truck cycle is marked by the activation of the dipper trip relay.

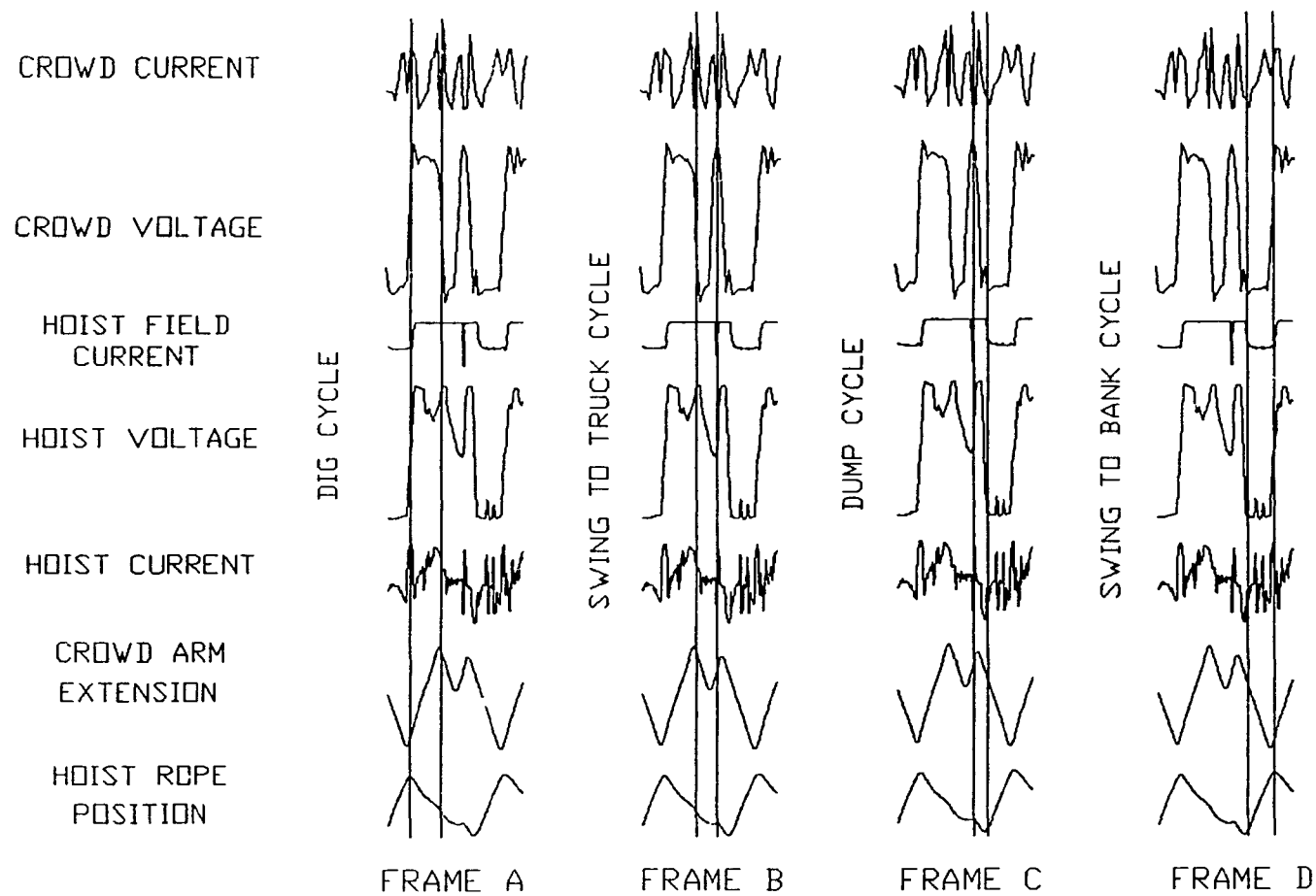
Dump Cycle.

The duration of the dump cycle is as indicated in frame C of Figure 8.3.9 and defined as follows: This cycle initiates with the activation of the dipper trip relay, upon which material flows from the dipper into the waiting truck. Synchronous with this action is the onset of full crowd arm retraction and hoist, in order to facilitate evacuation of the full dipper, and get a "jump" on the swing to bank cycle. Note that "immediately" following activation of the dipper trip relay, the hoist armature voltage signal reaches maximum values; a distinct peak associated with hoisting the still fully loaded dipper. The hoist armature current quickly approaches zero as the material leaves the dipper. The dump cycle is said to end upon engagement of the hoist weak field current.

Swing to Bank.

The fourth and final cycle, is swinging back to the bank in preparation for the next dig cycle. The duration of this cycle is equivalent to the duration of the hoist current weak field mode of operation, during which the dipper is returned to ground level.

Figure 8.3.9 - Isolation of all four load cycle elements.



8.3.3 Polycorder Motor Performance Data.

An example of the data recorded by the Polycorder data-logger is presented in Figure 8.3.10. This data was recorded at the same time as that depicted in Figure 8.3.1 by the G.E. monitor, in that sense they represent the same actions of the shovel for each loading cycle. The traces correspond in order to the instrumented parameters listed below:

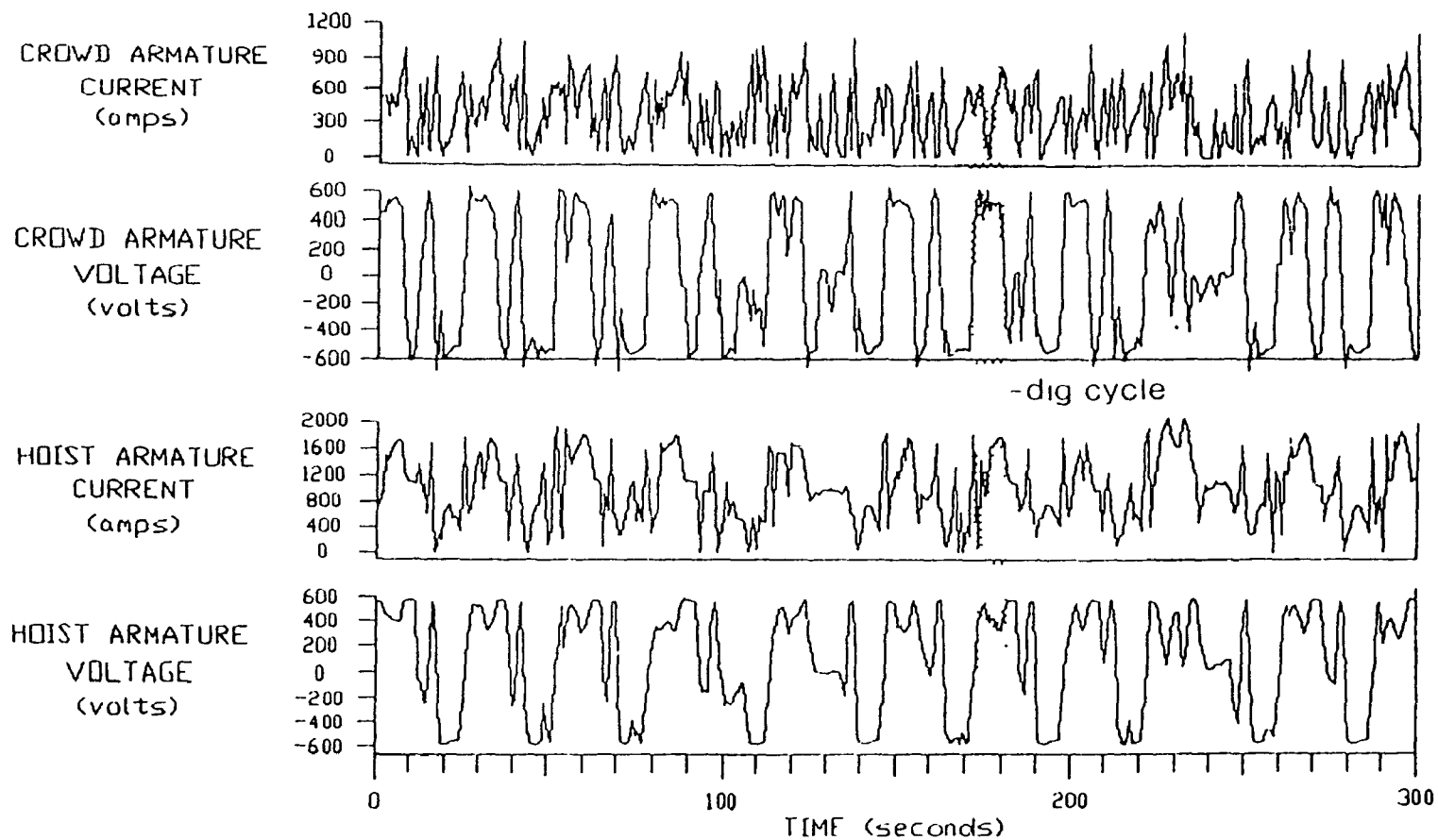
1. Hoist armature voltage.
2. Hoist armature current.
3. Crowd armature voltage.
4. Crowd armature current.

It is immediately evident that the particulars of the Polycorder traces, when compared to those of the G.E. monitor, are less defined or detailed. This a consequence of the comparatively slow maximum sampling interval of the instrument - 820 ms. The G.E. traces were recorded at 100 ms sampling interval, hence, the Polycorder will plot only one data point for every eight recorded by the G.E. monitor.

Initially, the application of the Polycorder was intended solely to collect data on the behaviour of the crowd motor, for analysis in conjunction with the data recorded by the G.E. monitor. However, when a second field period became necessary, in the absence of the G.E. monitor, the Polycorder provided the sole instrumentation of the hoist and crowd motors. Additionally, the Polycorder was used on occasion to collect motor performance data while operating the G.E. monitor in it's conventional capacity. In this manner dipper load weights could be compared with the responses of the hoist and crowd motors as recorded by the Polycorder.

In the absence of the position sensor data recorded by the G.E. monitor, the dig cycle interval for the Polycorder data is determined through inspection of the crowd voltage signal as illustrated in Figure 8.3.10.

Figure 8.3.10 - Performance parameter traces recorded by the polycorder in easy digging conditions.



It is to be noted that the actions of the crowd motor are linked with those of the hoist motor by way of a governor. The function of which to ensure that the operators do not apply excessive crowd force during the dig cycle, a practice that can cause considerable damage to the shovel. The governor acts to limit crowd force by linking it with the amount of hoist force employed by the operator. For example, if the operator is only using 20% of available hoist power he can access 20% of his total crowd power, when full hoist is applied he can apply full crowd. This relationship explains why a component of the crowd voltage traces so clearly define the duration of the dig cycle. It is only during the dig cycle that the operator applies full hoist, hence this interval will correspond with the interval of the crowd voltage trace that represents maximum crowd speed (voltage). This relationship does not exist during controlled gravity lowering of the dipper.

8.3.4 Signal Interpretation - Polycorder data.

Having established a means of isolating the dig cycle from the remainder of the loading cycle, the responses of the hoist and crowd motors in easy and difficult digging conditions are examined. For purposes of comparison with the G.E. data, the same examples of easy and difficult digging considered in section 8.3.2 are employed.

Easy Digging

The data of Figure 8.3.10 is characteristic of easy digging conditions. It was recorded at the same time as the G.E. monitored data presented in Figure 8.3.1. The same attributes of hoist motor behaviour associated with easy digging are reflected in the Polycorder monitored hoist data. Hoist voltage values are characteristically high and display the same "plateau and valley" traces previously identified. The hoist current responses are also reflective of the relationships identified within the G.E. monitored data for this signal.

The crowd motor responses associated with easy digging conditions display the following characteristics: Crowd armature voltage reaches maximum values almost immediately, and remains high throughout the dig cycle. The secondary peak on the crowd voltage trace for each cycle corresponds to retraction and then extension of the crowd arm during dumping.

The crowd current signal is considerably less responsive to the actions of the shovel during each of the four load cycle elements. Based on this signal alone, it would not be possible to determine the actions of the shovel during any time frame.

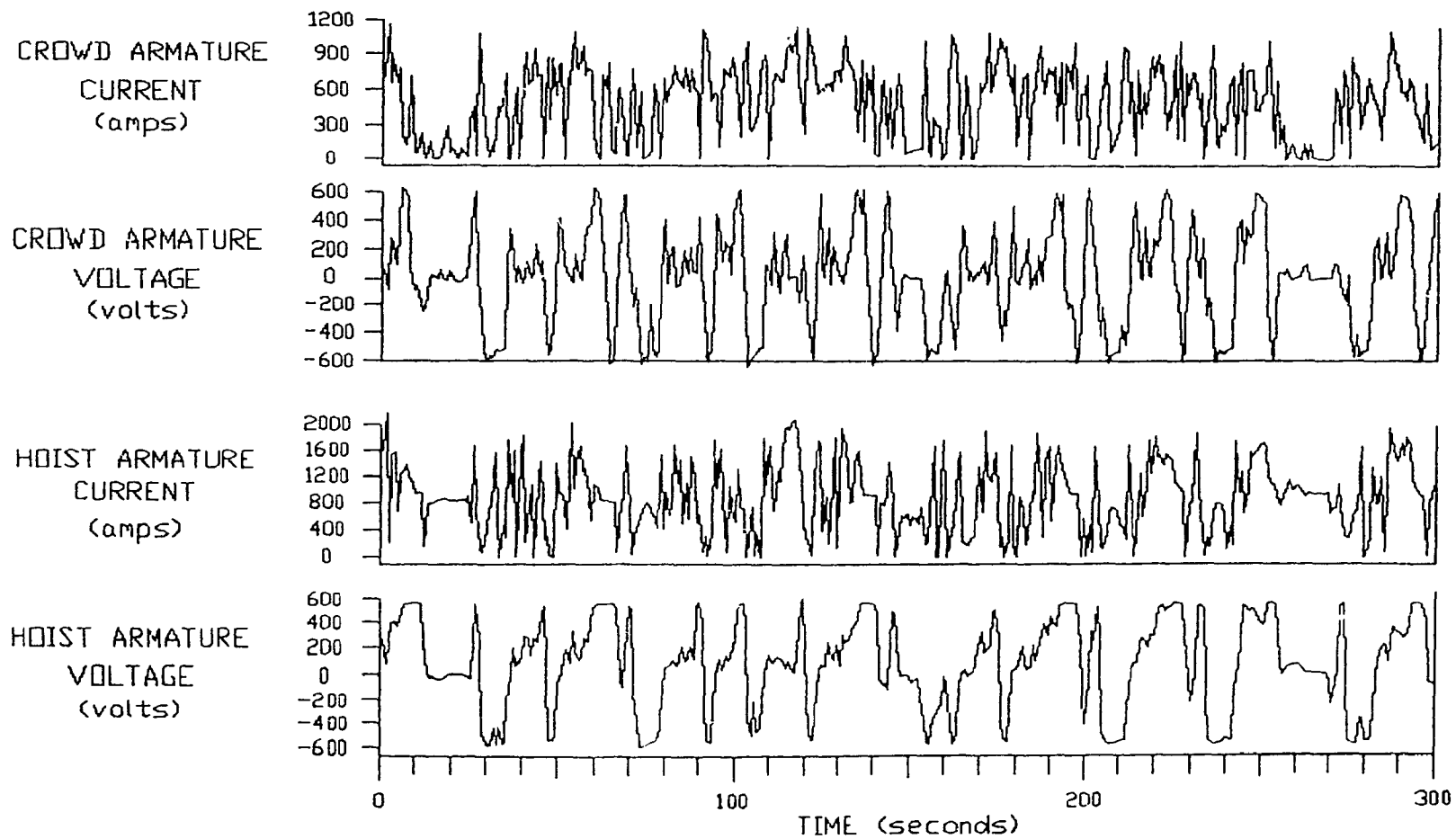
Difficult Digging.

Figure 8.3.11 illustrates the responses of the Polycorder monitored data for difficult digging conditions. Again, the hoist motor responses are seen to exhibit the same trends previously defined by the G.E. monitored data. Low hoist voltage values and erratic hoist current responses dominate during the dig cycle.

The values of crowd armature voltage for difficulty digging conditions are lower and more erratic than those associated with easy digging. This reflects the added effort required for dipper penetration, as in this case, for conditions characterized by hard toe. Note, as will be born out in the remainder of this chapter, variations in crowd motor behaviour are almost exclusively related to very hard toe. Under normal operations (ie. digging properly blasted ground) little variation in the response of this parameter is seen.

The crowd armature current trace again appears relatively insensitive to the actions of the shovel throughout the loading cycle. It can be said however, that in difficult digging conditions there exist episodes during which high currents are sustained. This is of little significance in that these high current episodes transcend the boundaries which define the various load cycle elements. It is concluded that the crowd

Figure 8.3.11 - Performance parameters recorded by the Polycorder in difficult digging conditions.



current signal is incapable of providing reliable feedback on the actions of the shovel, or its response to variations in ground diggability.

8.4 CONTROLLED STUDIES - "DIGGING AIR" AND ANALOG DATA.

It was elected to conduct a series of brief, controlled studies in order to ensure that the fundamental conclusions drawn regarding motor responses were accurate. The first involved instrumenting and monitoring hoist, crowd and swing armature current with a commercial analog strip chart recorder. For this, a HOIKI 8802 MEMORY Hi CORDER was connected to the same test points in the shovel control cabinet to which the Polycorder was connected. Typical records of the analog signals as recorded in easy digging conditions, are presented in Figure 8.4.1. This data helped to confirm the validity of the responses recorded by both the General Electric monitor and the Polycorder. The same observations regarding hoist and crowd armature current behaviour, are readily recognized in the analog traces of these parameters. The fact that analog records of crowd armature current responses are equally ambiguous, allay concerns that faulty instrumentation (the Polycorder) may have been the cause.

The second controlled study involved what is referred to as "digging air". It was requested of the shovel operator that he conduct a series of digging cycles with the dipper following a trajectory just above the surface of the muckpile. In this manner it would be possible to establish a base reference ie. extremely easy digging, against which the responses of all monitored parameters could be contrasted. The traces of the monitored parameters obtained from this effort are presented in Figure 8.4.2.

The results of the study further support the conclusion that hoist motor responses are the more sensitive to variations in diggability. The hoist armature voltage traces show that with zero load (ie. empty dipper), hoist drum rpm achieves and maintains maximum values throughout the dig cycle. The hoist armature current signal is also extremely reflective of the zero

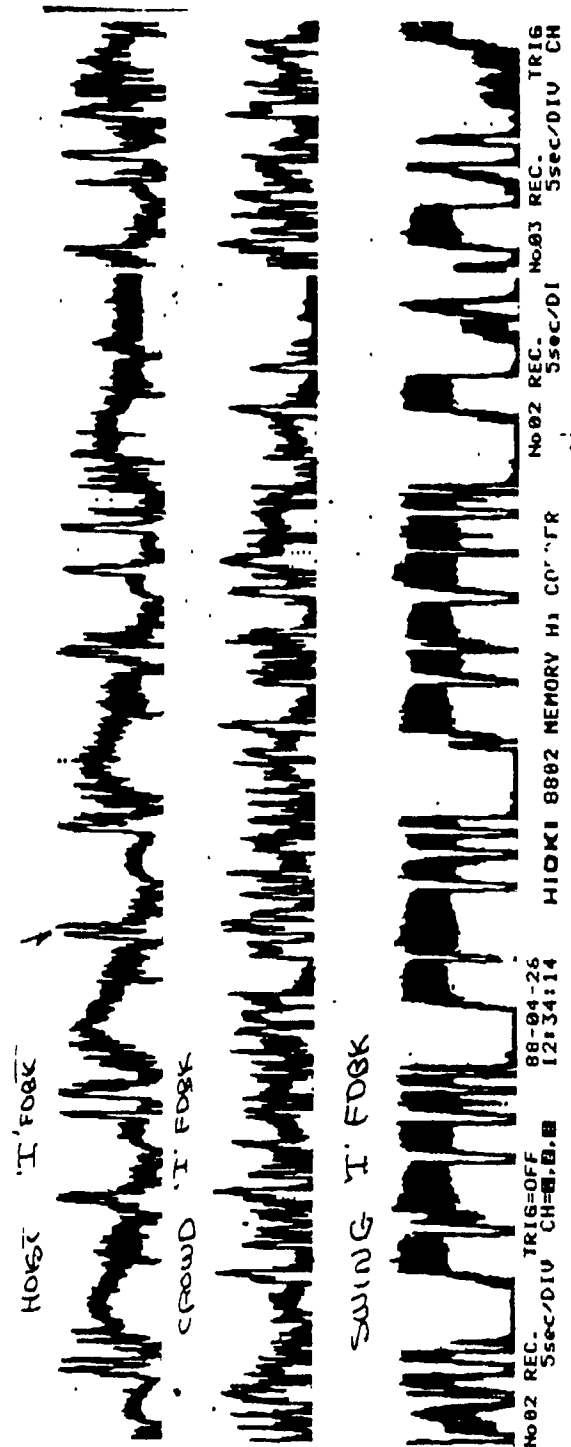


Figure 8.4.1 - Analog records of hoist, crowd and swing motor armature currents for easy digging conditions.

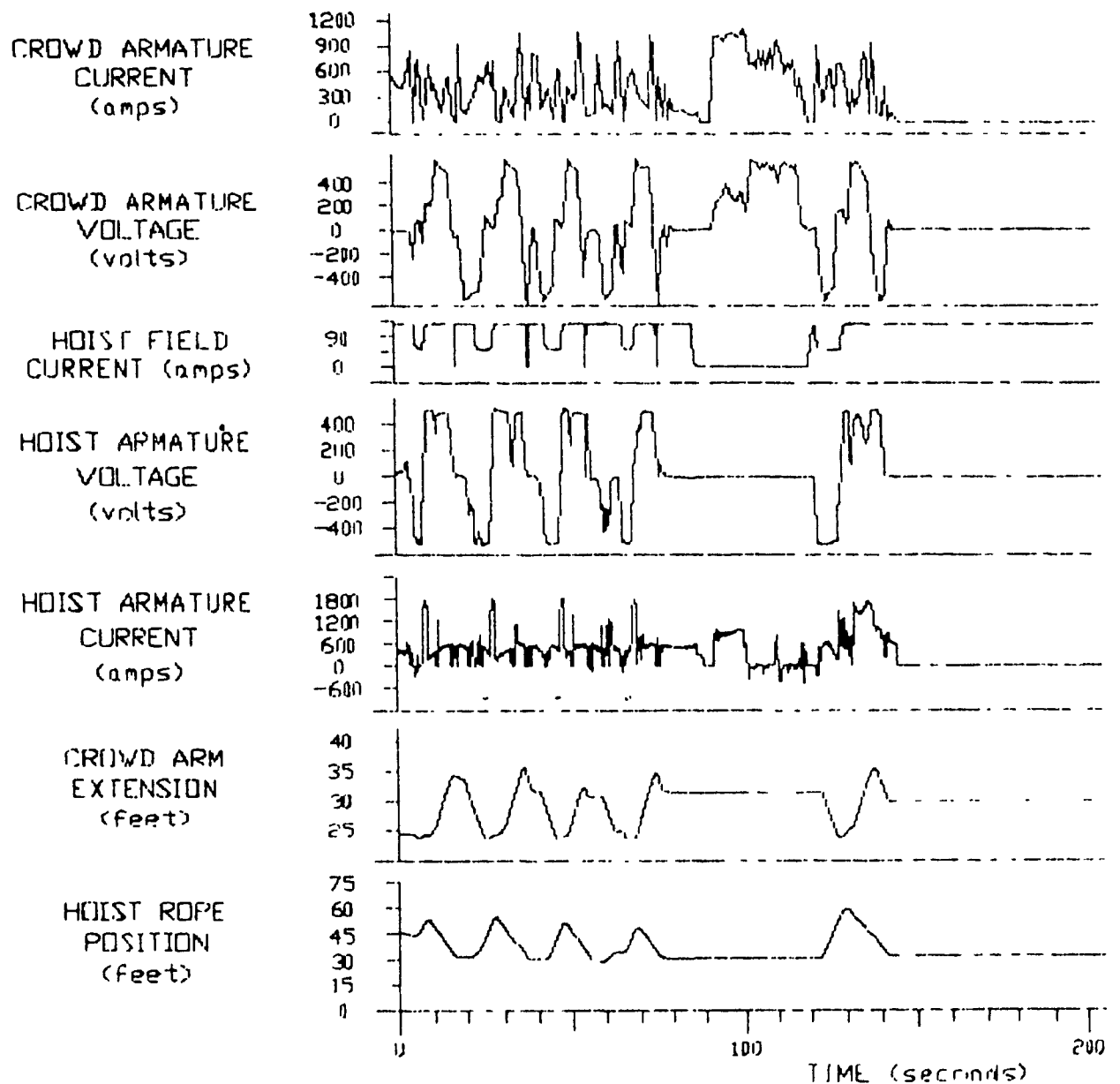


Figure 8.4.2 - Controlled digging study - performance parameters recorded while "digging air".

load conditions. Following the initial surge upon hoisting, the signal drops rapidly and remains relatively static throughout the dig cycle. This reflects the constant load imposed by the combined weight of the crowd arm and empty dipper; approximately 200,000 lbs or 100 tons (95,000 kg). The response of the hoist motor during this test was encouraging, though not surprising. It was possible that the results of the test would have revealed that the monitored data from normal digging was predominantly a product of moving a 200,000 lb mass about; be it through air or a fragmented mass.

This is exactly what appears to be evidenced by the crowd motor responses during the digging air trials. It is difficult to discern any real differences between the crowd motor responses for imaginary versus actual digging. Note that upon a return to normal operation of the shovel, cycle #4 Figure 8.4.2, the hoist motor parameters display a marked contrast between real and imaginary digging. However, the crowd motor traces display little if any difference. Potential reasons for the lack of crowd motor responsiveness to digging conditions are now discussed in the following section.

8.5 CROWD MOTOR DYNAMICS.

To begin, it must be appreciated that the crowd motor is not a work producing motor in the sense that it does not greatly contribute to dipper filling. It is designed to assist the hoist drive as the dipper penetrates the muckpile (P&H technical bulletin X-695 D1-378). In that sense the true effort of the crowd motion is restricted to the beginning of the dig cycle. The penetration of the muckpile places significant demands upon the crowd motor. Thrusting the dipper into a mass of irregularly fragmented material, often striking un-fractured projections and occasionally un-broken ground, will result in frequent massive shock loading of the crowd motor and drive mechanism. In order to protect the crowd motor from instantaneous and progressive damage, the shovel manufacturer employs a "V" belt transmission system, to transfer power from the crowd motor to the shipper shaft gear train. This configuration is displayed in Figure 8.5.1. The net effect of these elastic belts is to absorb the high amplitude energy of the shocks from sudden impact loads, damp it and gradually transfer the dissipated energy to the crowd motor and drive mechanism. It is believed that the action of these energy absorptive belts is responsible for the relative lack of crowd motor responsiveness (especially crowd armature current) to variation in ground diggability.

Theoretically, the belts are tensioned to allow the full stall torque of the crowd motor (ie. shock loads are absorbed by the "V" belts). The presence of these energy absorbing belts introduces a second variable that must be addressed if crowd motor responses are to be considered in assessing diggability. Both prior investigations (Williamson et al., 1983 and Mol et al., 1987) have involved monitoring performance of several shovels (in both studies only P&H shovels were investigated, all of which employed the same crowd drive mechanism described above) through crowd motor responses. It is possible that the condition of the "V" belts varied from one shovel to another. For example,

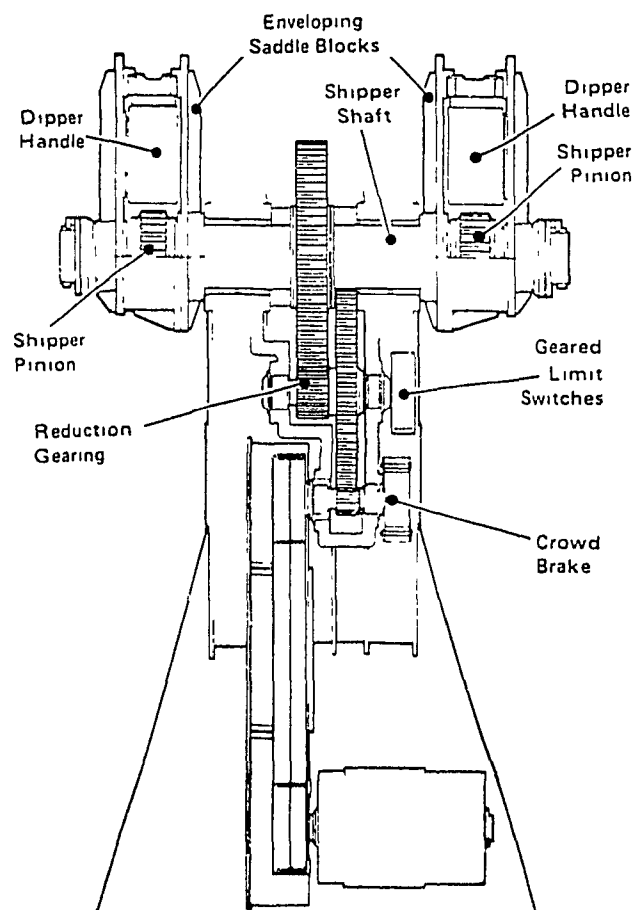
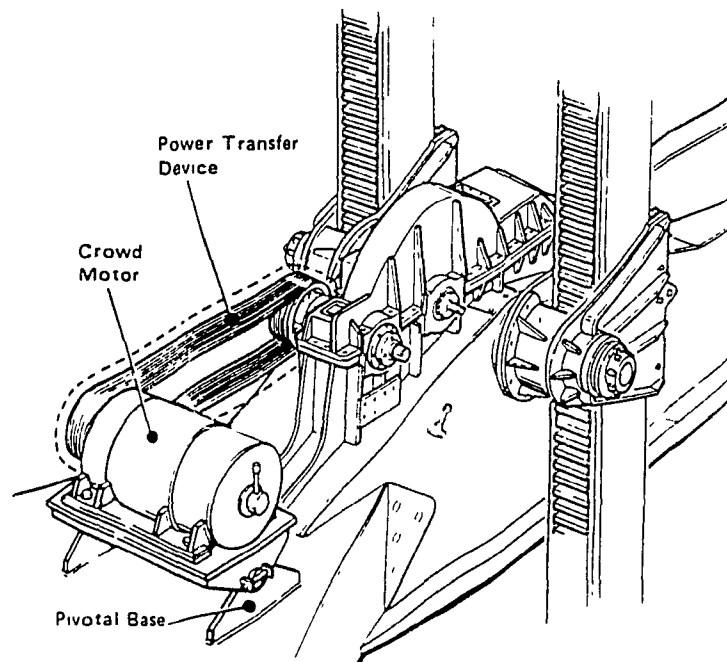


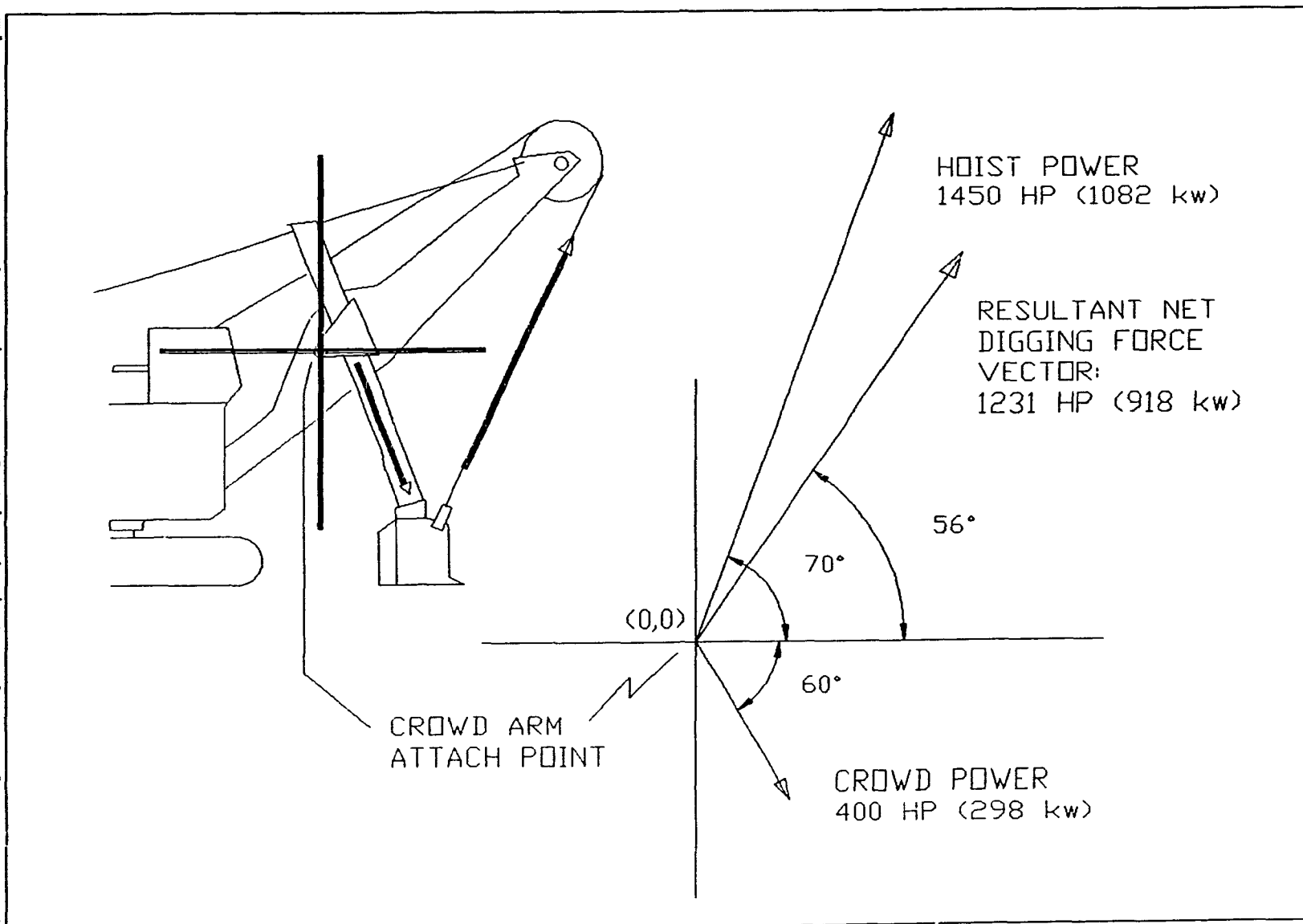
Figure 8.5.1 - Schematic of crowd motor "V" belt power transmission system, (Source, P&H Harnischfeger).

the belts on each shovel may not have been of the same age. Some belts may have been broken or excessively worn. It is possible that all were not equally tensioned. Excessive build up of dust in the transmission housings of the shovels investigated could increase friction between the belts and the drive drum, and reduce slippage. Each of these would be of limited concern to mine maintenance, however, it is possible that within the confines of a controlled scientific investigation, their combined effect could be significant.

An additional concern in addressing the response of the crowd motor is the fact that hoist motion antagonizes crowd motion at the start, and during much of the dig cycle. Figure 8.5.2 illustrates the digging force vector components associated with hoist and crowd motion during the dig cycle. The hoist motor, as the primary work motion motor, is considerably stronger (1450 H.P. @ 475 volts - 2250 peak H.P.) than the crowd motor (400 H.P. @ 475 volts - 630 peak H.P.).

The above observations and conclusions drawn regarding crowd motor behaviour have been corroborated by (through) the shovel manufacturer; P&H Harnischfeger, (Czubkowski, pers. comm., 1988). This also revealed, that in past trials conducted by P&H on crowd motor behaviour, it was observed that crowd responses were relatively independent of material diggability. Tests which involved thrusting the crowd arm into coarse blocky fragmentation tended to produced the same responses as for thrusting into looser, well broken material. It is the general consensus of P&H that crowd responses were "ERRATIC" (unpredictable) and that attempts to relate them to ground diggability would lead to erroneous conclusions.

Figure 8.5.2 - Schematic of relative hoist and crowd component forces during dig cycle.



8.6 ANALYSIS OF SHOVEL PERFORMANCE PARAMETERS - METHODOLOGY.

This section will describe the approach adopted to process and analyze the extensive data base collected during the field period. Prior sections of this chapter have addressed the means identified to enable isolation of the dig cycle from each load cycle. The following subsections will describe software designed to permit the manual and automated extraction and processing of dig cycle data from the continuous records of shovel performance for each monitoring period.

8.6.1 Software Development - KSHOVEL.

KSHOVEL is a program written in Microsoft "C", with the assistance of Mr. R. Akhras (software programmer, McGill University), designed specifically for the purposes of the current research. It enables the graphic display, extraction and further processing of the monitored data associated with specific operating cycles namely, the dig cycle. The program generates a display of the monitored data as illustrated in Figure 8.6.1. This was an important step in the analysis of the data as no available commercial software was able to handle a graphic display of the complete data. There are an average 42,000 data points collected for every 10 minutes of shovel monitoring. AutoCAD version 9.0 was used to generate the records of shovel performance displayed in the previous sections. Special routines written in "C" (with the assistance of C. Morency, software programmer, McGill University) were used to generate the "script files" required by AutoCAD, to process and display the records of monitored shovel performance data. AutoCAD provided an excellent medium with which to generate original and modified views of the data traces. However, AutoCAD required extensive treatment of the data prior to its import, each data point must be read as a vector. This meant the already large data files would swell to 5 Megabytes for every 10 minutes of data. Furthermore, the memory (storage space) required to save the generated traces of the data required an additional 6 Megabytes.

Once the data is displayed by KSHOVEL, the user can select one of two modes - manual or automated, to identify the duration of the dig cycle from each complete loading cycle.

Automated Identification of Dig Cycles.

Automated selection of the dig cycle is accomplished as the software reads and generates the graphic display of the monitored data. The dig cycle is identified by the program through analysis of the crowd arm extension signal, as it's trace is generated on the screen. Three user adjustable calibration factors are used to modulate a moving average applied to the signal, to isolate the duration of the dig interval. The approach follows the logic described in previous sections; sustained positive gradients on the crowd arm extension (C.A.E.) signal correspond to dig cycle intervals.

TIME CALIBRATION is a calibration factor set to a theoretical "assumed" minimum dig cycle duration. Upon identifying a change in slope from positive to negative on the C.A.E. signal, the program references the duration of positive slope against TIME CALIBRATION. If the duration is greater than TIME CALIBRATION the interval is assumed a dig cycle, and subjected to verification by IDLE CALIBRATION and SLOPE CALIBRATION.

IDLE CALIBRATION calibrates the time within an assumed dig cycle during which the shovel was not digging. If during an assumed dig cycle there was a non digging element of greater duration than IDLE CALIBRATION the cycle is discarded. A large IDLE CALIBRATION value will allow for more disturbance in the continuity of the C.A.E. which defines each dig cycle. Recall that in difficult digging the C.A.E. signal trace is often characterized by small peaks and valleys, indicative of strained dipper advance during the dig cycle.

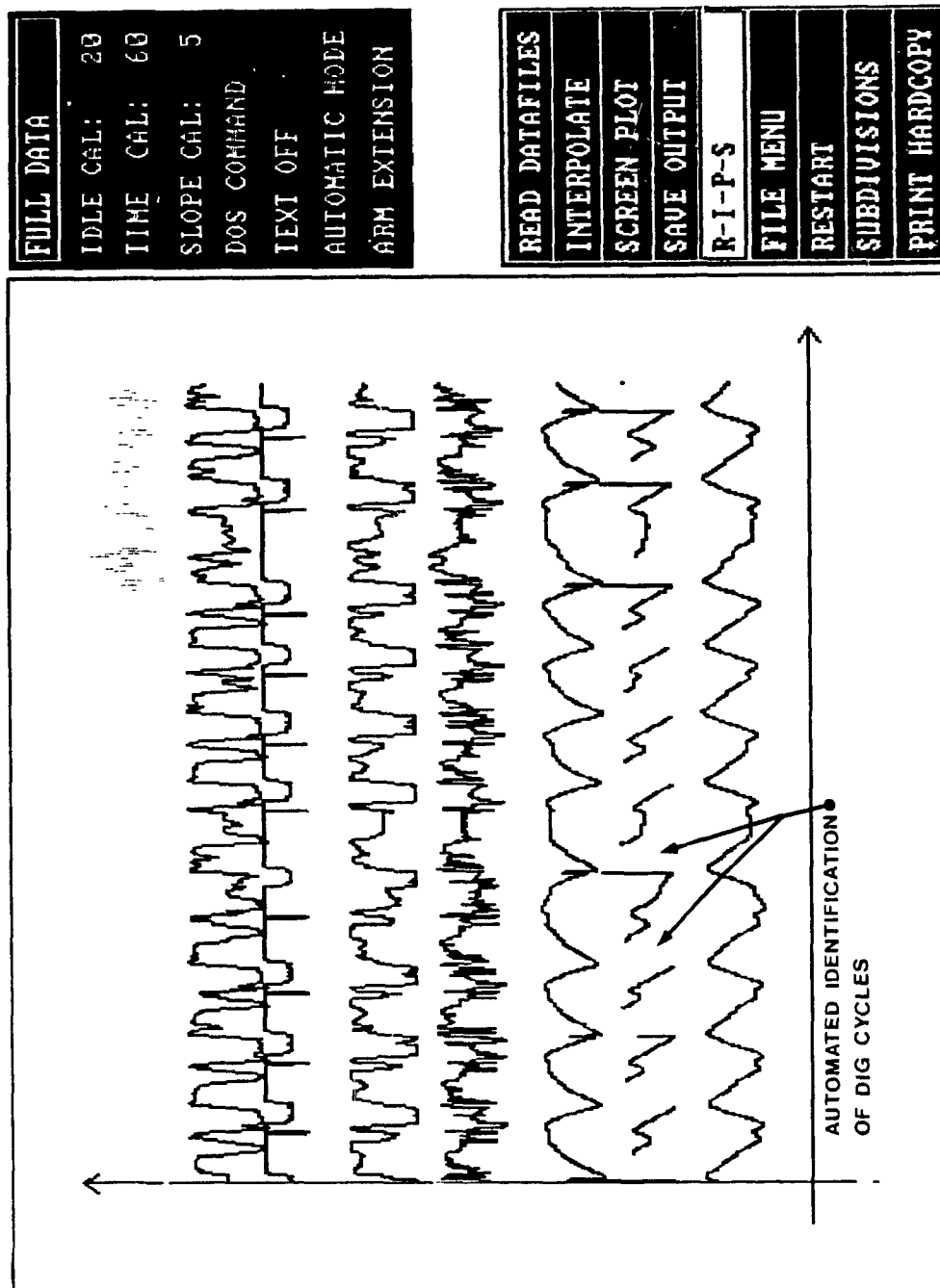


Figure 8.6.1 - Example of monitored performance parameters displayed by KSHOVEL software.

SLOPE CALIBRATION calibrates slope summation after each point during an assumed dig cycle. The program verifies that during an assumed dig cycle the slope of the C.A.E. remains above that specified by the SLOPE CALIBRATION value.

The above describes only one of several approaches that could have been adopted for automated isolation of dig cycles. This method proved reliable for about 90% of the cycles processed through the software. As mentioned, the "disturbances" in the C.A.E. trace associated with difficult digging occasionally proved confounding to the software. Additional algorithms based on the responses of the other parameters will have to be developed to ensure 100% accuracy in determining the duration of each dig cycle. It for this reason that a "manual mode" of dig cycle isolation was built into the KSHOVEL program.

Manual Identification of Dig Cycles.

In the manual mode, the software generates the traces for all monitored parameters on the screen and then displays a set of cross hairs that are maneuverable by a mouse. The user can then "pick" the start and end of the dig cycle for each complete loading cycle displayed on the monitor. In the interest of accuracy, all of the dig cycle data presented in this thesis was manually picked.

The KSHOVEL program works with either the General Electric data (FULL DATA) or the Polycorder data (POLYCORDER). When analyzing the Polycorder data by itself only the manual mode is available since no dipper position data is available upon which to base automated processing.

8.6.2 Calculated Dig Cycle Statistics.

Whether manually or automatically picked, the data associated with each cycle, for all parameters is written to a separate file. The isolated dig cycle data is then further processed by the software to derive the following statistics for each dig

cycle:

1. Dig cycle time, in seconds.
2. Average hoist armature voltage, in volts.
3. Average hoist armature current, in amps.
4. Hoist current signal length. This value is the absolute total length of the hoist current trace over the length of the dig cycle. This value has proven a reliable indicator of digging effort. Long hoist signal lengths are commonly associated with difficult digging conditions.
5. Percentage of hoist current values > 1100 amps. This value is also a good indicator of relative digging effort. A high percentage of hoist current values greater than 1100 amps during a dig cycle, is characteristic of easy digging conditions. The 1100 amp threshold was selected because average daily hoist values below 1100 amps were associated with the more difficult digging conditions.
6. Hoist armature voltage standard deviation, calculated over the dig cycle.
7. Hoist armature current standard deviation, calculated over the dig cycle.
8. Average crowd armature voltage, in volts.
9. Average crowd armature current, in volts.
10. Crowd armature voltage standard deviation, calculated over the dig cycle.
11. Crowd armature current standard deviation, calculated over the dig cycle.
12. Minimum crowd arm extension. Defines the point of maximum crowd arm retraction (ie. position at the start of the dig cycle) for each dig cycle.
13. Maximum crowd arm extension. Defines the point of maximum crowd arm extension (ie. position at the end of the dig cycle) for each dig cycle.
14. Crowd. The amount of crowd arm travel associated with each dig cycle, calculated from 13 minus 12 above.

15. Maximum hoist rope length. Defines the length of the hoist rope at the start of each dig cycle.
16. Minimum hoist rope length. Defines the length of the hoist rope at the end of each dig cycle.
17. Hoist. Defines the amount of hoist rope travel associated with each dig cycle.
18. Cut Ratio. Is the ratio of crowd to hoist travel associated with each dig cycle (ie. number 14 ÷ 17). This value provides an indication of the proportions of crowd (lateral thrust) and hoist (vertical motion) applied for each dig cycle. In dig cycles with more crowd than hoist motion, the dipper pursued a deep cut through the muckpile.
19. XY-Ratio. This value is associated with the actual dipper trajectory followed through the muckpile for each cycle. The manner in which dipper trajectories are determined is presented in the following section. The XY-ratio refers simply to the ratio of the total horizontal (X-axis) to vertical (Y-axis) motion for each trajectory. Cut Ratio (18 above), refers only to the relative motion of the components used to define dipper trajectory, not the trajectory itself.
20. Trajectory length (feet). The length of the dipper trajectory for each dig cycle.
21. Velocity (feet/sec.). Is the average velocity of the dipper for each dig cycle. It is calculated as trajectory length over dig cycle time.
22. Hoist power. The average hoist power for each dig cycle. Calculated as hoist voltage times hoist current.
23. Crowd Power. The average crowd power for each dig cycle. Calculated as crowd voltage times crowd current.

A summary of daily averages for each of the parameters in presented in Table 8.6.1, which covers the period during which blasts EZ#3 and EM#1 were excavated; August 15th to 25th, 1988. Typical examples of the monitored performance parameters (signal traces) for each of these days are presented in Appendix G.

In addition, the KSHOVEL program writes the values of the crowd arm extension and hoist rope position for each dig cycle to a separate file (these are the position readings for each 100 ms during the dig cycle). This data is used to determine the actual dipper trajectory for each dig cycle. Figure 8.6.2 illustrates the manner in which dipper trajectories are determined.

Table 8.6.1 - daily statistics of performance parameters

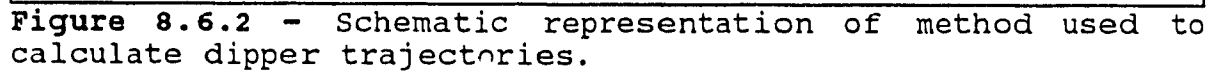
DATE	OPERATOR NUMBER	TIME (seconds)	HOIST VOLTAGE	HOIST CURRENT	HCSL	% >1100 AMPS	HOIST-V STD	HOIST-I STD	CROWD VOLTAGE
AUG 15 S.D.	5	9.47 2.15	252.13 71.52	1190.79 173.48	14154.22 4719.97	65.36 13.62	148.95 34.98	478.30 77.24	0.00 0.00
AUG 17 S.D.	1	11.96 2.86	285.30 52.73	1203.42 186.58	14256.12 4896.01	66.43 14.51	145.96 32.82	478.95 83.55	387.16 72.45
AUG 18 S.D.	2	8.68 1.46	236.93 48.44	1102.56 173.41	12455.22 4951.49	56.76 13.99	157.39 46.26	501.45 67.45	375.36 74.84
AUG 19 S.D.	2	8.34 1.50	303.77 60.73	1191.47 126.49	9789.82 3307.07	65.80 12.86	156.54 40.51	448.97 85.53	443.78 59.82
AUG 20 S.D.	3	9.07 1.21	294.13 66.42	1121.90 128.38	9773.12 2903.64	58.40 14.96	169.57 45.85	439.18 74.78	400.50 88.06
AUG 21 S.D.	3	9.52 1.91	247.55 91.90	1037.97 303.46	14400.52 6221.23	52.28 24.12	171.19 33.64	459.90 70.25	384.90 87.52
AUG 22 S.D.	4	9.42 1.86	300.80 66.01	1206.90 174.57	12808.02 7114.64	66.92 16.49	153.25 40.21	401.53 83.12	385.60 81.29
AUG 23 S.D.	4	8.92 1.49	335.37 62.30	1301.52 138.91	7365.33 3408.74	75.89 13.36	138.16 41.03	364.21 87.04	394.95 91.72
AUG 24 S.D.	1	11.59 2.26	280.31 60.70	1108.62 157.91	15696.35 5622.80	61.31 14.03	180.75 36.42	468.77 74.80	373.92 70.39
AUG 25 S.D.	1	12.36 3.10	290.25 48.09	1217.83 157.79	16404.11 6027.46	67.22 13.27	149.46 30.56	456.21 55.36	371.37 89.81

Table 8.6.1 contd., - Daily statistics of performance parameters

DATE	OPERATOR NUMBER	CROWD CURRENT	CROWD-V STD	CROWD-I STD	min AE (feet)	max AE (feet)	CROWD (feet)	min HR (feet)	max HR (feet)
AUG 15 S.D.	5	0.00 0.00	0.00 0.00	0.00 0.00	25.20 0.90	38.82 1.58	13.62 1.88	41.97 4.42	56.09 0.87
AUG 17 S.D.	1	488.10 108.71	203.04 58.61	200.98 42.02	24.64 0.55	39.88 1.78	15.25 1.85	37.96 3.15	57.09 1.41
AUG 18 S.D.	2	451.73 67.72	248.23 73.22	205.01 42.75	25.35 0.97	38.43 1.85	13.08 1.60	44.58 2.46	56.86 0.81
AUG 19 S.D.	2	440.83 90.59	153.23 55.55	206.33 37.24	25.23 0.69	37.82 2.09	12.59 2.03	40.97 3.72	56.09 1.55
AUG 20 S.D.	3	420.60 72.92	237.51 114.59	206.71 38.56	25.48 1.21	39.53 2.64	14.05 2.26	40.04 3.43	55.94 1.37
AUG 21 S.D.	3	470.30 98.93	239.43 93.72	215.97 39.92	25.08 1.10	38.75 2.08	13.67 1.88	41.76 2.81	56.67 1.31
AUG 22 S.D.	4	514.06 100.46	175.52 56.37	217.46 43.51	25.27 0.82	37.68 2.01	12.41 1.91	40.36 3.65	56.84 1.47
AUG 23 S.D.	4	470.92 90.99	146.11 102.63	169.04 131.74	24.98 0.84	37.54 2.49	12.56 2.36	38.48 3.64	55.87 1.29
AUG 24 S.D.	1	523.54 105.05	142.20 101.00	160.03 101.48	25.11 1.01	40.37 1.47	15.27 1.36	39.11 3.77	58.16 1.47
AUG 25 S.D.	1	581.17 91.57	188.75 61.72	228.30 41.54	24.65 0.56	40.17 1.43	15.52 1.40	36.35 3.02	56.55 1.16

Table 8.6.1 cont'd. - Daily statistics of performance parameters.

DATE	OPERATOR NUMBER	HOIST (feet)	CUT RATIO	XY-RATIO	T.L. (feet)	VEL. (ft/sec)	HOIST POWER	HOIST-V INDEX	HOIST-I INDEX	DIGGING INDEX
AUG 15 S.D.	5	14.12 4.68	1.04 0.26	2.89 0.94	34.04 6.46	3.65 0.52	338583 84343	0.92 0.29	1.45 0.53	1.41
AUG 17 S.D.	1	19.14 3.68	0.83 0.20	2.26 0.70	41.99 4.58	3.63 0.57	370760 70075	0.68 0.22	1.19 0.63	0.88
AUG 18 S.D.	2	12.28 2.63	1.11 0.24	3.33 0.71	32.18 3.71	3.76 0.40	307936 74371	0.90 0.26	1.49 0.60	1.43
AUG 19 S.D.	2	15.12 3.76	0.88 0.24	2.65 0.72	33.05 4.91	4.00 0.40	381723 78344	0.68 0.19	1.16 0.36	0.82
AUG 20 S.D.	3	15.90 3.77	0.93 0.25	2.53 0.51	36.45 4.85	4.03 0.31	375708 74457	0.59 0.17	1.12 0.32	0.68
AUG 21 S.D.	3	14.91 3.21	0.96 0.23	2.74 0.60	35.89 4.58	3.84 0.47	316502 104403	0.89 0.37	1.66 0.77	1.67
AUG 22 S.D.	4	16.48 4.14	0.79 0.20	2.47 0.54	34.82 5.21	3.78 0.58	396193 94581	0.66 0.27	1.31 0.68	0.99
AUG 23 S.D.	4	17.40 3.77	0.76 0.22	2.17 0.43	35.04 5.36	3.97 0.51	448293 80090	0.50 0.18	0.79 0.35	0.43
AUG 24 S.D.	1	19.04 3.91	0.84 0.21	2.26 0.48	43.56 4.55	3.85 0.53	371576 73174	0.69 0.23	1.40 0.60	1.05
AUG 25 S.D.	1	20.20 3.37	0.79 0.16	1.96 0.34	42.86 4.09	3.61 0.63	379135 61362	0.70 0.21	1.25 0.58	0.92



8.7 DETERMINATION OF A DIGGABILITY INDEX.

With the previously described statistics calculated for each dig cycle, it was possible to approach their manipulation, in efforts to derive the most suitable equation for a diggability index as a measure of digging effort. The data can be regarded as grouped into two categories; that which defines digging effort (motor parameters) and that which relates to dipper trajectory (position data).

After careful consideration it was decided to derive an equation of diggability based solely upon motor responses. This is not to suggest that dipper trajectory is unimportant, rather, its influence is considerable and complex. The interdependence of muckpile diggability and dipper trajectory will be addressed in detail in section 8.9. The effect on motor responses, of variations in the trajectory followed through the muckpile, have been determined and are well understood. Complicating factors however, such as localized variation in muckpile characteristics, precise knowledge of the distance between the shovel and the muckpile etc., inhibit the direct inclusion of trajectory data for each cycle, in the calculation of a diggability index. The final approach adopted (and explained fully in section 8.11) considers the use of trajectory data as a qualifier of the calculated diggability indices for each day of monitoring. For the present, it is concluded that the inclusion of trajectory data in the calculation of a diggability index would lead, due to lack of control over several variables, to erroneous measures of muckpile diggability.

8.7.1 Diggability Equation.

A diggability index has been established which considers the product of the ratios of the total vertical signal lengths of both hoist armature voltage and current, to the area under each of these signals. The approach is similar in practice to that adopted in both prior shovel instrumentation studies (Williamson

et al., 1983 and Mol et al., 1987), refer section 7.2 - prior investigations. In both cases, however, the crowd voltage signal was considered as the primary indicator of ground diggability. As discussed, the present research has revealed crowd motor responses to be considerably less sensitive to variations in diggability than are hoist responses. The current approach to the determination of a diggability index is therefore unique. Furthermore, by considering current as well as voltage, the index is tempered by including the element of digging effort. The hoist based diggability index equation is given below (ref. Figure 8.7.1):

$$\text{Hoist DI} = \frac{\sum_{i=1}^n |HV_{i+1} - HV_i|}{\sum_{i=1}^n |SR * HV_i|} \times \frac{\sum_{i=1}^n |HI_{i+1} - HI_i|}{\sum_{i=1}^n |SR * HI_i|} \dots (8.7.1)$$

(HOIST VOLTAGE DI) X (HOIST CURRENT DI)

Where: n = Number of readings taken during the dig cycle.
 DI = Diggability Index.
 HV = Hoist armature Voltage.
 HI = Hoist armature Current.
 SR = Sampling Rate - 0.10 seconds for G.E. data.
 - 0.82 seconds for Polycorder data.

The above equation takes into account elements of force or effort, speed and distance as:

- Hoist current is proportional to hoist motor torque or effort.
- Hoist voltage is proportional to the velocity of dipper travel through the muckpile.
- The area under the hoist voltage signal is proportional to the vertical distance travelled by the dipper.

The equation makes use of the previously identified response characteristics of the hoist motor in easy and difficult digging conditions. For example in difficult digging the hoist current trace is typically very "ragged", hence, the absolute length of which will be very long. Additionally, the average value of the

hoist current is correspondingly low, and therefore, the area under the signal is also low. In such cases (difficult digging) the ratio of a high signal length to a low area under the signal

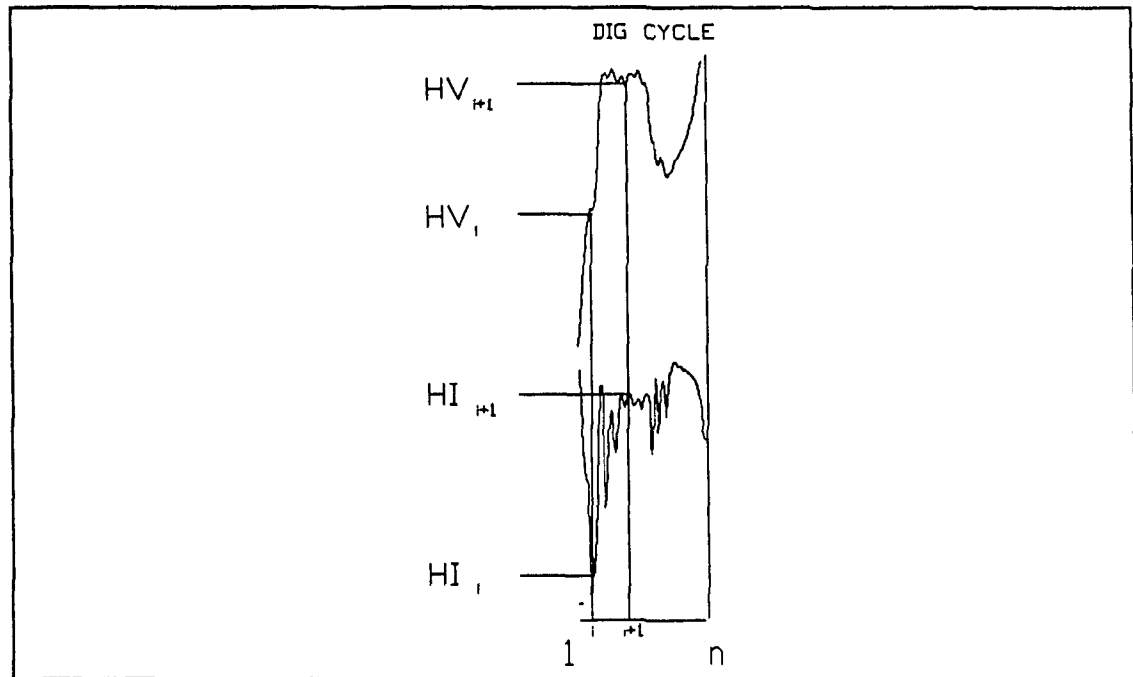


Figure 8.7.1 - Calculation of signal lengths for DI.

will be high. High values of the diggability index are therefore indicative of difficult digging conditions. Conversely, in easy digging conditions signal lengths are low and areas under the signals are high. Diggability indices associated with easy digging are therefore, characteristically low.

The rationale for the above described method of calculating / determining diggability is as follows: Other approaches for rating diggability, considered in the past by shovel manufactures have typically considered only the average values of hoist motor voltages and currents. As will be illustrated in subsequent sections the average values can be misleading. For example, in difficult digging conditions one would expect the highest hoist armature currents; indicating the greatest digging effort. Following this logic, shovel manufactures have tried to assess digging conditions on the basis of power (hoist armature voltage

x current); higher kilowatt-hours (kW·h) being indicative of less favourable digging conditions (Czubkowski, 1989). However, it has been indicated through the current research, that the lowest average currents are associated with the most difficult digging conditions.

This phenomena is explained on the basis of previous observations regarding the hoist current response in easy vs difficult digging. It was illustrated in section 8.3.2 that in the more difficult digging conditions the hoist armature current signal is very ragged. The net effect of which is a lower average current value than recorded in easy digging conditions, in which, the hoist armature current maintains more stable values, the average of which is greater than that typically associated with difficult digging conditions. Treated in its entirety therefore, (i.e. not represented as an average value) the hoist current signal for each dig cycle is much more informative: The spikes in the signal (refer Figure 8.3.7) represent maximum motor stall current values (motor torque overloads) as caused by trying to negotiate a trajectory through a difficult to dig muckpile (eg. coarse interlocked fragmentation or hard poorly fractured toe). Each spike can therefore be viewed as representing a momentary or transient hoist motor overload, resulting in near dipper stalling as the motors strain to overcome the digging resistance. The sudden drop in hoist current, to near zero amps, following each transient spike in the signal, indicates surmounting of the transitory impediment, as the armature torque load is released (eg. fragment interlock was overcome). In essence the signal as a whole chronicles a series of load-release, load-release cycles as the hoist motors experience numerous torque overloads, while the dipper strains to negotiate a trajectory through difficult digging conditions.

In essence, averaging the motor responses, masks information that permits establishing not how much power has been developed, but more importantly for the purposes of assessing diggability,

the manner in which the power has been developed: In hard digging, spikes in the hoist current signal are indicative of motor overloads. While smooth, stable current traces, reflect easy digging conditions that permit the hoist motors to perform optimally by developing maximum power output levels.

The same approach was followed in calculating a diggability index based upon crowd motor responses. This was done for comparative purposes, ie. to evaluate how well does the crowd based diggability index used in prior research, correlate with assessments of diggability from the hoist based diggability index above. The crowd based equation for calculation of the diggability index is given below:

$$\text{Crowd DI} = \frac{\sum_{i=1}^n |CV_{i+1} - CV_i|}{\sum_{i=1}^n |SR * CV_i|} \times \frac{\sum_{i=1}^n |CI_{i+1} - CI_i|}{\sum_{i=1}^n |SR * CI_i|} \dots (8.7.2)$$

$$\{\text{CROWD VOLTAGE DI}\} \times \{\text{CROWD CURRENT DI}\}$$

Where: n = Number of readings taken during the dig cycle.
 CV = Crowd armature voltage.
 CI = Crowd armature current.
 SR = Sampling rate.

It is to be noted that for each dig cycle the voltage and current component of both the crowd and hoist based diggability indices have remained isolated. For example, for each dig cycle the following diggability indices and component indices have been recorded:

1. HOIST DI - DI.
2. Hoist voltage DI - HVDI.
3. Hoist current DI - HCDI.
4. CROWD DI - CDI.
5. Crowd voltage DI - CVDI.
6. Crowd current DI - CCDI.

In this manner it was possible to compare the degree to which each of the above correlated with assessments of diggability as observed during excavation of the test bench, and establish whether some were more sensitive than others.

A summary of the diggability indices determined for each day of the study is presented below in Table 8.7.1. Subjective assessments of muckpile diggability were made for each day of monitoring based on operator comments and were classified/rated as easy, average or difficult.

A comparison of the diggability indices presented in Table 8.7.1, corroborate the observations regarding the lack of crowd motor responsiveness to variations in muckpile diggability. All three crowd based digging indices; CVDI, CCDI and CDI show little correlation with digging conditions as observed during monitoring. The hoist based indices however, consistently reflect the daily operator ratings of diggability. Based on the hoist diggability indices it was possible to define the following boundaries for easy, average and difficult digging:

	DI
Easy Digging	0.4 - 0.8
Average Digging	0.8 - 1.2
Difficult Digging	1.2 - 1.6

Table 8.7.1, Summary of Diggability Indices According to Ratings of Diggability - G.E. Data.

<u>EASY DIGGING</u>	<u>HVDI</u>	<u>HCDI</u>	<u>DI</u>	<u>CVDI</u>	<u>CCDI</u>	<u>CDI</u>
August 20	0.59	1.12	0.68	0.28	0.71	0.20
August 23	<u>0.50</u>	<u>0.79</u>	<u>0.43</u>	<u>0.48</u>	<u>0.67</u>	<u>0.33</u>
avg:	0.54	0.96	0.55	0.38	0.69	0.27

AVERAGE DIGGING

August 17	0.68	1.19	0.88	0.43	0.57	0.24
August 19	0.68	1.16	0.83	0.35	0.71	0.25
August 22	0.66	1.31	0.99	0.49	0.67	0.32
August 24	0.69	1.40	1.05	0.35	0.60	0.21
August 25	<u>0.70</u>	<u>1.25</u>	<u>0.92</u>	<u>0.40</u>	<u>0.52</u>	<u>0.20</u>
avg:	0.68	1.26	0.93	0.40	0.61	0.24

DIFFICULT DIGGING

August 15	0.92	1.45	1.41	not instrumented		
August 18	0.90	1.49	1.43	0.36	0.68	0.24
August 21	<u>0.81</u>	<u>1.48</u>	<u>1.27</u>	<u>0.45</u>	<u>0.71</u>	<u>0.31</u>
avg:	0.91	1.47	1.37	0.41	0.70	0.28

8.7.2 Diggability Indices - Polycorder Data.

This section will present the diggability indices that have been determined per day of monitoring based upon the Polycorder data records. Recall that the Polycorder was the more extensively used instrument; 19 days of shovel monitoring versus only 10 days of monitoring with the G.E. monitor.

In analyzing the data recorded by the Polycorder the following performance parameters, determined by the KSHOVEL programme, have been considered:

1. Dig time (seconds).
2. Hoist armature voltage.
3. Hoist armature current.
4. Hoist current signal length.
5. Percentage of hoist current readings > 1100 amps.

6. Hoist voltage standard deviation.
7. Hoist current standard deviation.
8. Crowd armature voltage.
9. Crowd armature current.
10. Crowd voltage standard deviation.
11. Crowd current standard deviation.
12. Hoist power.
13. Crowd power.

As with the General Electric data, it was concluded that only the hoist motor responses are reflective of changes in muckpile diggability. Digging indices have been calculated based on the Polycorder data. Unfortunately, the slower sampling rate of the Polycorder (1/8 th that of the G.E. monitor) has generated data which is considerably less responsive to variations in digging effort. This has had the effect of damping, by essentially filtering or averaging out the rapid "fluctuations" observed in the G.E. based data, that helped define clear distinctions between ranges of digging effort from easy to very difficult. Since the diggability equations are heavily weighted by signal length, it is not surprising that with the "damped" Polycorder data, the definition of as clear a spectrum of digging conditions has not been possible. Table 8.7.2 list the calculated diggability indices associated with Polycorder monitoring. As with the G.E. data the results have been grouped according to daily operator assessments of diggability.

The lack of range in the Polycorder derived indices makes the above grouping less confident than was possible with the G.E. data. Still, however, it is reassuring that the broadest differences in diggability identified within the G.E. based DI (hoist based) indices are repeated within the Polycorder derived DI indices. Data recorded during the easiest digging conditions; the heavily blasted zone encountered in blast EM#1 on August 23rd, have the lowest DI. The most difficult digging as defined by the Polycorder DI's (0.34), is coincident with that defined by

the G.E. based DI for August 21st.

The comparison of calculated diggability indices with elements of blast design and muckpile characteristics is discussed in greater detail in section 8.13.

Again, the crowd based digging indices (CDI, CVDI and CCDI) show little relation to the nature of digging. Their values from easy to difficult remain relatively constant, reinforcing the view that they are of little benefit to the current research.

Table 8.7.2, Summary of Diggability Indices According to Ratings of Diggability - Polycorder Data.

<u>EASY DIGGING</u>	<u>HVDI</u>	<u>HCDI</u>	<u>DI</u>	<u>CVDI</u>	<u>CCDI</u>	<u>CDI</u>
August 20	0.45	0.49	0.23 (A)	0.28	0.71	0.20
August 23	0.48	0.36	0.18	0.48	0.67	0.33
October 14	0.41	0.44	0.19	0.33	0.62	0.20
avg:	0.45	0.43	0.20	0.36	0.67	0.24
<u>AVERAGE DIGGING</u>						
August 17	0.58	0.53	0.32 (D)	0.43	0.57	0.24
August 19	0.52	0.50	0.27	0.35	0.71	0.25
August 22	0.49	0.43	0.23	0.49	0.67	0.32
August 24	0.46	0.46	0.22	0.35	0.60	0.21
August 25	0.49	0.40	0.19 (E)	0.40	0.52	0.20
October 8	0.49	0.45	0.23	0.32	0.73	0.24
October 9	0.52	0.39	0.22	0.50	0.71	0.36
October 11	0.51	0.47	0.24	0.42	0.70	0.28
October 12	0.48	0.51	0.24	0.33	0.73	0.24
October 26	0.48	0.41	0.21	0.41	0.71	0.29
October 27	0.46	0.50	0.24	0.33	0.70	0.13
October 28	0.45	0.47	0.24	0.35	0.65	0.24
October 29	0.52	0.47	0.25	0.33	0.73	0.24
avg:	0.50	0.46	0.24	0.39	0.67	0.25
<u>DIFFICULT DIGGING</u>						
August 18	0.56	0.55	0.32	0.36	0.68	0.24
August 21	0.57	0.56	0.34	0.45	0.71	0.31
avg:	0.56	0.55	0.33	0.40	0.69	0.27

NOTE: an (E), (A) or (D) beside a DI denotes it's suggested inclusion in an alternate category, either E-easy digging, A-average digging or D-difficult digging.

8.8 ANALYSIS OF G.E. SHOVEL PRODUCTION DATA.

This section will discuss the processing and analysis of the production data collected by the General Electric monitor. When used in its conventional capacity the G.E. monitor prints out the following data; load time, fill time, swing time, delay time, and propel time as well as the load weight for each cycle. These values for each cycle during the study were manually entered into a database from which the summary statistics per day of monitoring presented in Table 8.8.1 were generated. It is to be noted that no actual dig time is recorded by the G.E. monitor. Rather, values of load and fill time are considered. Load time represents the time during which the crowd arm of the shovel is lower than 35 degrees w.r.t. the horizontal. Fill time is the interval during which the hoist motor is drawing greater than 100 volts. Dig time can therefore be regarded as lying somewhere between fill and load times. Note also that the total study time per day of monitoring is missing an average of 60 minutes, from the four hours spent on the shovel. This "lost study time" corresponds to the combined duration of the six daily performance monitoring studies, each approximately 10 minutes long. During these intervals the monitor is dedicated to sensor data collection and is unable to collect conventional production data.

8.8.1 Analysis of Dipper Load Weights.

The most significant data presented in Table 8.8.1 is considered to be the average load weight associated with each day of monitoring with the G.E. system. It is this data that has permitted relating machine digging effort with actual measures of productivity for varying degrees of digging difficulty.

Virtually all commercial shovel monitoring systems are marketed as capable of providing the user with data on dipper load weights (Radomilovich, 1986, LaPrairie, 1985, P&H Tech. Bull. "P&H Load Monitor", 1986 and General Electric, SPM-8000 operations manual, 1987). The experience gained during the course

DATE OPERATOR No.		AUG17 1	AUG18 2	AUG19 2	AUG20 3	AUG21 3	AUG22 4	AUG23 4	AUG24 1	AUG25 1
TOTAL STUDY	(min.)	92.40	161.70	159.20	135.70	162.90	161.40	162.10	158.60	122.60
TOTAL LOADING	(min.)	42.70	68.30	62.30	62.30	73.40	64.20	66.70	67.20	50.70
TOTAL FILLING	(min.)	20.07	36.70	31.00	29.50	37.70	32.10	32.50	32.40	26.50
TOTAL SWINGING	(min.)	28.60	55.80	55.30	42.70	47.40	45.60	51.90	47.80	35.60
TOTAL DUMPING	(min.)	8.60	15.60	14.70	15.30	16.30	18.40	16.20	12.90	9.20
TOTAL DELAY	(min.)	10.40	18.70	22.40	10.00	15.10	27.30	21.70	27.20	23.30
TOTAL PROPEL	(min.)	2.10	3.30	4.70	5.40	10.60	5.80	5.60	3.50	3.80
TOTAL DEL+PPL	(min.)	12.5	22	27.1	15.4	25.7	33.1	27.3	30.7	27.1
TIME LOADING	%	46.20	42.20	39.10	46.20	45.10	39.50	41.20	42.40	40.80
TIME FILLING	%	22.40	22.70	19.50	21.90	23.10	19.70	20.00	20.40	21.30
TIME SWINGING	%	30.92	34.50	34.70	31.70	29.10	28.10	32.00	30.20	28.60
TIME DUMPING	%	9.35	9.60	9.20	11.40	10.00	11.30	10.00	8.20	8.80
TIME DELAY	%	11.29	11.60	14.00	7.40	9.30	16.80	13.40	17.10	18.70
TIME PROPEL	%	2.29	2.00	2.90	4.00	6.50	3.60	3.50	2.20	3.10
AVG. CYCLE TIME	(sec.)	37.00	36.10	36.10	32.30	35.90	36.50	34.30	40.70	43.80
AVG. LOAD TIME	(sec.)	17.10	15.20	14.10	14.80	16.20	14.70	14.10	17.20	18.10
AVG. FILL TIME	(sec.)	8.30	8.20	7.00	7.20	8.30	7.30	6.90	8.30	9.50
AVG. SWING TIME	(sec.)	11.40	12.50	12.50	10.20	10.40	10.30	11.00	12.30	12.70
AVG. DUMP TIME	(sec.)	3.50	3.50	3.30	3.60	3.60	4.20	3.40	3.30	3.30
AVG. SWING ANGLE	(sec.)	53.60	60.50	63.80	50.20	49.50	53.80	58.60	55.30	63.50
No. OF DELAYS		46.00	111.00	79.00	70.00	58.00	51.00	54.00	49.00	43.00
AVG. DELAY TIME	(sec.)	13.60	10.10	17.00	8.50	15.70	29.58	24.10	33.30	32.50
No. OF PROPELS		9.00	15.00	18.00	13.00	16.00	6.00	3.00	13.00	10.00
AVG. PROPEL TIME	(sec.)	14.10	13.00	15.50	24.80	39.70	58.30	112.30	16.10	22.90
AVG WEIGHT/LOAD	(Tons)	36.60	39.00	39.50	40.60	37.20	43.30	44.20	37.60	38.50
No. OF LOADS		150.00	269.00	265.00	252.00	272.00	265.00	284.00	234.00	168.00
TOTAL WT. MOVED	(Tons)	5485.58	10489.80	10467.10	10234.50	10128.40	11465.84	12557.00	8803.20	6470.00
No. OF TRUCKS		37.50	67.25	66.25	63.00	68.00	66.25	71.00	58.50	42.00
AVG. LOAD/TRUCK	(Tons)	146.28	155.98	157.99	162.50	148.95	173.07	176.86	150.48	154.05
PRODUCTION	(T/min.)	59.34	64.88	65.73	75.40	62.19	70.53	77.44	55.51	52.77
AVG. TRK LD TIME	(min.)	2.47	2.40	2.40	2.20	2.40	2.44	2.28	2.71	2.92
PRODUCTION LESS OLYS+PPL	(T/min.)	68.65	75.09	79.24	85.07	73.82	89.37	93.15	68.83	67.75
AVG. TRK LD TIME	(min.)	2.13	2.08	1.99	1.91	2.02	1.94	1.90	2.19	2.27
PRODUCTION T/min LOADING		128.47	153.58	168.01	164.28	137.99	178.60	188.26	131.00	127.61

Table 8.8.1 - Summary statistics of daily shovel production.

of the present research however, has revealed that no presently available commercial package, is able to provide reliable, consistently accurate measures of dipper load weights.

Fording Coal Ltd. conducted year long evaluations of three shovel monitoring systems; Bucyrus Erie's BEPAL, P&H's Dig Mate system and General Electric's SPM-8000. Of the three, the G.E. monitor came the closest to satisfying the mine's requirement for dipper load weight measurements $\pm 2\%$, 95% of the time. The systems manufactured by Bucyrus Erie and P&H failed early on in the trials. Ultimately, even the G.E. monitor was rejected by the mine.

The accuracy of each system was determined by comparing the "actual" truck load weight (from truck weigh scales) against the combined weight of each dipper load (calculated by the monitoring systems) placed in the weighed truck.

The process by which each monitoring package calculates load weights is proprietary. Common to the approach of all systems however, is the use of the hoist armature current signal as the root or primary measure of the suspended load weight. The weight is based upon multiple readings of this signal taken during swinging, at **constant** acceleration, to the truck. Further discussion of the mechanics of load weight measure is beyond the scope of this thesis. In the interest of justifying the use of load weights in the current research the following discussion is offered in support of their legitimacy.

Although the G.E. monitor failed to satisfy the strict criterion set by the mine, the inclusion of load weights in a relative analysis of shovel performance/productivity is warranted on the basis of the following: Firstly, when considered over a large time frame the variability in calculated weights will become increasingly less significant. The tests undertaken at the mine to evaluate the accuracy of the G.E. monitor were based on

very small sample sets. The results of the most comprehensive trial are presented in Figure 8.8.1, in which the predicted and actual load weights for 19 trucks are presented. It is to be noted that during this test, the system was being continually recalibrated (ie. the weight of the empty dipper was being played with in order to increase the calculated load weight). It is still evident however, that the predicted weights do in fact correlate quite well with the actual weights. Following this trial it was generally concluded that the G.E. determined load weights were under, by an average 5 tons per truck.

G.E. Monitor Load and Truck Scale WTS

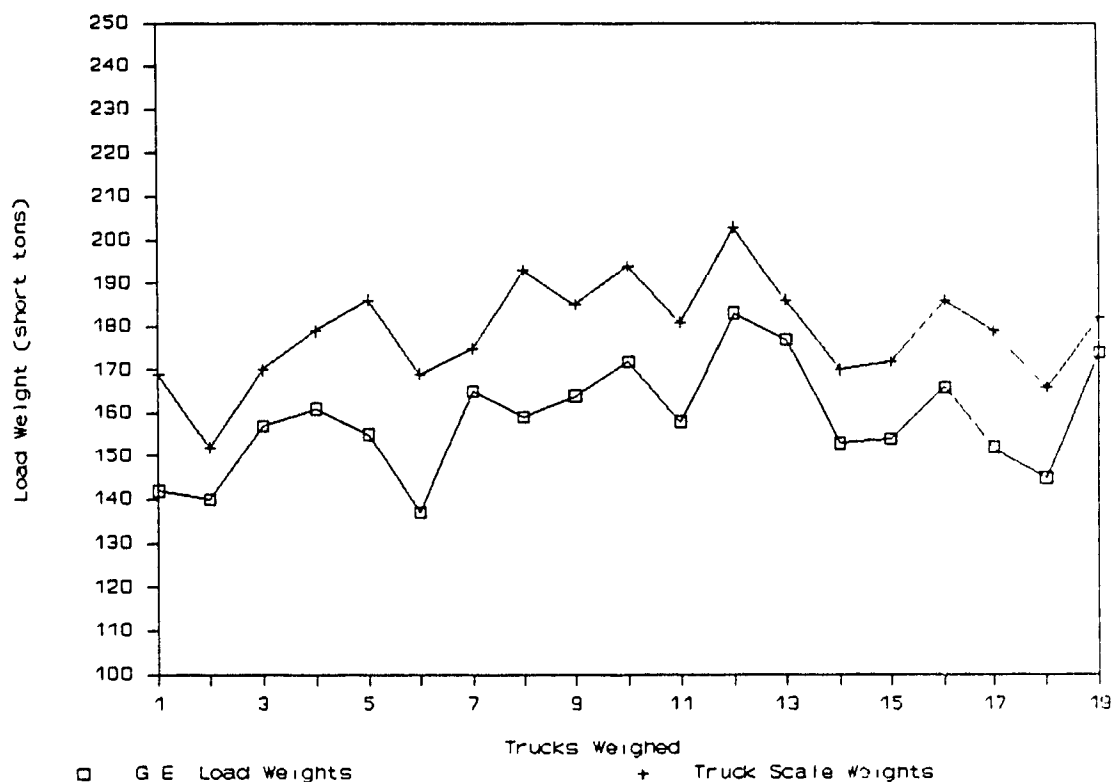


Figure 8.8.1 - G.E. monitor determined load weights and weights determined by truck scales.

Although the values for true and predicted weights do not coincide, the G.E. determined weights present a reasonable reflection of the true weights. Hence, while the G.E. weights

may not be true, they remained equally untrue throughout the study since no further attempts at recalibration were attempted. They provide therefore, a reliable, relative indication of the weights of material being moved per dipper pass, and in this respect, reflect average dipper fill factors for each day of monitoring. Although not listed in Table 8.8.1, standard deviations of load weights were calculated and found to be consistent at between 5 to 6 tons, throughout the monitoring period. This further suggests the acceptance of load weights as valid indicators of shovel productivity.

When average dipper load weights are considered in conjunction with load times, it is possible to gain an actual measure of shovel productivity per day. The final five rows of Table 8.8.1 are calculations of productivity, as determined by the number of tons of material moved per minute of the study, and the average load time per truck (based on four loads per truck). These values are then calculated per minute of study time less delays and propel episodes. In this manner external influences such as truck availability and spotting time, which could otherwise compromise assessments of diggability, are removed from consideration. Finally, productivity in terms of minutes of actual loading is determined.

The highest daily average load weights 43.3 and 44.2 tons and productivities 178.6 and 188.26 tons per minute of loading time, are associated with August the 22nd and 23rd respectively. The high load weights and productivity on the 23rd were expected, since on this day the shovel was excavating the heavily blasted zone in blast EM#1. The high loads and productivity recorded on the 22nd cannot be explained by the same argument since the shovel had not yet fully entered into the heavily blasted zone. This is confirmed by the diggability indices (DI) calculated for each day; 0.43 on the 23rd versus 0.99 on the 22nd. It is very noteworthy however, that the highest shovel productivities were recorded by the same operator on two days characterized by

varying levels of digging difficulty. This raises the following fundamental concerns: Does variability in operating practices have a more pronounced influence over shovel performance than originally thought? Can this variability manifest itself in the form of higher or lower productivities? Furthermore, do such variations transcend boundaries of digging difficulty? For example will a good operator maintain a high productivity irrespective of digging conditions, as is suggested by the production statistics associated with operator number 4 in Table 8.8.1.

With regard to the role of operator variability the prior works in shovel instrumentation studies arrived at the following conclusions.

Williamson et al., 1983: With regard to the crowd based diggability index (equivalent to the present CVDI) "Preliminary testing of the index has shown it to be relatively independent of operator characteristics and to be a good indicator of digging conditions".

Mol et al., 1986: "The shovel digging performance variation due to operator has not been found to be very significant as, in general, the digging conditions dominate the result"

The above suggest that measures of digging effort as calculated by diggability indices show little variation from one operator to another. Theoretically, this conclusion can only be substantiated by measuring the performance of two or more operators in the same digging environment. Each of the prior research efforts has monitored the performance of several shovels (including different models equipped with various size dippers) each in separate environments characterized by unique geology and blast designs. It is considered impossible therefore, based on the design of past experiments, to conclude that operator performance is truly insignificant. This especially, when no quantifiable measure of operator practice is available upon which to base comparisons. The current research has available to it, data on measures of productivity and dipper trajectory. The

remainder of this chapter is therefore, dedicated to an analysis of this data and how it relates to quantifying/qualifying the influence of dipper trajectory and operator variability, in assessments of muckpile diggability.

8.9 THE INFLUENCE OF DIPPER TRAJECTORY.

Analysis has indicated that the dipper trajectory is a fundamental consideration in evaluating muckpile diggability. The resolution of digging forces from either hoist or crowd motors can only be accomplished through a knowledge of dipper position in the muckpile at all points during the dig cycle. It is impossible to fully interpret shovel performance solely as a function of hoist or crowd motor signals, since the response of these motors are more heavily influenced by the position of the dipper in the bank than by material characteristics. For example, hard digging can be manifested as easy if a short shallow pass of the dipper is taken through the material. Similarly, in easy digging conditions, if excessive crowd is applied the ensuing motor responses can be approximate to those typical of hard digging conditions.

Figure 8.9.1 is a plot of hoist armature voltage against what has been termed the cut ratio (the proportion of crowd to hoist motion per dig cycle). Shown are the average voltage values for all operators across all digging conditions monitored. Clearly the response of the hoist motor is, to a significant degree, linked to the depth of the cut taken through the muckpile. Low depths of muckpile penetration are associated with high armature voltages while increasingly deeper cuts result in lower voltages for the dig cycle.

Figure 8.9.2 is a plot of average hoist armature current per dig cycle against cut ratio. Again, this signal is seen to be related to the depth of cut for each dig cycle. Dig cycles characterized by deep paths through the muckpile are associated with low average hoist armature currents.

Figures 8.9.3 and 8.9.4 are plots of crowd armature voltage and current respectively, versus cut ratio. As expected, based on the relative lack of crowd current responsiveness, there

HOIST ARMATURE VOLTAGE vs CUT RATIO

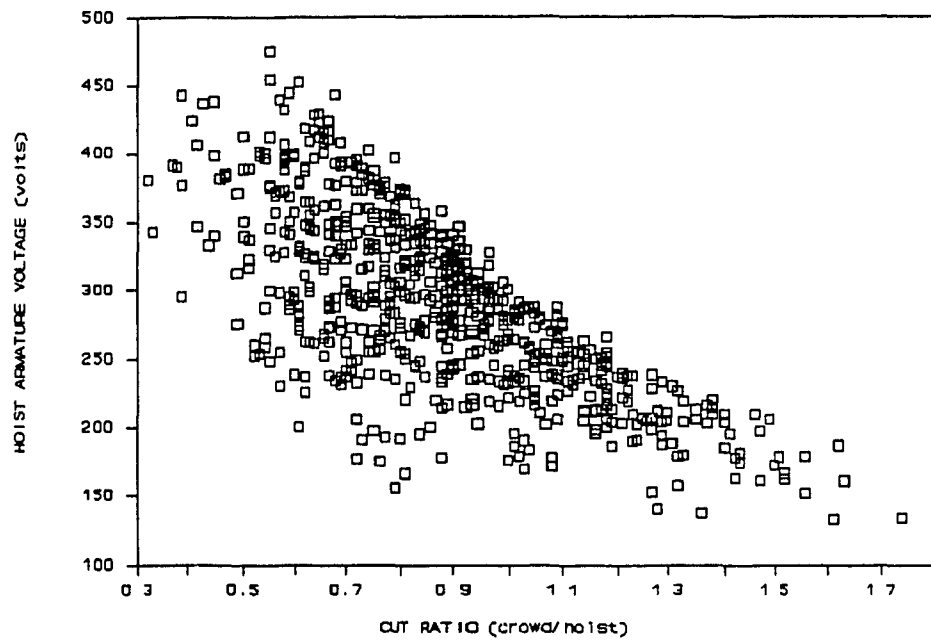


Figure 8.9.1 - Relation between depth of cut and hoist voltage.

HOIST ARMATURE CURRENT vs CUT RATIO

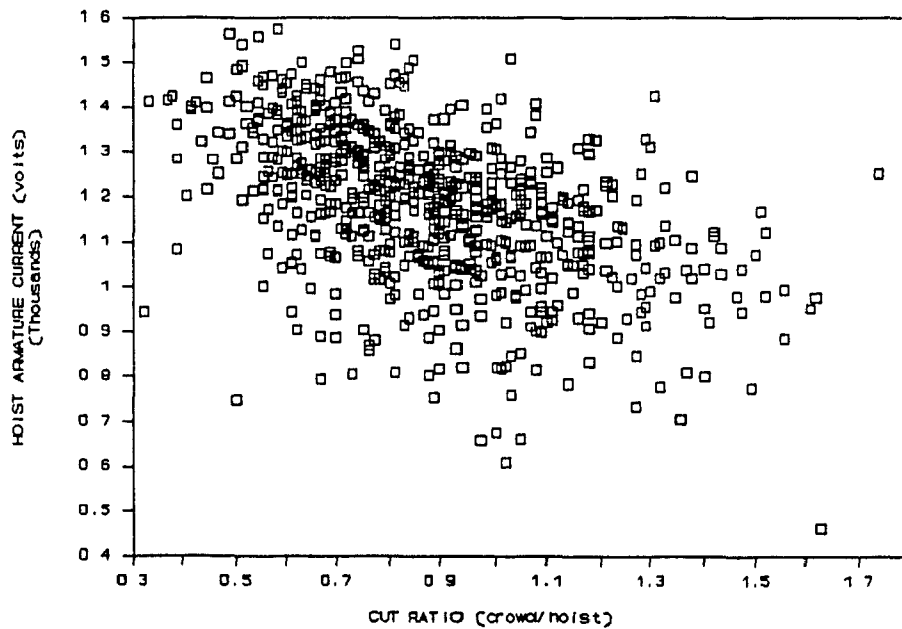


Figure 8.9.2 - Relation between depth of cut and hoist current.

CROWD ARMATURE VOLTAGE vs CUT RATIO

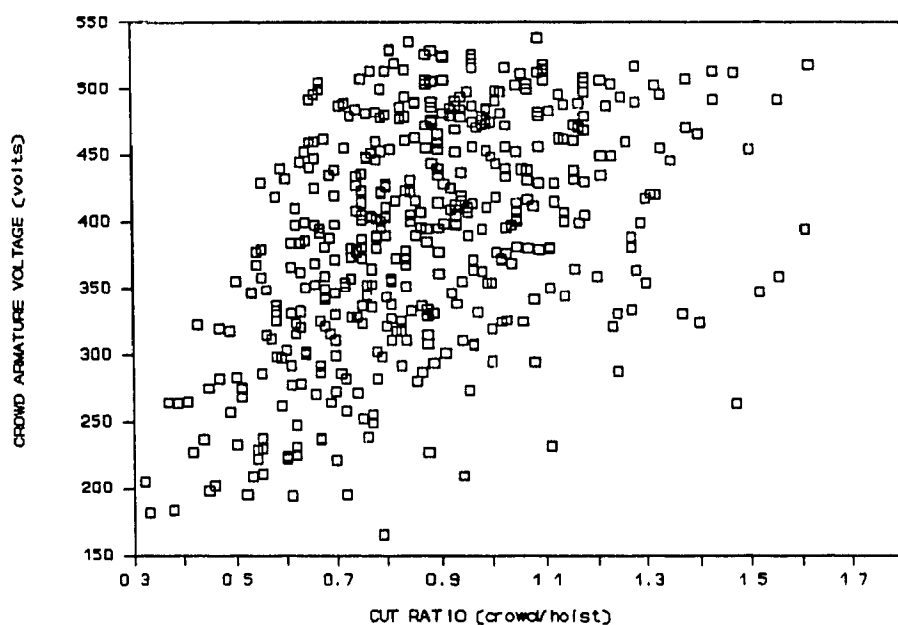


Figure 8.9.3 - Relation between depth of cut and crowd voltage.

CROWD ARMATURE CURRENT vs CUT RATIO

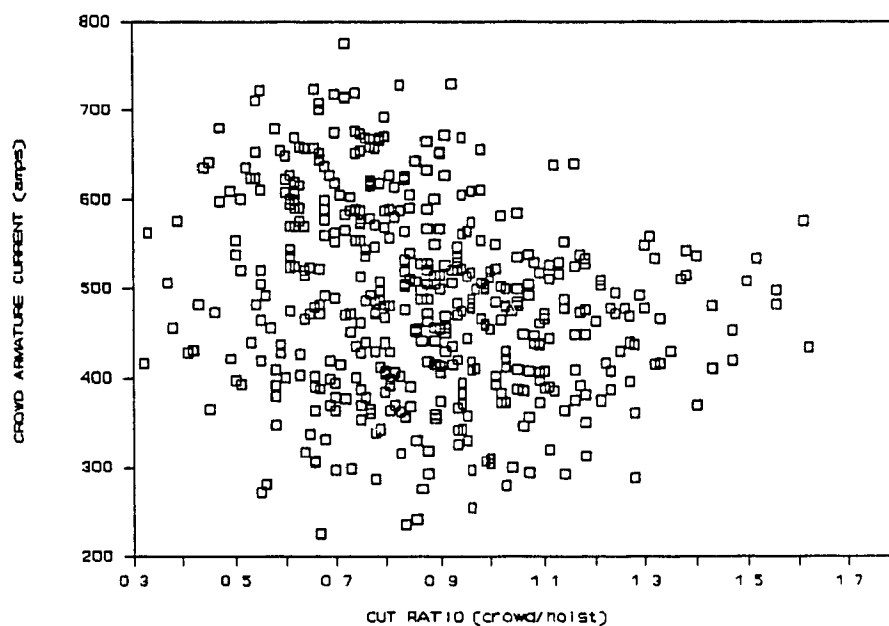


Figure 8.9.4 - Relation between depth of cut and crowd current.

exists no clear relation between depth of cut and average crowd current values. This further supports prior observations and claims by the shovel manufacturer (P&H) that it is hard to differentiate between crowd effort, as measured through crowd motor responses, for easy and hard digging conditions. The relationship between cut depth and crowd voltage was surprising. The highest average crowd voltages are associated with deep cut depths. This relationship is explained on the basis of the relative contribution by the crowd and hoist motor forces during the dig cycle. Recall that the hoist motor contributes more than 3.5 times as much power during the dig cycle than does the crowd motor. In low cut ratio dig cycles, the crowd arm travel is short. Conversely, in a high cut ratio pass, the crowd arm must travel much further. In such cases the crowd motor must rev at higher speed to keep up with hoist motion. The significance is that high crowd rack speeds are available irrespective of the depth of cut. It appears that hoist generated dipper momentum is great enough to facilitate unhindered crowd motion at all cut depths.

It is to be noted, that all data presented in this and other sections, refer to normal dig cycles - all ground preparation cycles have been removed from consideration. In ground preparation cycles, the dipper was usually contending with unbroken toe. In such instances, the crowd motor responses will not adhere to the behaviour described above; a high cut ratio will result in low crowd voltages as the dipper will tend to stall against the unbroken toe. Such cycles are not representative of normal operation, and have therefore been omitted from the analysis. Furthermore, since each incidence of hard toe was associated with either the operator working a few feet below design grade or the result of a misfire or too much fill at the bottom of a blasthole, they represented conditions generated as a result of influences external to the overall efficiency of the various blasts excavated. As such, the inclusion of ground preparation cycles, within the confines of

"this" investigation would only bias assessments of post-blast conditions from one blast design to another. If every dig cycle throughout the excavation of an "entire" blast were monitored, the inclusion of ground preparation cycles would be expected.

8.9.1 Relations Between Depth of Cut and Motor Parameters.

In light of the interdependence between motor responses and the depth of cut through the muckpile, the data has been classified on the basis of a set of "cut ratio classes". With the data grouped according to cut depth, it is possible to examine and contrast, the behaviour of the monitored shovel performance parameters and operators, within each of these classes. An inspection of all cut ratios suggested their grouping into five classes:

		Cut Ratio Range
Class 1 - crowd ft./hoist ft.	=	< 0.50
Class 2 - crowd ft./hoist ft.	=	0.50 - 0.75
Class 3 - crowd ft./hoist ft.	=	0.75 - 1.00
Class 4 - crowd ft./hoist ft.	=	1.00 - 1.25
Class 5 - crowd ft./hoist ft.	=	> 1.25

A - Hoist Voltage.

Table 8.9.1 documents the average responses for hoist voltage in each of the cut ratio classes. This data confirms the trends seen in actual performance traces of the hoist voltage signal. The highest average voltage values are associated with the shallowest cuts through the muckpile. As cut depth increases the average voltage per class is seen to grade progressively from a maximum average of 374 volts for class 1 to 191 volts for class 5. It is determined from the sample sizes for each class that 88% of the data falls within classes 2 (31%), 3 (37%) and 4 (20%), class 1 accounts for only 4% of the data and the final 8% is associated with class 5. The presence of classes 1 and 5 are justified in order to verify that the trends identified amongst classes 2, 3 and 4 are reflective of the upper and lower extremes

for cut depths. In other words, the trends are not solely restricted to extremes of cut depth, clear distinctions exist in the monitored parameters between the intermediary classes - 2, 3 and 4.

The data of Table 8.9.1 suggest that dispersion amongst values of hoist voltage decreases as cut depth increases. This is evidenced by correspondingly low standard deviations, ranges, lower quartiles¹, upper quartiles² and interquartile ranges³ for classes 4 and 5. These classes exhibit tight dispersion as a consequence of deep dipper penetration. Recall that the data for all classes is a composite of each operator's performance across all ranges of digging conditions. With sufficiently deep cuts however, the influence of other variables such as fragmentation, looseness and operating characteristics become increasingly less significant. This explains why the greatest dispersion of data is associated with classes 2 and 3, typical of normal trajectories. In such classes, variables external to cut depth have a greater influence over the monitored data.

B - Hoist Current.

Table 8.9.2 documents the summary statistics calculated for hoist armature current in each of the five cut ratio classes. The data clearly demonstrates that as cut depth increases the amount of hoist motor torque developed (proportional to hoist current amps) decreases, from a maximum average of 1325 amps for class 1 to 1021 for class 5. Again, a good spread in the average values between classes 2, 3 and 4 is apparent. The parallels between hoist current and voltage (ie decreasing values with increasing cut ratio) are explained on the basis of the hoist motor's speed

¹ Lower Quartile is a value less than which is 25% of the data.

² Upper Quartile is a value greater than which is 25% of the data.

³ Inter Quartile Range is a measure of the spread or dispersion between the lower and upper quartile values.

control, which acts to inhibit high torques at low motor speeds.

C - Crowd Voltage.

Table 8.9.3 presents the summary statistics calculated for crowd armature voltage across all five cut ratio classes. Recall from previous sections, that the Polycorder was not used as extensively as the G.E. monitor during monitoring periods in which both instruments were available. This explains why the sample sizes for crowd motor data are smaller than those associated with the G.E. data. The statistics confirm the observation based on Figure 8.9.3; average crowd voltage per dig cycle increases as cut depth increases. As was observed with hoist voltage, the least dispersion of data is associated with classes 4 and 5. Note that little variation is apparent in the crowd armature voltage data between classes 4 and 5. It is believed that for cut ratios greater than 1.0 the crowd motor operates at maximum speed. And as observed with the hoist data; the influence of deep cuts, dominates over motor responses.

Table 8.9.1 - Summary statistics; hoist voltage for each cut ratio class.

	<u>HOIST VOLTAGE (volts)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	373.70	330.92	292.11	239.13	191.16
Median:	381.14	333.37	293.77	239.22	197.82
Mode:	380.20	325.11	272.20	238.76	204.18
Variance:	886.66	3733.96	2041.39	843.33	726.27
Standard Dev:	43.44	61.11	45.18	29.04	26.95
Standard Err:	8.68	4.21	2.84	2.47	3.70
Minimum:	275.66	177.20	155.54	169.46	133.23
Maximum:	442.92	475.11	395.94	291.88	238.74
Range:	167.26	297.91	240.40	122.42	105.51
Lower Quartile:	342.83	287.36	266.35	219.11	174.14
Upper Quartile:	398.64	379.53	322.52	259.05	210.40
Int.Qrtl.Range:	55.81	92.17	56.17	39.94	36.26

Table 8.9.2 - Summary statistics; hoist Current for each cut ratio class.

	<u>HOIST CURRENT (amps)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	1314.66	1274.80	1167.59	1099.13	1021.15
Median:	1359.83	1286.05	1179.33	1111.96	1027.32
Mode:	1341.69	1285.08	1291.76	1110.96	1020.75
Variance:	30996.00	20383.20	22386.51	22393.70	28444.10
Standard Dev:	176.07	142.77	149.62	149.64	168.65
Standard Err:	35.21	9.82	9.44	12.74	23.16
Minimum:	746.86	793.91	658.50	610.51	462.57
Maximum:	1562.76	1573.67	1539.29	1508.86	1423.96
Range:	815.90	779.76	880.79	898.35	961.39
Lower Quartile:	1281.94	1196.86	1071.68	1018.96	952.33
Upper Quartile:	1413.69	1371.68	1263.32	1194.16	1103.36
Int.Qrtl.Range:	131.75	174.82	191.64	175.20	151.03

Table 8.9.3 - Summary statistics; Crowd voltage for each cut ratio class.

	<u>CROWD VOLTAGE (volts)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	19	134	184	90	31
Average:	256.55	348.73	410.99	432.61	428.80
Median:	264.14	350.29	413.46	438.69	446.10
Mode:	256.64	349.20	411.18	435.12	421.13
Variance:	2534.81	6113.43	5706.43	4049.21	4919.63
Standard Dev:	50.35	78.19	75.54	63.63	70.14
Standard Err:	11.55	6.75	5.57	6.71	12.60
Minimum:	182.31	194.44	165.78	232.14	263.89
Maximum:	355.60	507.57	535.36	538.22	517.79
Range:	173.29	313.13	369.58	306.08	253.91
Lower Quartile:	204.84	291.30	354.93	381.27	364.23
Upper Quartile:	282.71	401.43	476.70	488.50	492.15
Int.Qrtl.Range:	77.87	110.13	121.77	107.23	127.79

D - Crowd Current.

Table 8.9.4 below documents the crowd armature current readings for each of the five cut ratio classes. The data confirms the parameters lack of responsiveness, identified through inspection of performance monitoring traces. Crowd current exhibits an attenuated relation with depth of cut. While

average values are seen to decrease across the bulk of the data (classes 2, 3 and 4), the spread between classes is narrow and characterized by relatively high data dispersion. Therefore, little confidence is attached to any segregation and/or classification of crowd current responses on the basis of cut depth.

E - Percent of Hoist Current Readings > 1100 Amps.

Although not a direct measure of hoist motor response, this value provides insight into the digging effort and amount of hoist motor torque developed per dig cycle. It was recognized that in easy digging conditions the hoist current trace during digging was not only very stable, but characteristically high. A high percentage of hoist current readings in excess of 1100 amps in a dig cycle are common in such conditions. Conversely, hard digging conditions are typified by a low proportion of current readings greater than 1100 amps. Table 8.9.5 documents the relation between the "percent" and cut depth. As with the previous performance parameters, "percent" is also seen to vary significantly as a function of cut depth. As the cut ratio increases the percentage of readings greater than 1100 amps drops significantly, from a maximum average of 78% for class 1 to a minimum average of only 48% in class 5. It is noteworthy that the spread between classes is consistent at approximately 9% (with the exception of class 1).

Table 8.9.4 - Summary statistics; Crowd current for each cut ratio class.

	<u>CROWD CURRENT (amps)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	19	134	184	90	31
Average:	514.72	524.35	487.72	448.70	467.27
Median:	507.10	541.52	488.40	448.75	477.76
Mode:	482.87	526.27	529.09	448.19	468.72
Variance:	8757.79	14805.40	11243.5	5616.40	4236.70
Standard Dev:	93.58	121.67	106.04	74.94	65.09
Standard Err:	21.47	10.51	7.82	7.89	11.69
Minimum:	365.93	225.83	236.33	279.76	288.57
Maximum:	680.62	776.47	731.46	640.87	576.29
Range:	314.69	550.64	495.13	361.11	287.72
Lower Quartile:	428.51	419.33	411.59	390.00	419.16
Upper Quartile:	598.06	619.46	564.04	502.91	515.08
Int.Qrtl.Range:	169.55	200.13	152.44	112.91	95.92

Table 8.9.5 - Summary statistics; Percent of hoist current readings > 1100 amps per dig cycle.

	<u>PERCENT HOIST CURRENT > 1100 AMPS (%)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	78.54	73.56	64.13	55.53	48.30
Median:	83.67	75.82	66.27	57.15	47.78
Mode:	83.67	82.42	57.14	67.47	49.45
Variance:	341.70	162.66	178.35	170.72	158.68
Standard Dev:	18.48	12.75	13.35	13.06	12.60
Standard Err:	3.70	0.87	0.84	1.11	1.73
Minimum:	5.95	11.48	7.55	15.38	7.69
Maximum:	94.44	97.17	87.91	85.28	77.55
Range:	88.49	85.69	80.36	69.90	69.86
Lower Quartile:	77.08	67.58	57.14	48.53	41.86
Upper Quartile:	89.38	81.82	73.49	64.65	55.42
Int.Qrtl.Range:	12.30	14.24	16.35	16.12	13.56

F - Hoist current signal length.

It has already been demonstrated that the absolute length of the hoist current trace is proportional to the relative ease of excavation, long signal lengths being typical of hard digging conditions. Table 8.9.6 illustrates the relation between cut ratio and hoist current signal length. As cut depth increases so does the average hoist current signal length. There are significant differences in the average values for cut ratio classes 1, 2 and 3. It appears that for cut ratios in excess of 1.0 (ie classes 4 and 5) there is little increase in the average signal length.

Table 8.9.6 - Summary statistics; Hoist current signal length.

	<u>HOIST CURRENT SIGNAL LENGTH</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	7958	11445	13210	13363	13752
Median:	6542	10224	12075	12807	12283
Mode:	6399	10193	12070	12646	11692
Variance:	1.52E7	3.97E7	3.11E7	3.02E7	4.03E7
Standard Dev:	3901	6306	5580	5490	6344
Standard Err:	780	434	351	467	871
Minimum:	3117	2424	2731	5519	5095
Maximum:	19599	38438	39973	32069	30553
Range:	16482	36013	37242	26550	25458
Lower Quartile:	5217	9270	9270	9165	8401
Upper Quartile:	9837	16090	16090	16197	17050
Int.Qrtl.Range:	4620	6820	6821	7033	8649

Box and Whisker Plots.

The summary statistics presented in the preceding section are now presented graphically, in the form of box and whisker plots for each of the four motor parameters, Figures 8.9.1 to 8.9.6. These plots clearly communicate the relations established between motor responses and to depth of cut.

Box and Whisker plots facilitate the interpretation of summary statistics by allowing quick recognition of the extent of data dispersion within groups. Referring to the plots, the central "box" for each cut ratio class, covers the middle 50%

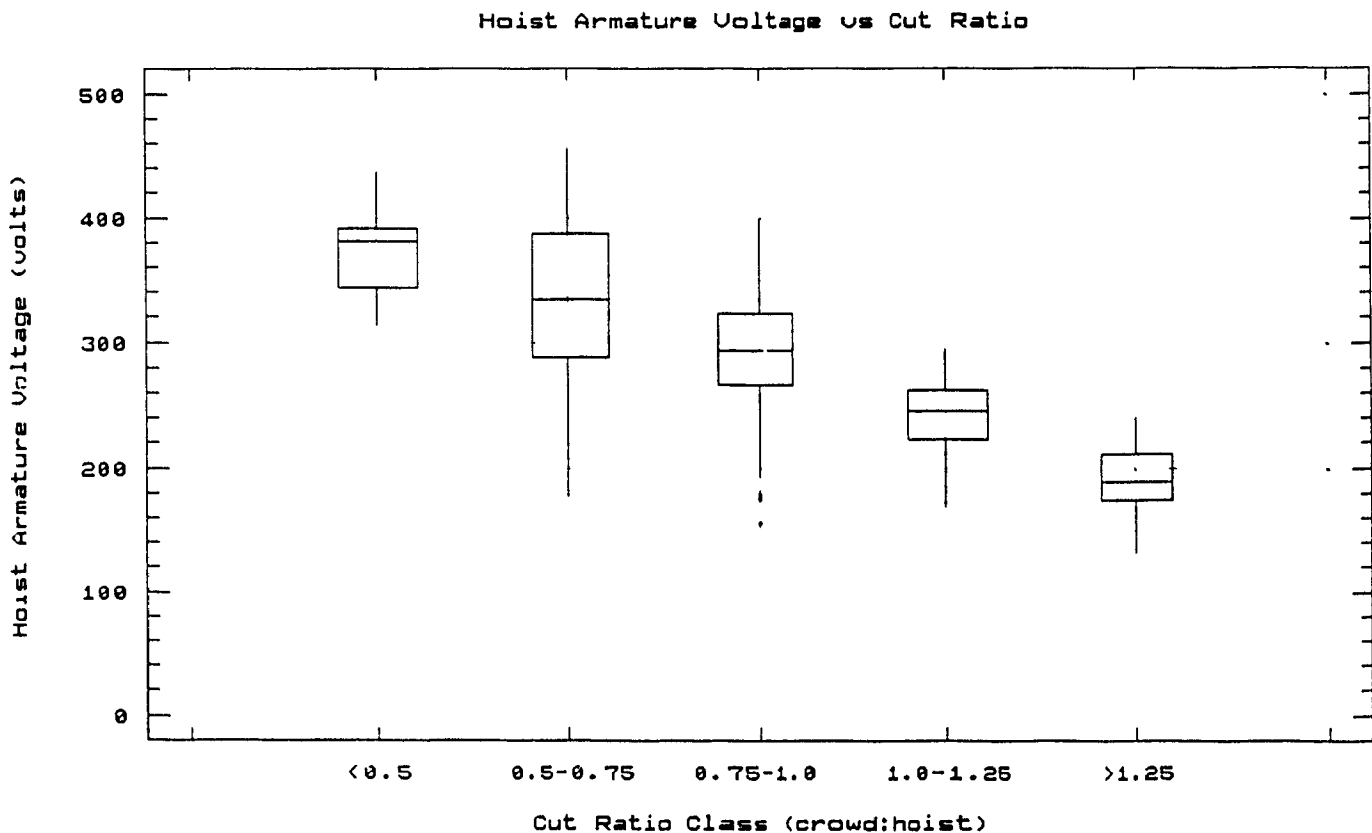


Figure 8.9.5 - Box and whisker plot of summary statistics in Table 8.9.1.

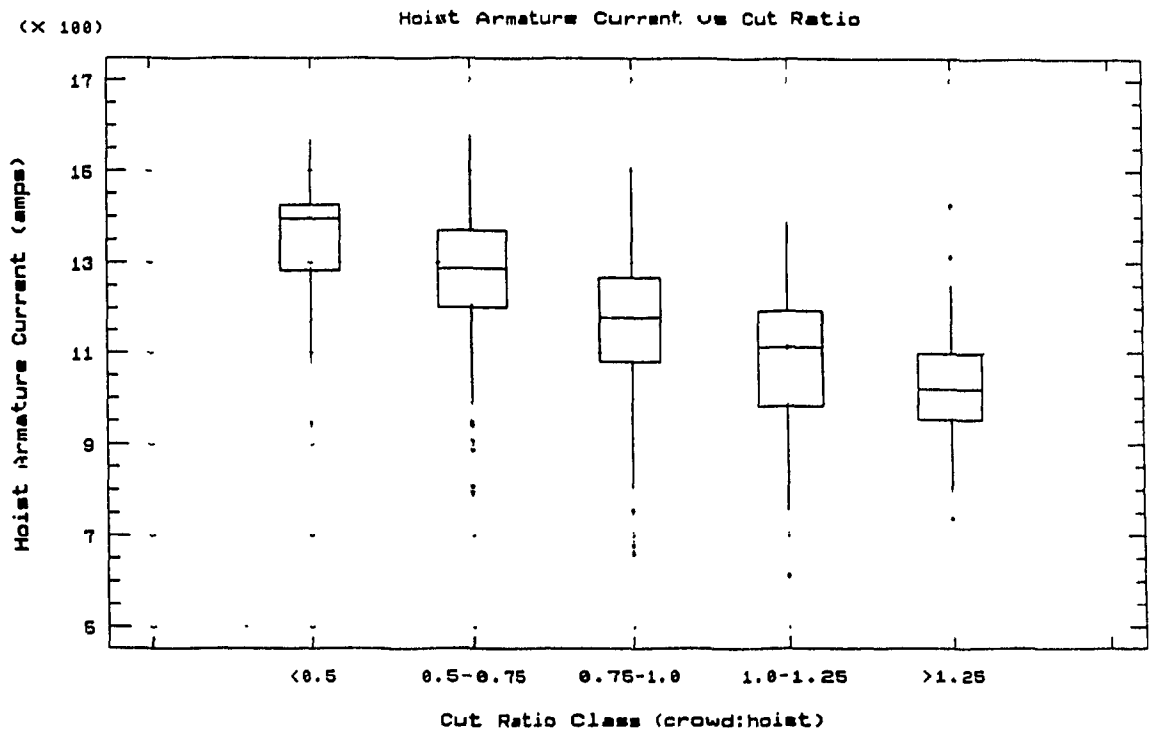


Figure 8.9.6 - Box and whisker plot of summary statistics in Table 8.9.2.

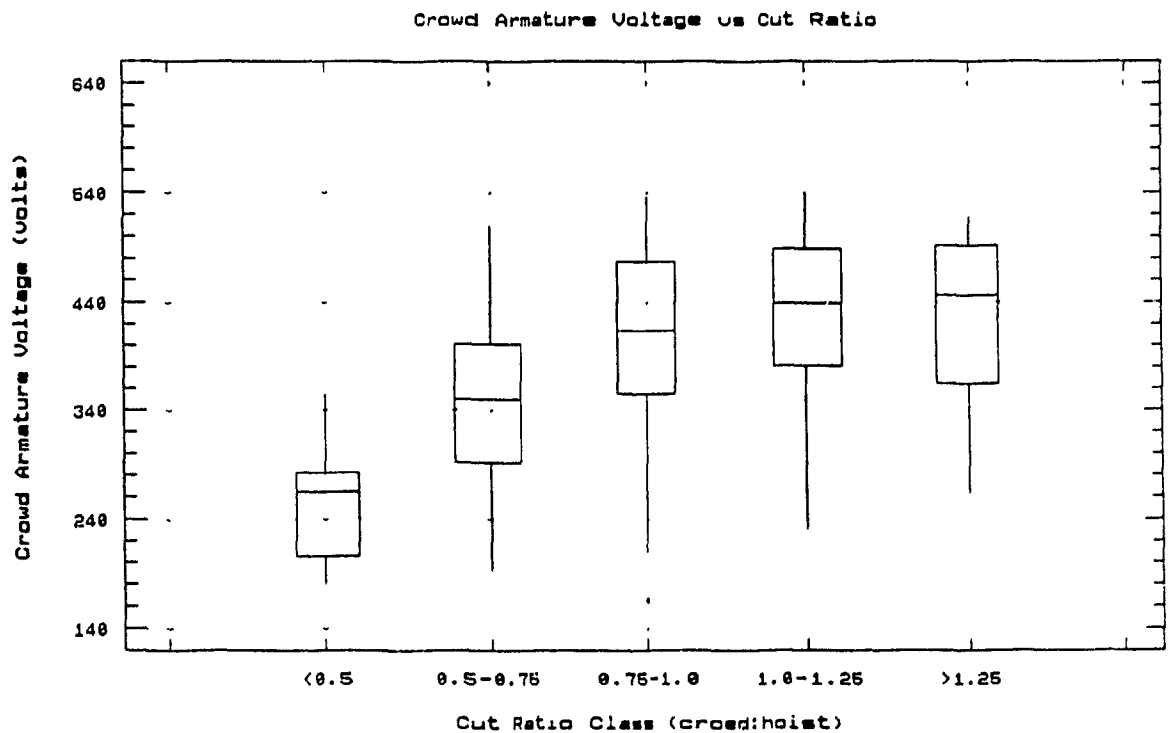


Figure 8.9.7 - Box and whisker plot of summary statistics in Table 8.9.3.

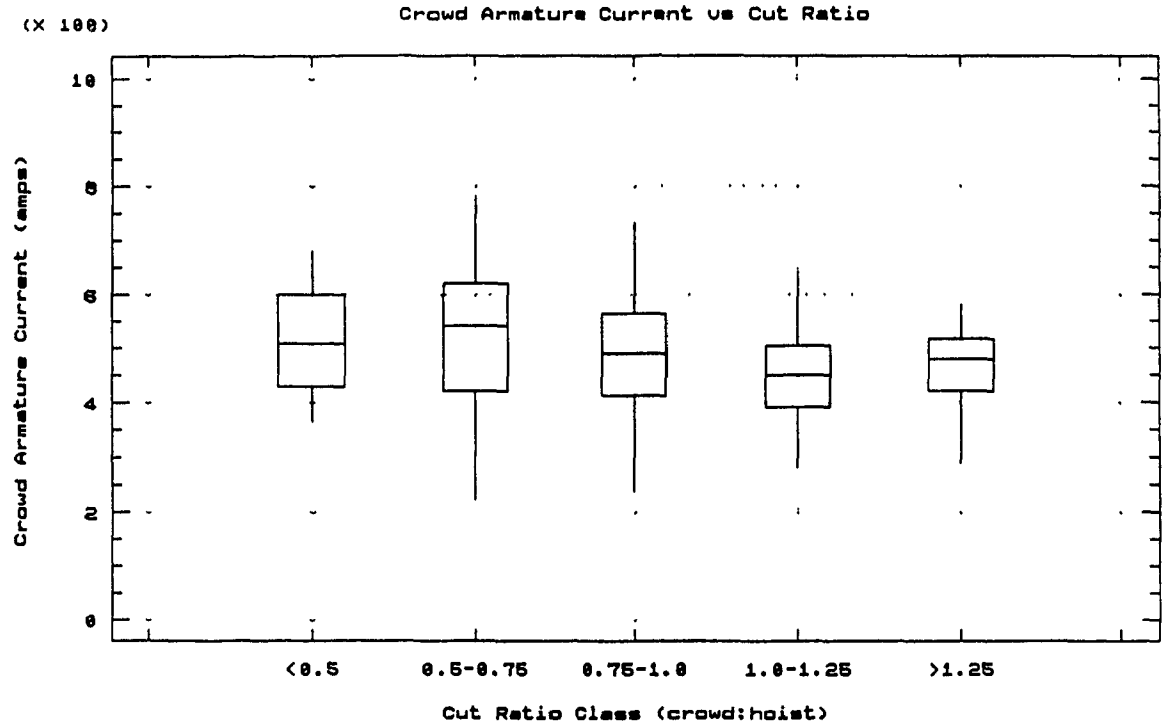


Figure 8.9.8 - Box and whisker plot of summary statistics in Table 8.9.4.

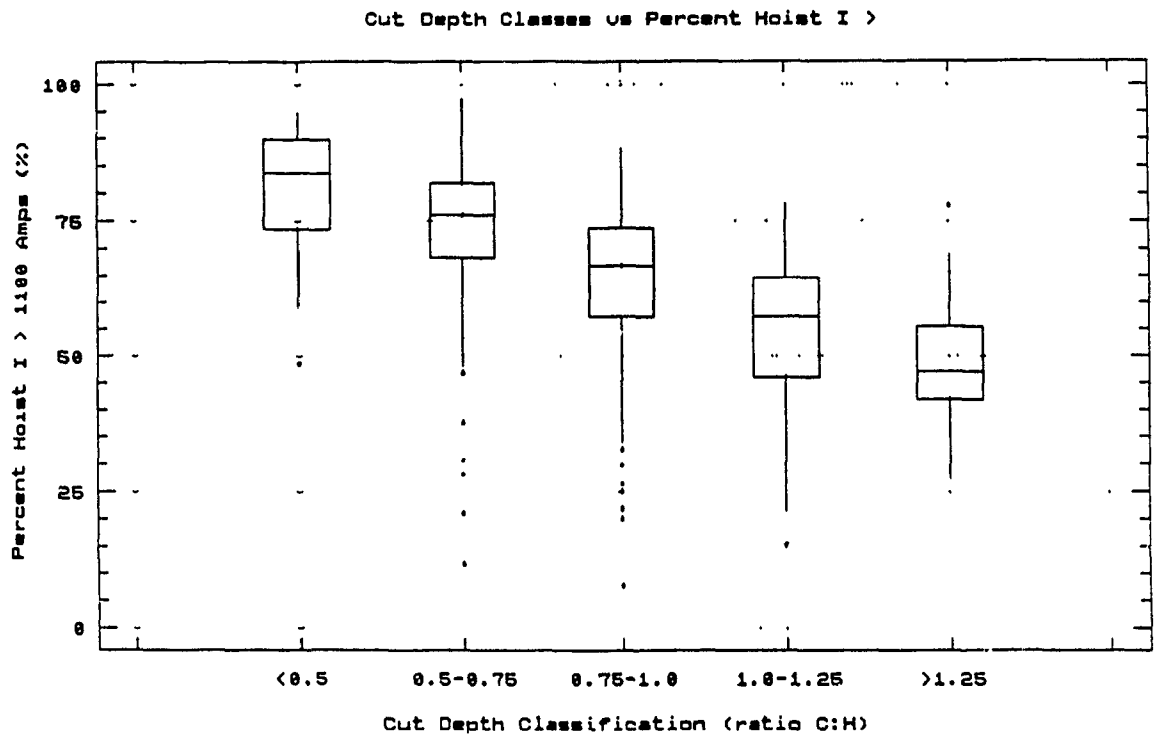


Figure 8.9.9 - Box and whisker plot of summary statistics in Table 8.9.5.

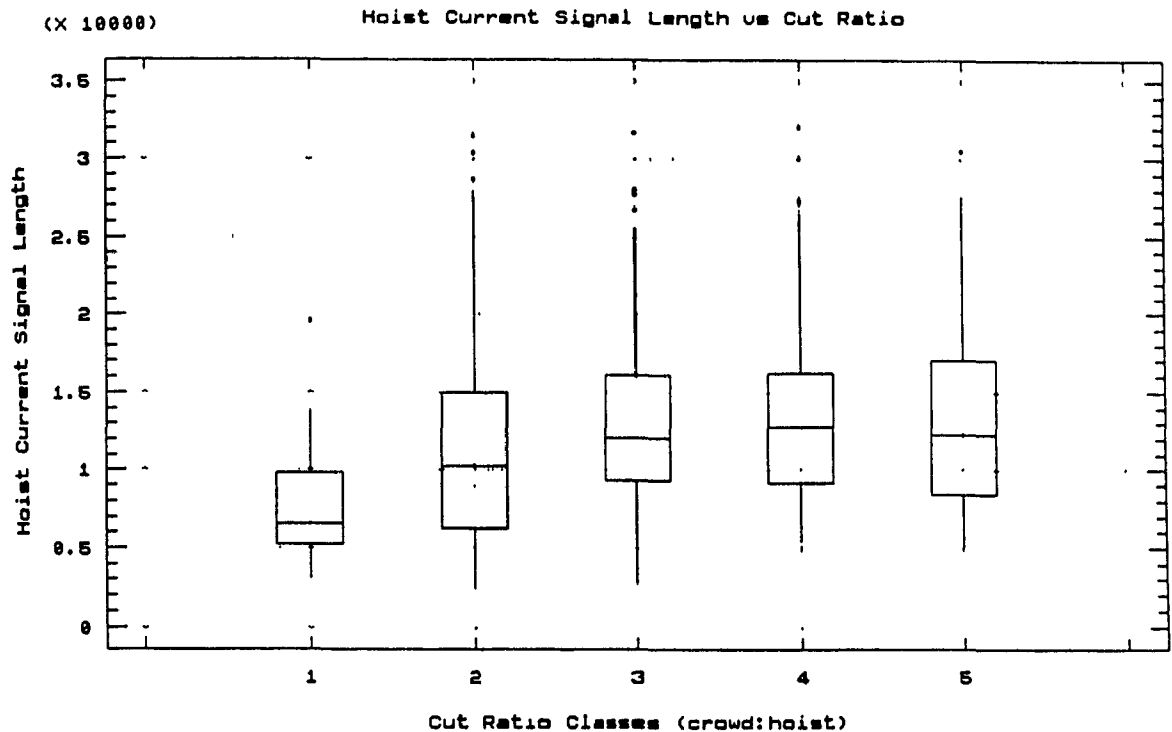


Figure 8.9.10 - Box and whisker plot of summary statistics in Table 8.9.6.

of the data values - the inter quartile range between upper and lower quartile values. The central line through the box is the median of the data. The lines extending from each box - "whiskers", define the upper and lower quartile ranges. The end points are the maximum and minimum values for the data group. Data values in excess of 1.5 times the interquartile range are plotted as separate points either above or below the whisker end points. The position of the median within the box, and length of whiskers, provide a visual measure of any data skewness (Statgraphics Manual, 1987).

8.9.2 Dipper Trajectories - Plots for Cut Ratio Classes 1 - 5.

Data on dipper position has been used to generate plots of actual dipper trajectories for each of the five cut ratio classes. Figure 8.9.11 illustrates the shape and average range of dipper trajectories associated with each of the five classes.

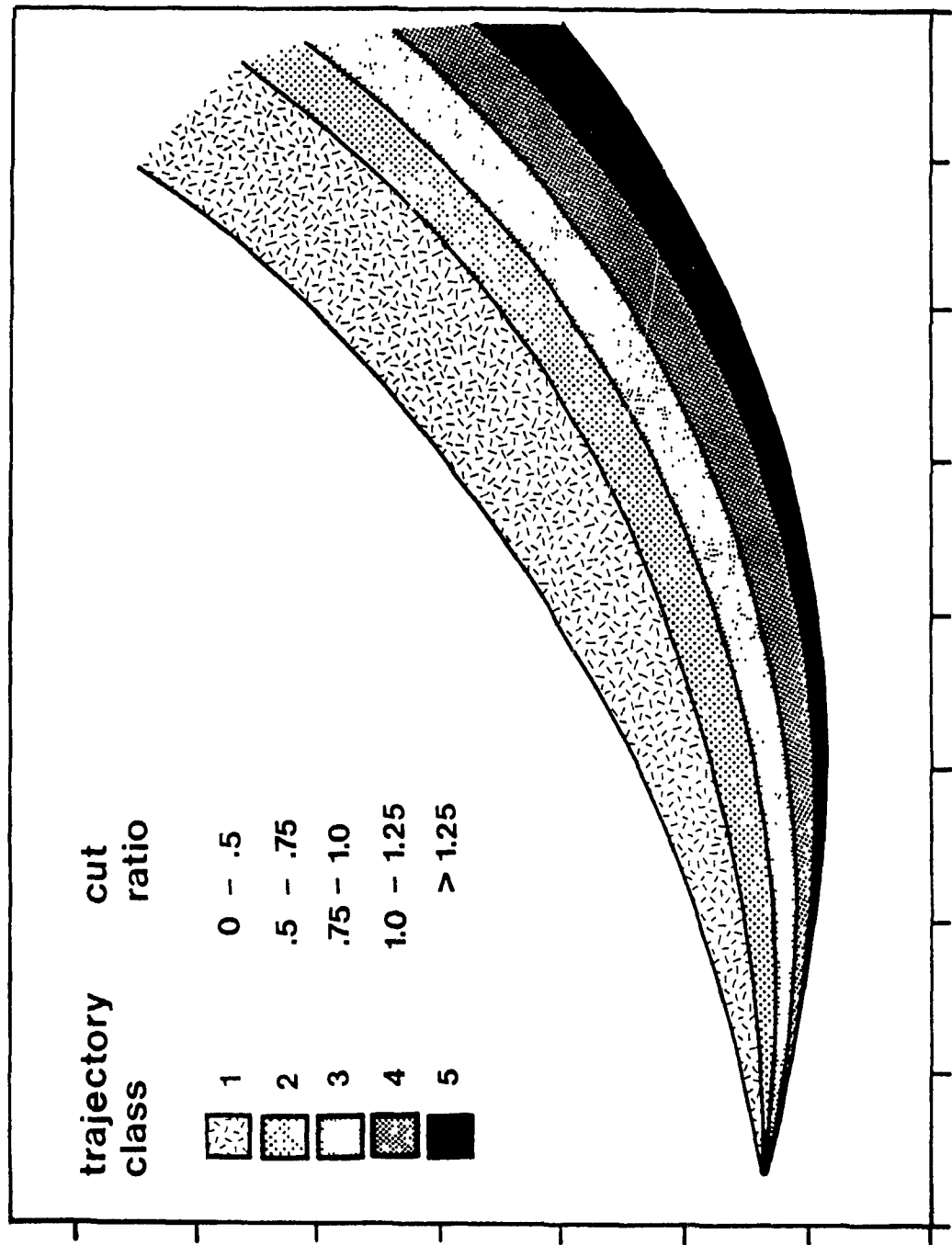


Figure 8.9.11 - Plots of trajectory ranges associated with cut ratio classes 1 to 5.

The upper left hand corner (0,0) of Figure 8.9.11 represents the crowd arm attach point on the shovel boom. These plots clearly illustrate the relation between cut ratio and the ensuing dipper trajectory shape. Note that high cut ratios characteristically result in relatively short trajectories, and low ratios have the longest trajectories. Summary statistics for trajectory length, per cut ratio class are presented in Table 8.9.7, a box and whisker plot of the data is given in Figure 8.9.12.

Also indicated in Table 8.9.7 below is the average crowd arm and hoist rope travel associated with each of the trajectory classes. It is evident that the only appreciable difference in crowd travel is between class 1 trajectories and the remaining classes. For practicality, crowd travel associated with classes 3, 4 and 5 is identical. This means the higher cut ratios associated with these classes result from a reduction in hoist travel. This suggests that operators will generally employ the same amount of crowd for most dig cycles.

Table 8.9.7 - Summary statistics; Trajectory length (feet).

	<u>TRAJECTORY LENGTH (feet)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	38.80	40.18	37.47	33.90	32.16
Median:	37.92	40.72	37.85	33.84	32.38
Mode:	37.28	36.69	35.28	26.62	31.38
Variance:	41.15	38.81	27.56	23.48	20.67
Standard Dev:	6.41	6.23	5.25	4.85	4.54
Standard Err:	1.28	0.43	0.33	0.41	0.62
Minimum:	27.62	26.54	25.07	24.06	22.27
Maximum:	54.46	56.47	50.71	46.67	42.63
Range:	29.84	29.93	25.64	22.61	20.36
Lower Quartile:	35.32	35.13	33.08	30.40	28.89
Upper Quartile:	41.72	44.91	41.71	37.29	34.88
Int.Qrtl.Range:	6.40	9.78	8.63	6.89	5.99
	<u>AVERAGE CROWD AND HOIST TRAVEL (feet)</u>				
Crowd travel:	10.11	13.25	14.32	14.13	14.41
Hoist travel:	23.16	20.45	16.47	12.81	10.43

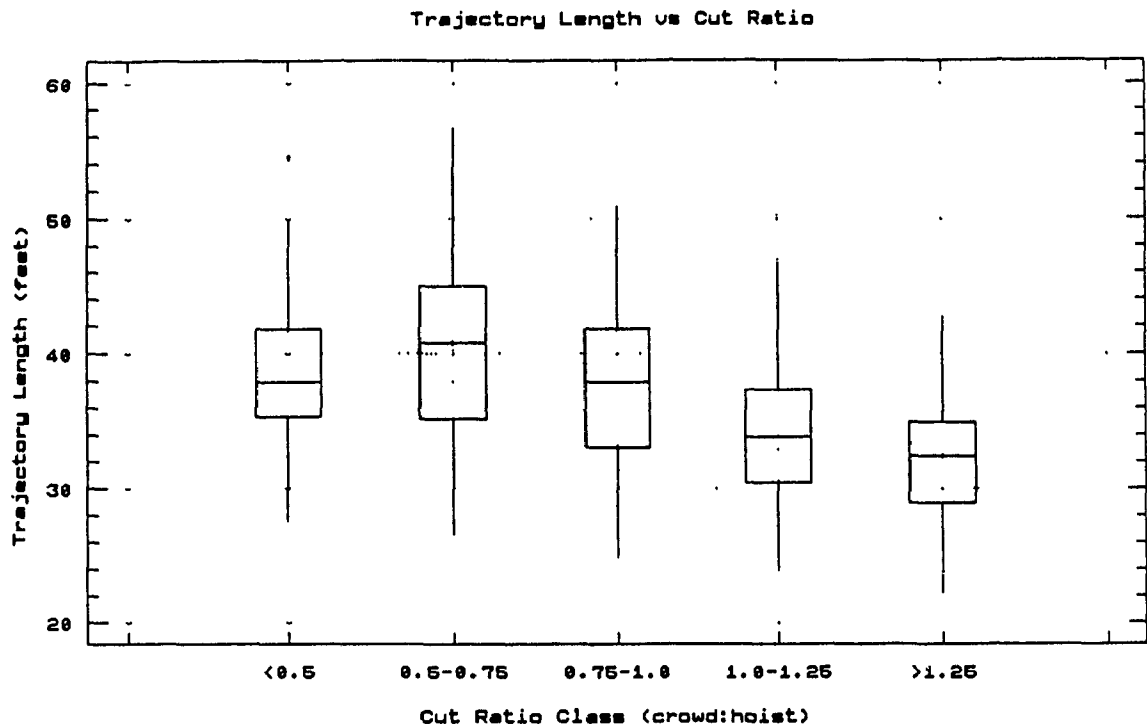


Figure 8.9.12 - Box and whisker plot of operator trajectory lengths

8.9.3 Correlation of Diggability Indices with Trajectory Classes.

Figure 8.9.13 is a plot of the average crowd voltage based diggability indices (CVDI) associated with each of the five cut ratio classes. Average index values along with standard deviations are given below in Table 8.9.8, for each trajectory class:

Table 8.9.8, Average crowd voltage diggability index (CVDI) values per cut ratio class.

	<u>CROWD VOLTAGE DIGGABILITY INDICES (CVDI)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	19	134	184	90	31
CVDI:	0.92	0.59	0.48	0.42	0.42
Standard Dev:	0.25	0.19	0.17	0.12	0.12

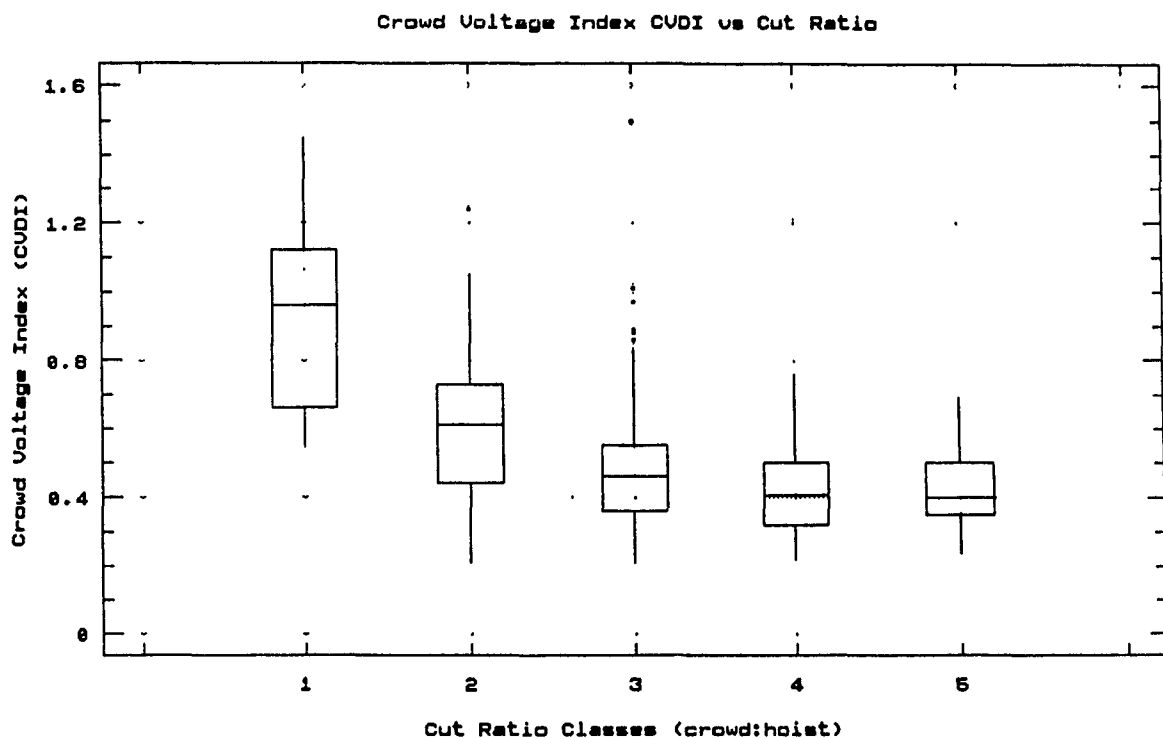


Figure 8.9.13 - Box and whisker plot of CVDI by cut ratio.

Clearly the highest average crowd diggability indices are associated with the shallowest cut depths. This observation counters the conclusions drawn from prior investigations that suggest the crowd motor to be responsive to (and correlate well with) levels of digging difficulty. The small correlation that does exist ie. classes 1 and 2 against classes 3, 4 and 5, exhibits an inverse relation with levels of digging difficulty. The greater the index, the easier the digging. This appears explainable on the basis of "operator available crowd motion". In classes 3, 4 and 5 the operator effectively exhausts all available crowd. Plots of crowd armature voltage against cut ratio presented in Figure 8.9.3 have shown that these classes are associated with the greatest average crowd rack speeds.

Conversely, in class 1 trajectories the operator maintains a store of available crowd motion. It is believed therefore, that the high index values associated with cut ratio class 1, arise from operator-induced crowd rack speed adjustments during the dig cycle. This would result in a more ragged crowd armature voltage trace (from minor increases and decreases in velocity), thereby increasing the signal length. This, coupled with lower velocities (ie. a lower area under the signal), would explain the surprisingly high index values for what is known to be easy digging associated with cut ratio classes 1 and 2.

Tables 8.9.9 and 8.9.10 below, present the summary statistics calculated for the hoist voltage (HVDI) and current (HCDI) based components of the diggability index (DI), for each of the five trajectory classes. It is evident that levels of digging effort as measured through hoist motor response, correlate well established levels of digging difficulty as defined by dipper trajectories - indices increase with cut depth. Table 8.9.11 documents the very good correlation that exist between the hoist based diggability index (DI) and trajectory class. Box and whisker plots of the summary statistics presented in Tables 8.9.9 to 8.9.11 are given in Figures 8.9.14 to 8.9.16.

Table 8.9.9 - Summary statistics; Hoist voltage based diggability index.

<u>HOIST VOLTAGE DIGGABILITY INDEX (HVDI)</u>					
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	0.420	0.561	0.700	0.820	0.975
Median:	0.400	0.530	0.660	0.780	0.920
Mode:	0.360	0.510	0.661	0.780	0.930
Variance:	0.019	0.044	0.053	0.045	0.067
Standard Dev:	0.138	0.208	0.229	0.213	0.258
Standard Err:	0.028	0.014	0.014	0.018	0.035
Minimum:	0.220	0.180	0.320	0.400	0.580
Maximum:	0.810	1.118	1.770	1.440	1.770
Range:	0.590	1.000	1.450	1.040	1.190
Lower Quartile:	0.330	0.410	0.540	0.670	0.760
Upper Quartile:	0.490	0.690	0.810	0.960	1.060
Int.Qrtl.Range:	0.160	0.280	0.290	0.290	0.300

Table 8.9.10 - Summary statistics; Hoist current based diggability index.

<u>HOIST CURRENT DIGGABILITY INDEX (HCDI)</u>					
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	0.684	0.907	1.321	1.526	1.664
Median:	0.590	0.850	1.230	1.410	1.510
Mode:	0.640	0.780	0.930	1.080	1.010
Variance:	0.112	0.149	0.262	0.321	0.463
Standard Dev:	0.335	0.387	0.512	0.567	0.679
Standard Err:	0.067	0.026	0.032	0.048	0.093
Minimum:	0.330	0.240	0.390	0.450	0.480
Maximum:	1.560	2.940	3.010	3.190	3.730
Range:	1.230	2.700	2.620	2.740	3.250
Lower Quartile:	0.430	0.660	0.970	1.090	1.180
Upper Quartile:	0.850	1.060	1.550	1.830	2.020
Int.Qrtl.Range:	0.420	0.400	0.580	0.740	0.840

Table 8.9.11 - Summary statistics; diggability index.

	<u>DIGGABILITY INDEX (DI)</u>				
	Class 1	Class 2	Class 3	Class 4	Class 5
Sample Size:	25	211	253	138	53
Average:	0.292	0.542	0.982	1.307	1.723
Median:	0.216	0.464	0.810	1.132	1.382
Mode:	0.205	0.264	0.454	1.111	1.060
Variance:	0.030	0.151	0.460	0.515	1.201
Standard Dev:	0.173	0.389	0.678	0.717	1.096
Standard Err:	0.035	0.027	0.043	0.061	0.151
Minimum:	0.073	0.076	0.187	0.351	0.336
Maximum:	0.764	3.469	4.921	3.541	4.991
Range:	0.692	3.393	4.733	3.189	4.655
Lower Quartile:	0.170	0.270	0.524	0.756	0.942
Upper Quartile:	0.408	0.682	1.173	1.698	2.162
Int.Qrtl.Range:	0.239	0.412	0.649	0.942	1.221

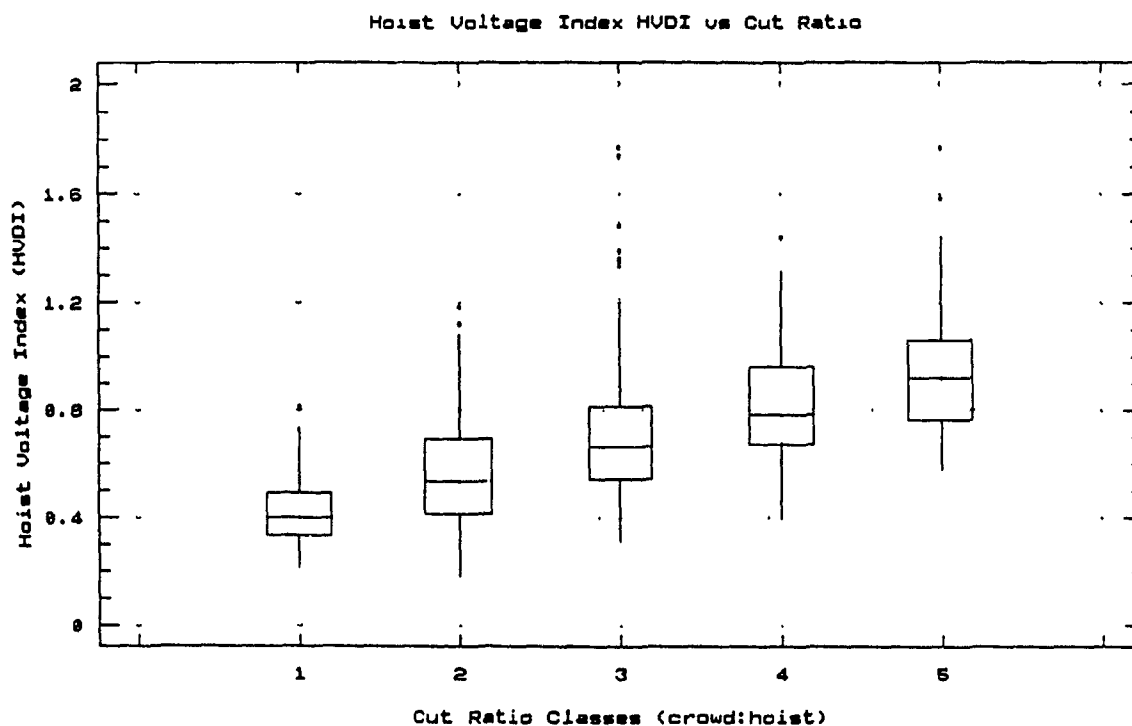


Figure 8.9.14 - Box and whisker plot of summary statistics in Table 8.9.9.

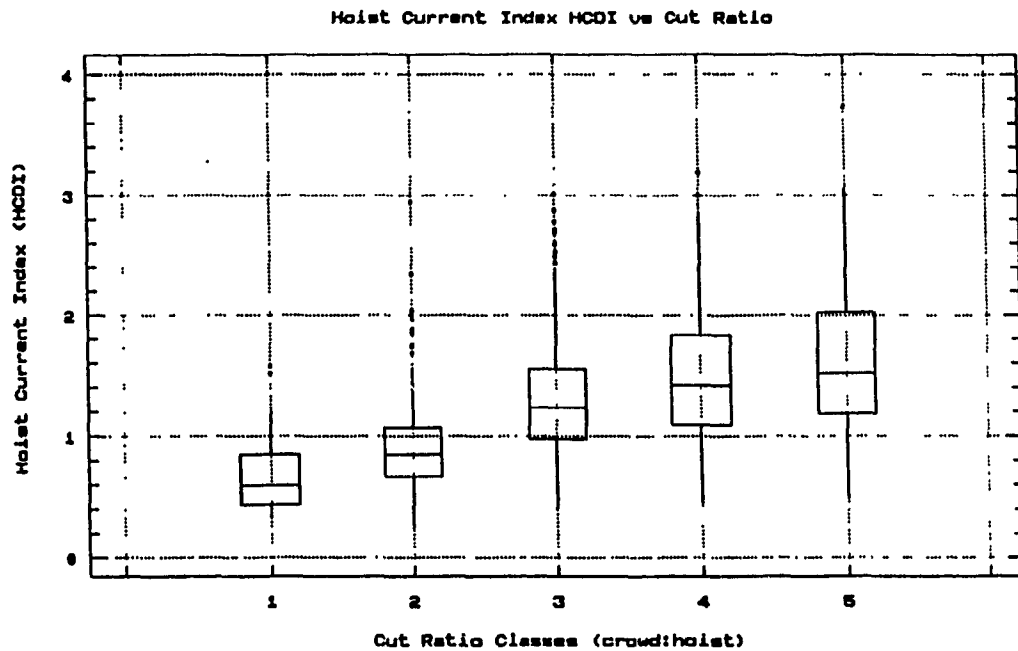


Figure 8.9.15 - Box and whisker plot of summary statistics in Table 8.9.10.

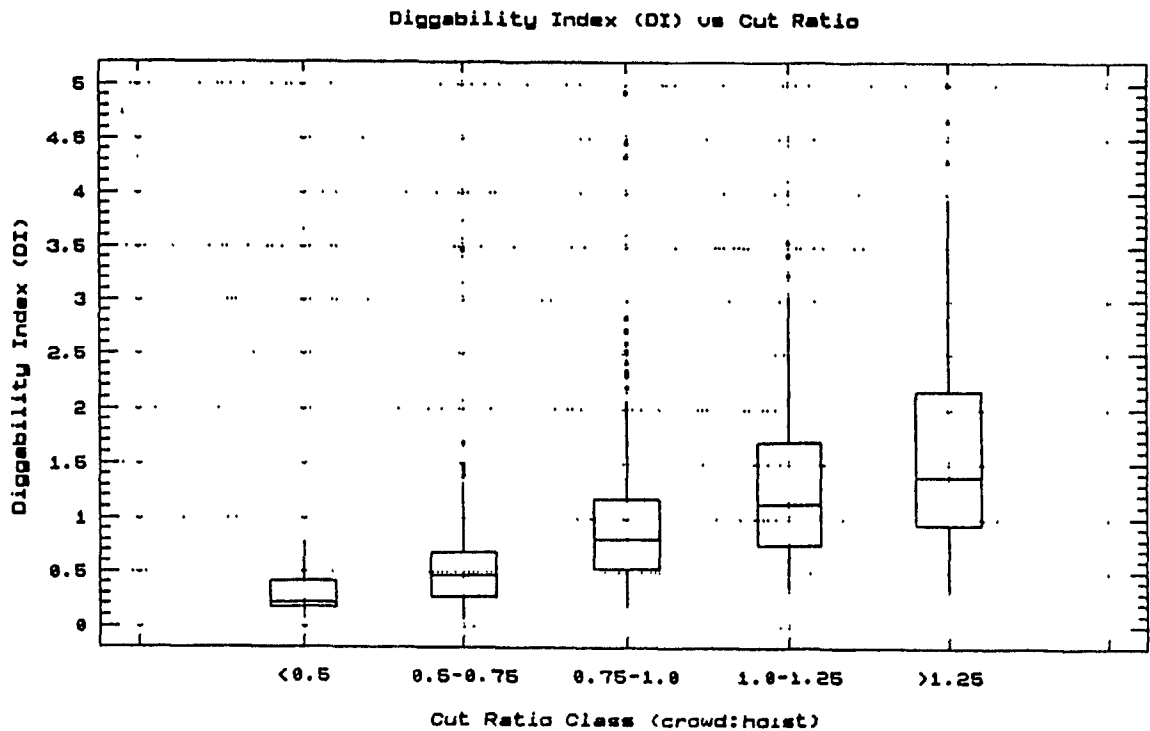


Figure 8.9.16 - Box and whisker plot of summary statistics in Table 8.9.11.

8.9.4 Discriminant Analysis - Classification of Motor Responses by Cut Ratio.

Interpretation of the statistics in the previous section, strongly suggests that the responses of the monitored parameters are more readily classified through elements of dipper trajectory, than on the basis of variability in muckpile characteristics. In order to confirm this, the data has been subjected to a multivariate statistical procedure - discriminant analysis. The following simplified explanation of it's theory and application is considered sufficient for this thesis, readers seeking a more thorough review of the procedure are referred to; Anderson, 1958 and Davis, 1986.

Discriminant analysis is appropriate to data which has been classified into two or more groups - in this case five cut ratio classes. The procedure will determine one or more functions of quantitative measurement that will help discriminate among the various groups. A method is evolved therefore, that will predict into which of the one or more groups, a new observation is most appropriately classified. The following elaboration, based on an analysis of only two groups, is taken from Davis, 1986:

"... A simple linear discriminant functions transforms an original set of measurements on a sample into a single discriminant score, ...representing the sample's position along a line defined by the linear discriminant function... Discriminant function analysis consists of finding a transform which gives the minimum ratio of the difference between a pair of group multivariate means to the multivariate variance within the two groups. If we regard the two groups as consisting of two clusters of points in multivariate space, we must search for the one orientation along which the two clusters have the greatest separation..."

The above concepts are represented graphically in Figure 8.9.17. Referring to Figure 8.9.17, groups A and B cannot be discriminated amongst on the basis of either variable X_1 or X_2 .

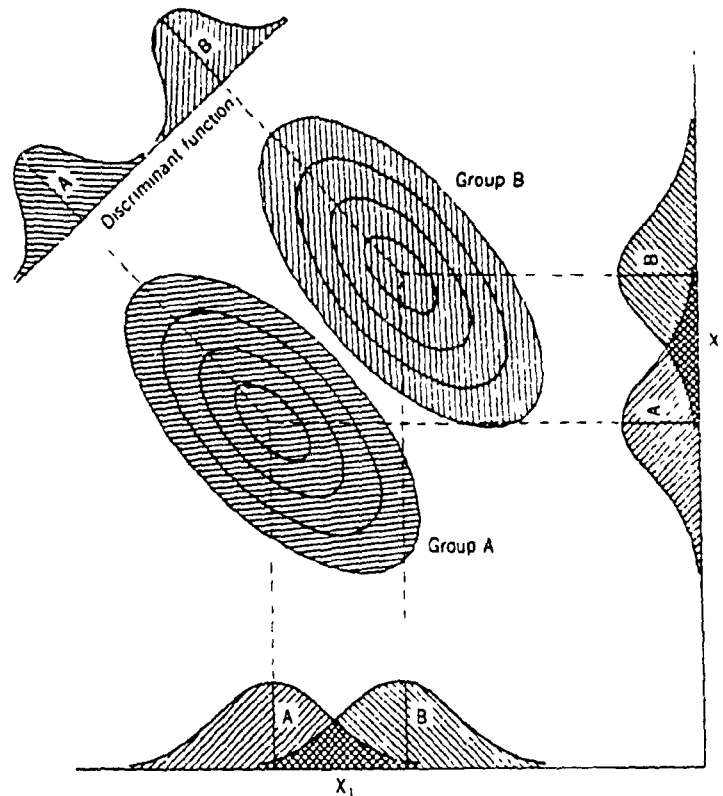


Figure 8.9.17 - Plot of discriminant function (Davis, 1986).

The orientation defined by the discriminate function however, is able to clearly distinguish between the two groups.

Listed below in Table 8.9.12, are the results of a discriminant analysis conducted using the five cut ratio classes as classification factors. The input variables upon which to base the classification included:

1. Hoist voltage..... - HV.
2. Hoist current..... - HI.
3. Percent hoist current readings > 1100 amps - PERCENT.
4. Hoist current signal length..... - SIGNAL-L.
5. Crowd Voltage..... - CV.
6. Trajectory length..... - TL.
7. Dig cycle time.....- TIME.

Table 8.9.12 - Discriminant analysis for Cut Ratio Class.

Discriminant Function	Eigenvalue	Relative Percentage	Functions Derived	Signif. Level
1	5.4062	96.93	0	.0000
2	.1511	2.70	1	.0000
3	.0270	.48	2	.0836
4	.0101	.18	3	.1004

Group	Centroids per Discriminant Function			
	1	2	3	4
1	5.1555	-1.1755	0.4706	-0.0783
2	2.4102	-0.0111	-0.1869	0.0157
3	-0.3204	0.3374	0.1263	0.0335
4	-2.5649	-0.1549	-0.0567	-0.1611
5	-4.2297	-0.7844	-0.0658	0.2489

Group Statistics

Group	1	2	3	4	5
COUNTS	19	134	184	90	31
MEANS:					
HV	375.32	332.05	291.55	241.20	190.32
HI	1339.33	1276.49	1172.06	1089.62	1034.84
PERCENT	81.28	73.39	64.15	54.15	49.42
SIGNAL-L	7739.81	11214.60	12997.60	12706.40	13049.90
CV	256.55	348.73	410.99	431.61	428.88
TL	38.52	39.57	37.41	34.44	32.66
TIME	10.72	10.86	9.67	8.99	9.01

STD.DEVIATIONS:					
HV	31.92	61.55	44.14	29.49	25.41
HI	145.29	146.91	146.49	148.74	147.03
PERCENT	11.99	13.47	13.67	13.69	11.78
SIGNAL-L	3964.88	6440.89	5690.19	5526.92	6529.74
CV	50.35	78.18	75.54	63.63	70.14
TL	5.26	5.99	5.23	4.55	4.76
TIME	2.06	3.10	1.91	1.38	1.70

Classification results for Cut Ratio Classes

Actual Group	Predicted Group (count,percentage)									
	1	2	3	4	5	1	2	3	4	5
1	19	0	0	0	0	100.00	.00	.00	.00	.00
2	14	106	14	0	0	10.45	79.10	10.45	.00	.00
3	0	17	145	22	0	9.24	78.80	11.96	.00	.00
4	0	0	8	72	10	.00	.00	8.89	80.00	11.11
5	0	0	0	3	28	.00	.00	9.68	90.32	.00

With reference to Table 8.9.12, four discriminant functions have been calculated from the five groups, and four positive eigenvalues have been found. The first, $\lambda_1 = 5.4062$ accounts for almost 97% of the between group variance, λ_2 , the second discriminant function, accounts for only 2.7% of the between group variance, and λ_3 and λ_4 account for the remaining 0.66% of the variance between the five groups. Note the significance levels of the first two discriminant functions indicate that they are both highly significant at the 0.05 level (95% confidence limits). The classification results from the discriminant analysis indicate that on average, greater than 80% of the observations have been classified correctly. Figure 8.9.18 is a plot of the discriminant analysis for cut ratio class. Discriminant function one, identified as accounting for 97% of the between group variance, is clearly able to discriminate amongst the five groups (although some small overlap is present between groups 2,3 and 4). Note that very little distinction between groups is possible with function 2, as it accounted for less than 3% of between group variance.

The results of the discriminant analysis confirm that classification of the monitored shovel responses is reliably accomplished on the basis of elements of dipper trajectory. The predominant influence of dipper trajectory over shovel responses, becomes a critical concern when one considers the operator dependency of this parameter. If muckpile characteristics were the sole controlling element in assessments of diggability, the influence of variations in operating practice during the dig cycle could be negated. However, it is the operator that selects a dipper trajectory for each cycle, and this parameter can, if not understood and accounted for, greatly influence assessments of diggability. The next section will discuss the approach used to determine the extent to which operating practices vary, and the significance of such variability. With this assessed, it will then be possible to relate/confirm that changes in digging effort per day of the monitoring study are a consequence of changes in muckpile characteristics - diggability.

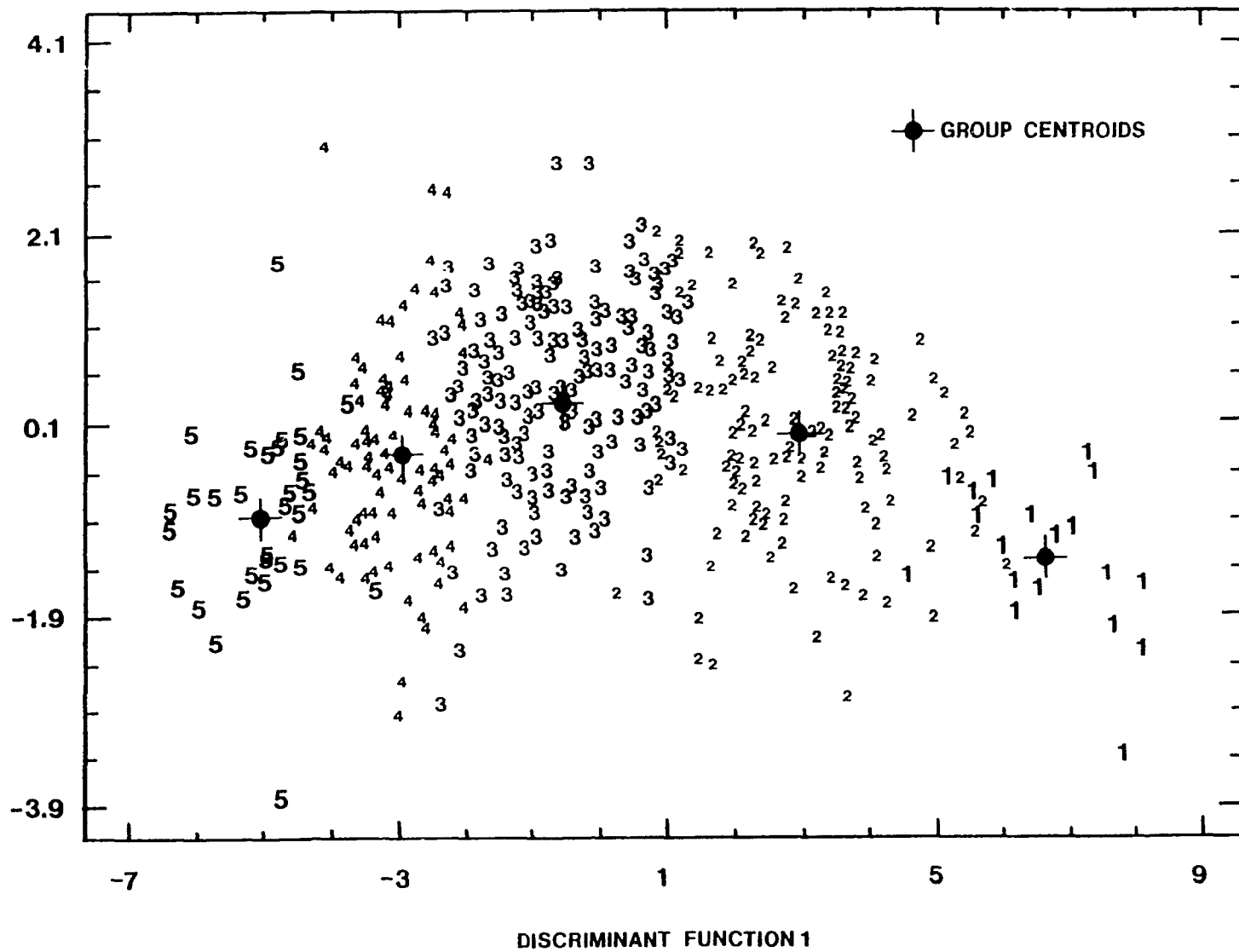


Figure 8.9.18 - Plot of discriminant functions for classification of motor responses by cut ratio classes.

8.10 OPERATING CHARACTERISTICS.

An objective assessment of the influence and extent of variation in the operating practices amongst the operators studied is intricate. The task is complicated by the fact that each day of monitoring was conducted in muckpiles that were to some extent distinct. It is impossible therefore to compare the performance of operators on the basis of equivalent digging conditions. For present purposes it must be assumed that if variation in digging practices exist, in a large enough sample the operating characteristics of each operator will overshadow variations in muckpile diggability, within said sample.

An assessment of operator variability was first conducted on the basis of the expansive Polycorder data. This exercise suggested that specific ranges for motor responses and calculated diggability indices were associated with individual operators. The analysis then turned to the General Electric data which offered measurements on dipper trajectory and load weights, in an effort to validate and explain observations from the Polycorder data. Unfortunately, the G.E. data only covers the performance of each operator for 48 successive hours, each in a unique blast environment. The original design of the experiment, focused on generating a series of blast locales in which to assess shovel responses, to the ensuing and varied muckpile characteristics. Although of obvious concern, the experiment was not originally designed to assess operating practices.

In total, the performance of six operators was evaluated on #15 shovel. Operators 1, 2, 3 and 4 are the regular operators on this shovel. Operators 5 and 6 were replacements taken from #14 shovel (another P&H 2800XP), when a regular operator was unable to work. Operators 5 and 6 stated that #15 shovel, although an identical model, was quite distinct from their regular (#14) shovel. It was their opinion that the hoist and swing motors on #15 shovel were noticeably faster than on #14 shovel, and that

they found it difficult to get the feel of the shovel. In light of this, their performance is not considered in an assessment of operator variability.

Operator identified variability amongst the performance of two identical shovels is very noteworthy, and should be seriously considered in any future monitoring trials that involve the instrumentation of more than one shovel, especially of different models. The reasons behind the faster hoist motor speeds on #15 shovel are not fully understood. It is believed a consequence of parts replacement, installation of governors etc. over its lifetime. It should not be taken for granted therefore, that two identical shovels will perform equally. It was not possible to evaluate / measure the calibration of each shovel (i.e. hoist motor power output) as this required specialized equipment and personnel. The procedure is extremely hazardous even to experienced personnel, and is only conducted when absolutely necessary in diagnosing shovel malfunctions (B. Daniels - Fording electrical maintenance, pers. comm., 1988).

Table 8.10.1 documents the average values of several important shovel performance parameters, recorded for each operator, across all levels of digging difficulty. Table 8.10.1 is divided into three parts: The first presents data collected by the G.E. monitor, this includes position and hoist motor data. The second part documents the productivities of each operator, sample sizes being determined by the number of loads registered by the G.E. monitor while operating in a production gathering mode. The third part represents data collected by the Polycorder on hoist and crowd motor performance. The Polycorder sample sizes are the largest, as it was used during all monitoring periods (August and October 1988). It has provided data on the performance of each operator for at least four days in a range of digging conditions.

Table 8.10.1; Average monitored parameter responses according to operator.

OPERATOR No:	1	2	3	4
GENERAL ELECTRIC MONITORED DATA				
Sample Size:	216	126	141	167
Hoist Voltage (V):	285.79	266.10	283.73	319.59
Hoist Current (A):	1179.95	<u>1141.37</u>	<u>1125.05</u>	1257.56
Cut Ratio:	0.82	<u>1.00</u>	<u>0.94</u>	0.77
Crowd Travel (ft.):	15.37	12.87	13.91	12.42
Hoist Travel (ft.):	19.55	13.52	15.61	16.94
Trajectory L (ft.):	42.84	32.56	36.41	34.73
% Hoist I >1100 A:	65.16	<u>60.71</u>	<u>58.64</u>	71.73
Hoist I Signal L:	15587.90	11291.80	11834.70	9856.67
Corrected hoist I Signal L:	1326.05	1313.34	1246.94	1067.21
Dig Time (sec.):	12.01	8.53	9.30	9.14
HVDI:	0.69	0.81	0.69	0.57
HCDI:	1.28	1.35	1.28	1.03
DI:	0.95	1.17	0.93	0.69
Hoist Power:	374464	340144	363203	424070

GENERAL ELECTRIC PRODUCTION STATS.				
Sample Size:	552	534	524	549
Average Load (Tons):	37.60	39.25	38.80	43.70
Productivity Tons**/: min. of dig time.	187.84	276.08	250.32	286.87

POLYCORDER MONITORED DATA				
Sample Size:	569	214	262	409
Hoist Voltage (V):	243.38	227.39	232.39	250.43
Hoist Current (A):	1185.29	1182.82	1146.57	1301.90
Crowd Voltage (V):	404.71	<u>425.00</u>	<u>427.96</u>	384.08
Crowd Current (A):	479.73	<u>439.03</u>	<u>436.06</u>	463.29
% Hoist I >1100 A:	63.49	60.56	58.97	71.01
Corrected hoist I Signal L:	5299.26	4542.95	4675.73	4157.41
Dig Time (sec.):	10.81	8.86	9.21	9.27
HVDI:	0.48	0.53	0.50	0.49
HCDI:	0.47	0.49	0.50	0.39
DI:	0.23	0.27	0.26	0.20
CVDI:	0.37	0.34	0.35	0.47
CCDI:	0.64	0.71	0.71	0.69
CDI:	0.23	0.24	0.25	0.32
Hoist Power:	395918	380550	376237	448523

NOTE: * When used to compare operator performance this parameter is divided by average dig cycle time per operator.

** Productivity above is as estimated on the basis of the average dig cycle time per operator, not load time as measured by the G.E. monitor.

It is evident that several parameters display clear associations with individual operators. If operator variability were truly insignificant, average values for all parameters would be approximately equal for each operator. The "bolded" values draw attention to the fact that for most parameters, operator number 4 has the most favourable responses. The underlining between operators 2 and 3 illustrates the close accordance between these operators for several parameters.

The statistics suggest the presence of three operating styles: Operator 1 being the least efficient, operators 2 and 3 are of equal and average efficiency and the operating practices of operator 4 are considered the most efficient. The following sections explain the rationale behind this grouping of operator performance:

8.10.1 Relation between operators and diggability indices.

Referring to Table 8.10.1 attention is drawn to the average values of the hoist and crowd based diggability indices (DI and CDI), and their component indices (HVDI, HCDI and CVDI,CCDI). With regard to the diggability indices calculated from the G.E. data, operator 4 is seen to have the lowest average values. The same relation is conspicuously seen to exist within larger sample sizes associated with the Polycorder derived index values: Operator number 4's diggability indices are distinct from those of the remaining operators. The average values falsely imply that operator number 4 consistently worked in easy digging conditions. This was not the case.

Most notably, the average crowd voltage digging index value (CVDI) for operator 4 is considerably greater than those of the remaining operators. In contrast to prior investigations (Williamson et al., 1983 and Mol et al., 1987), the CVDI value has been shown to be inversely related to levels of digging difficulty. However, irrespective of the index's significance in describing levels of digging effort, it displays a clear

sensitivity to the operating practices of operator number 4. It is known, based on field observations of digging conditions that he operated in a range of digging conditions; from very easy to average in difficulty. The operator's associated hoist based digging indices support these assessments of daily digging conditions. This, in conjunction with the operator's shallow average cut ratio refute conclusions suggested by the CVDI's that he operated in the most difficult digging conditions.

The average values of the hoist current diggability index (HCDI) are also seen to conspicuously respond to the operating characteristics of operator 4. In this regard, the HCDI suggest consistently easy to average digging conditions for operator 4.

Listed below in Table 8.10.2 are the average diggability indices (based on Polycorder data) determined for August 20th and 23rd. Each of these days offered near identical digging conditions; the easiest encountered during the study. Note the clear distinctions between the two operators (3 and 4), with respect to the HCDI and CVDI.

Table 8.10.2 Summary of Diggability Indices for the most easy digging conditions - Polycorder Data.

	<u>OPR. No.</u>	<u>HVDI</u>	<u>HCDI</u>	<u>DI</u>	<u>CVDI</u>	<u>CCDI</u>	<u>CDI</u>
Aug 20	3	0.45	0.49	0.23	0.28	0.71	0.20
Aug 23	4	0.48	0.36	0.18	0.48	0.67	0.33

Given the large population of dig cycles and the range of digging conditions therein, there exists good evidence that the diggability indices may indeed be biased by operating characteristics.

8.10.2 Classification of Operators.

In this section, use is made of the G.E. derived data on shovel load weight and dipper trajectories, in an evaluation of individual operating practices. It is postulated that if operator variability does exist it would be reflected in dipper

trajectories unique to one or more operators. Variability in operating practice may not exist amongst all operators. To label operator variability a significant concern, it is sufficient to identify deviation from the norm for a single operator.

Operator Number 4.

Both the Polycorder and G.E. data show that operator number 4 operates with the highest average hoist voltages, currents, and percentage of hoist current readings (71%) in excess of 1100 amps per dig cycle. All of these indicate fast easy travel through the muckpile. The very low average hoist current signal lengths confirm the relative ease of digging. The low average cut ratios (0.77) explain in part, the favourable motor responses.

Operator #4 has the highest average dipper load weights which, when combined with his average dig times translate into a very respectable level of productivity. This indicates that his favourable motor responses are **not** at the expense of dipper filling. The question arises therefore, why does operator number 1 with comparable motor responses and a near equal average cut ratio (0.82 vs 0.77) have such low productivity?

Operator Number 1.

After operator number 4, operator 1 has the highest average hoist voltages, currents and percent of hoist current readings greater than 1100 amps. Again, these indicate relatively easy digging. However, his digging practice is typified by higher average hoist current signal lengths, which are indicative of more difficult dipper advance.

Note that operator 1 employs the longest dipper trajectory; almost 43 feet on average. Listed below in Table 8.10.3 are the average start of dig cycle hoist and crowd values (min. crowd arm extension and max. hoist rope length) and end of dig cycle values (max. crowd arm extension and min. hoist rope length) in feet:

Table 8.10.3 - Average dipper coordinates by operator for start and end of dig cycle.

OPERATOR No:	1	2	3	4
START DIG CYCLE				
Min crowd extension:	24.8	25.3	25.3	25.1
Max hoist rope length:	57.2	56.5	56.3	56.3
END DIG CYCLE				
Max. crowd extension:	40.2	38.1	39.2	37.5
Min. hoist rope length:	37.6	43.0	40.7	39.4
Trajectory length:	42.8	32.6	36.4	34.7

The crowd and hoist travel values for operator number 1 show a proportional increases over operator number 4's values. The net result is a longer dipper trajectory of near equivalent ratio and shape that, due to its length, must pursues a deeper cut through the muckpile. The added time required to complete these long trajectories is responsible for operator number 1's low productivity.

Figure 8.10.1 illustrates plots of dipper trajectories for operator 4 on August 23rd. Figure 8.10.2 are plots of trajectories for operator 1 recorded for August 24th. There is a clear difference in the shapes of trajectories pursued on each of these days. On August 23rd the shovel was excavating that portion of blast EM#1 which was loaded with explosive columns of uncharacteristically high strength. It is likely that operator number 4's distinctively shallow trajectories on this date were motivated by a significant improvement in muckpile diggability at this location. When the trajectory ranges of operators 1 and 4 are compared on August 22nd (cut ratio = 0.79) and 25th (cut ratio = 0.79) more similarity is apparent, Figure 8.10.3 and 8.10.4. It is concluded that operators will adjust their digging practice to accommodate variations in diggability. It is theorized however, that based on the strong association of a unique range of motor responses to certain operators, that each operates within a characteristic range of dipper trajectories.

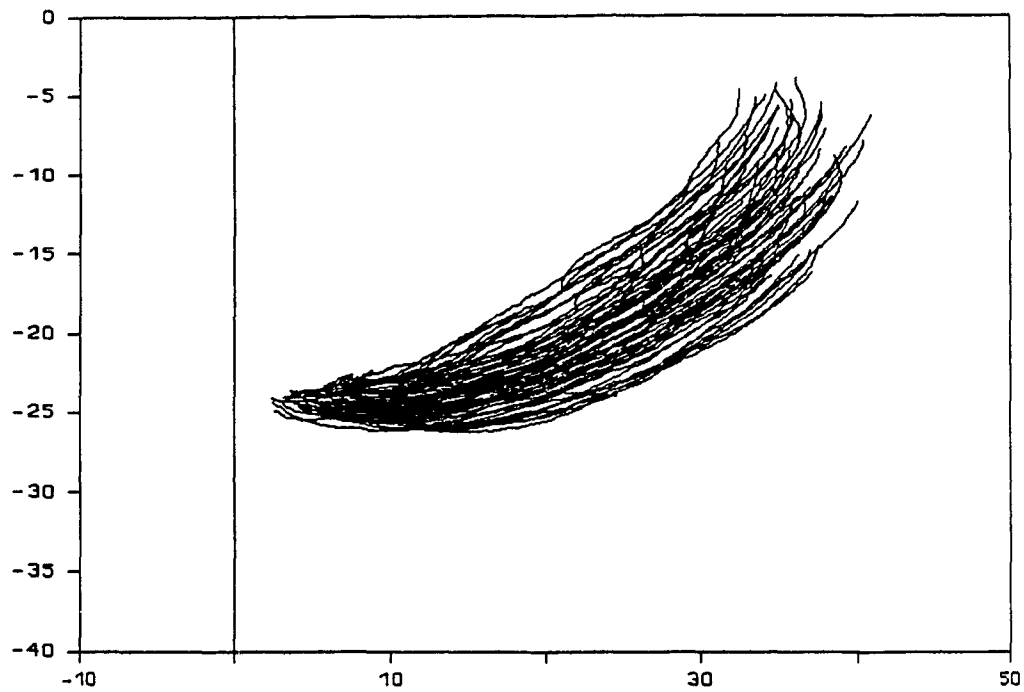


Figure 8.10.1 - Dipper trajectories - operator #4 August 23rd.

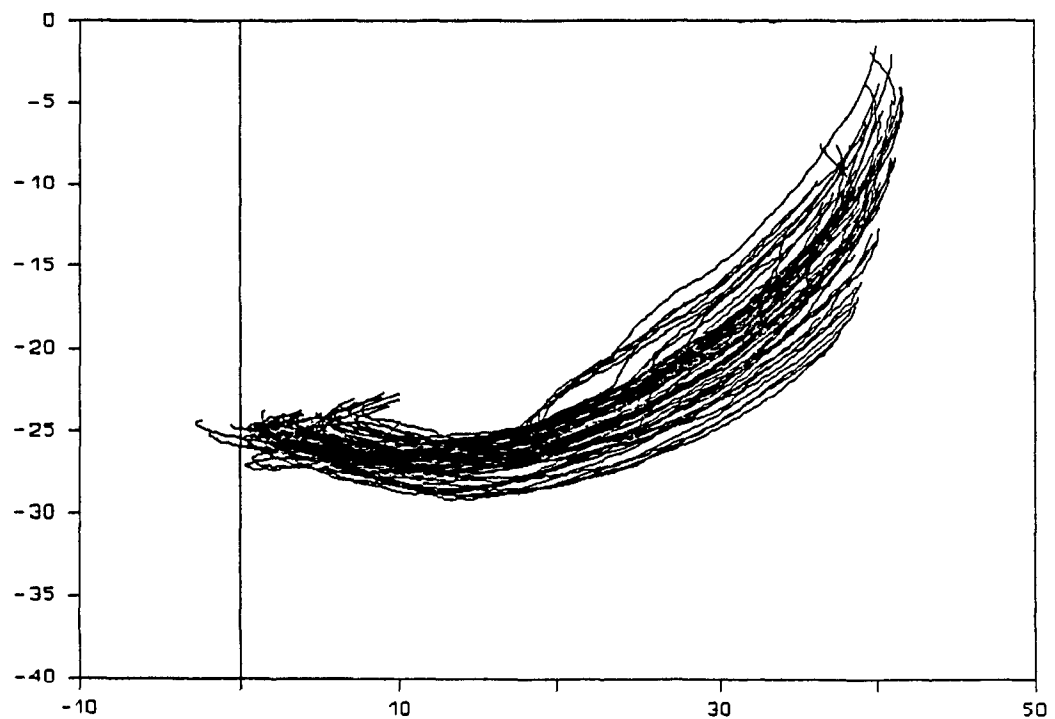


Figure 8.10.2 - Dipper trajectories - operator #1 August 24th.

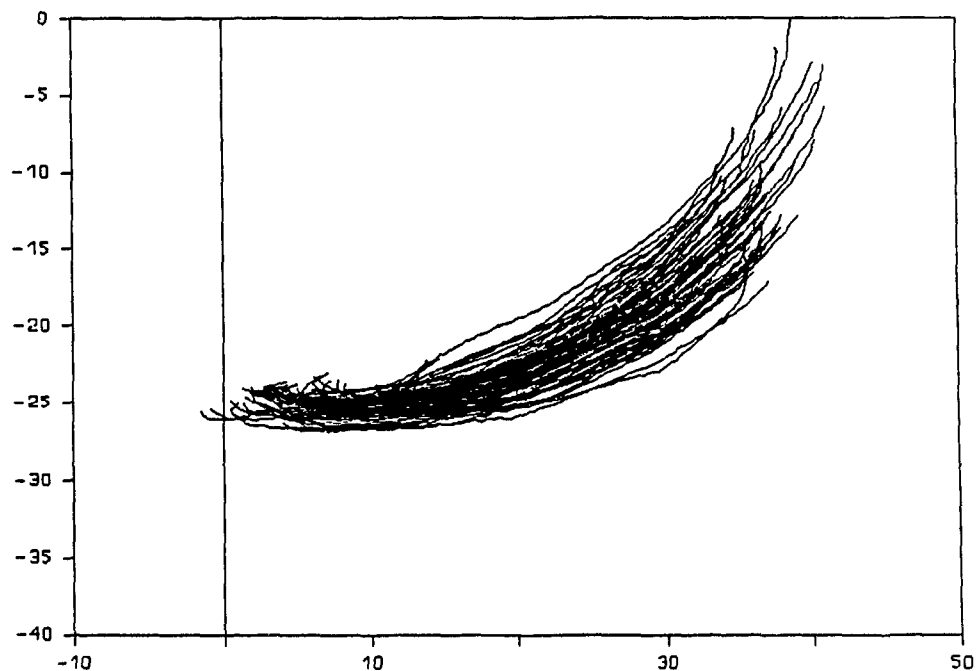


Figure 8.10.3 - Dipper trajectories - operator #4 August 22nd.

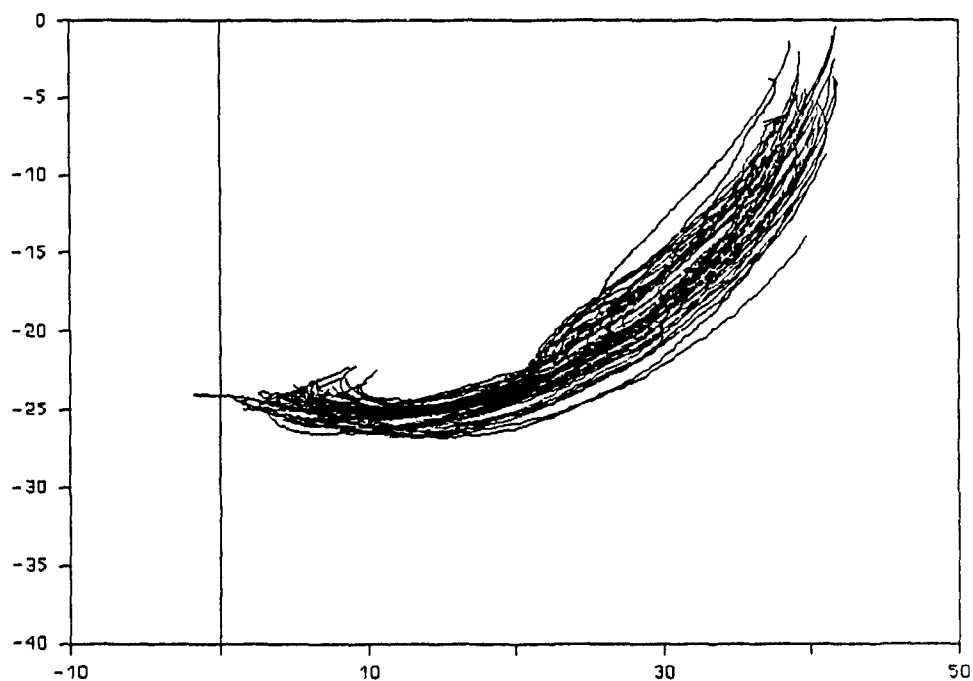


Figure 8.10.4 - Dipper trajectories - operator #1 August 25th.

Adjustments to the operating practices for each operator, prompted by changes in muckpile diggability, are believed to occur within the extremes of each operator's characteristic digging practice. For example consider the performance of operators 3 and 4 while working in very easy digging conditions. Table 8.10.2 has documented a clear distinction between the diggability indices calculated for each operator while working in very similar digging conditions. When operator number 3 moved from easy digging on August 20th (DI=0.68) to hard digging on the 21st (DI=1.27) he only adjusted his average cut ratio from 0.93 to 0.96. When operator number 4 shifted from average digging on August 22nd (DI=1.00) to very easy digging on the 23rd (DI=0.43) he changed his average cut ratio from 0.79 to 0.76. The net result is as displayed in Figure 8.10.5, which shows the most frequent trajectory ranges for each date. Clearly, there exists a marked difference in the manner in which each operator handles easy digging conditions.

Figure 8.10.6 illustrates trajectory ranges for each of the four operators throughout the entire period during which the G.E. monitor was employed (August 15th to 25th). Note that the average trajectory for each operator falls within class 3 cut ratios (0.75 - 1.00). The ranges represent the most frequent trajectories for each day. It is apparent from these plots that operator number 4 pursues trajectories that, on average are notably less deep than those of the remaining operators. It is surprising therefore, that operator 4 with his distinctively shallow passes, would have the highest average dipper load fills (weights). This will be addressed in greater detail in section 8.10.3.

Operators 2 and 3.

The practices of operators 2 and 3 are, as indicated by Table 8.10.1, very similar. Both employ comparatively high cut ratios and their average hoist and crowd voltages and currents are comparable. The productivity of each is near equivalent, both in

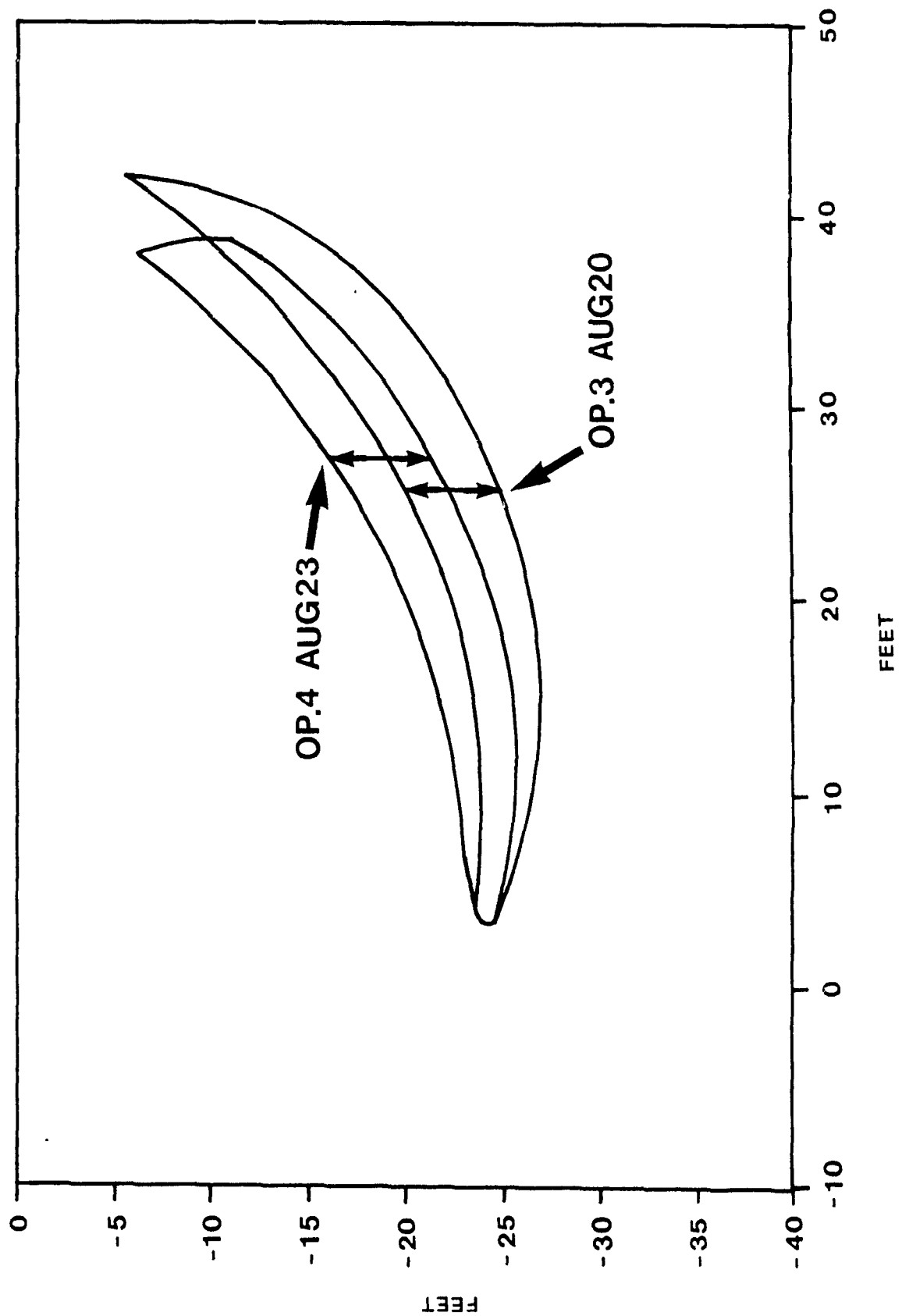


Figure 8.10.5 - Dipper trajectory ranges - easy digging operators 3 and 4.

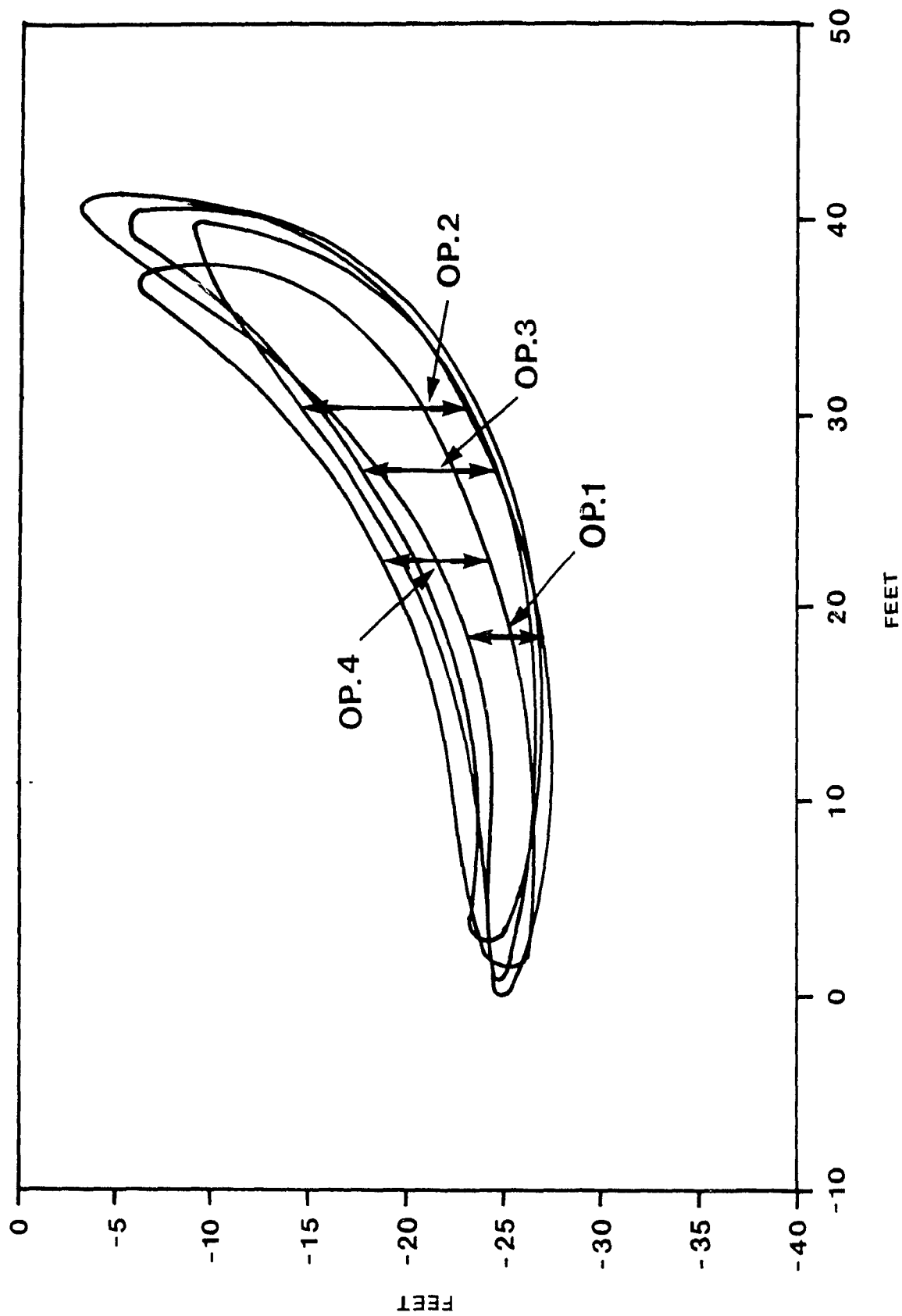


Figure 8.10.6 - Dipper trajectory ranges for operators 1,2,3, and 4.

terms of average load weight and in Tons per minute of dig time. As indicated in Figure 8.10.6 the trajectories of each are analogous.

8.10.3 Correlation of Dipper Load Weights and Hoist Motor Responses

An analysis of hoist current response, cut ratio and load weights has suggested that shallow dipper trajectories will result in more efficient dipper filling. Since the G.E. monitor could not be used to gather load weights at the same time as sensor monitoring, the Polycorder was used to monitor the motors while manually recording the load weight (from the G.E. monitor) for each cycle. When the hoist current signal from the Polycorder was compared with load weight, it was observed that the percent of hoist current readings in excess of 1100 amps increased with average load weight, Figure 8.10.7.

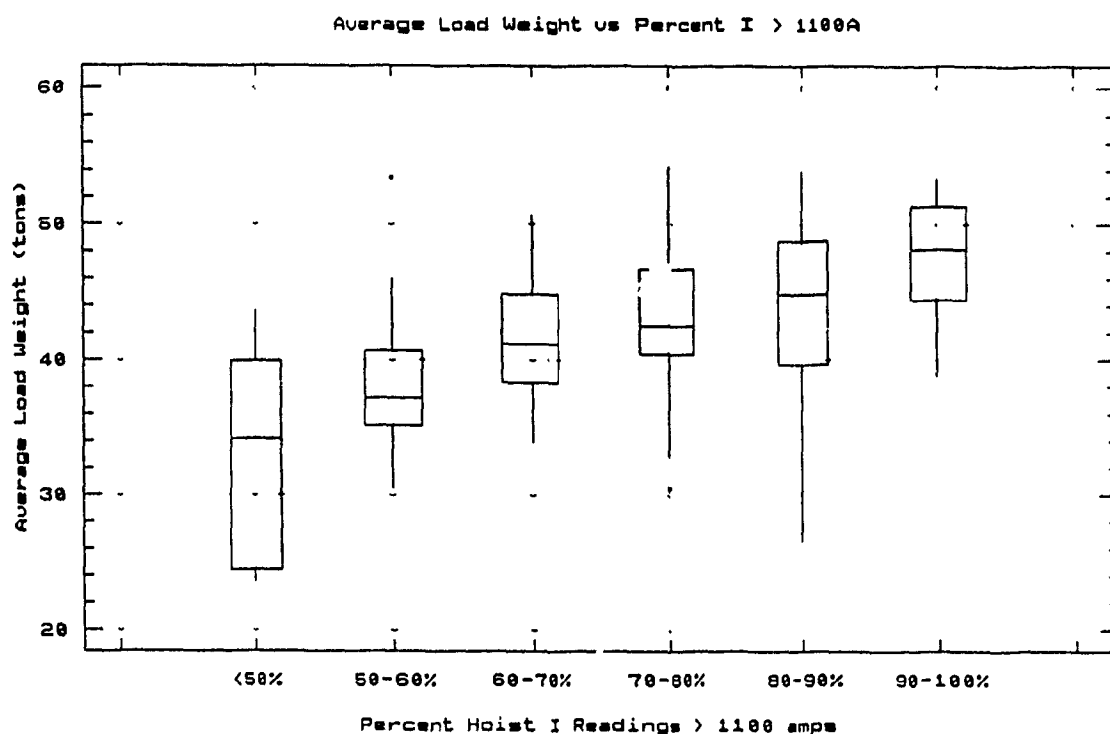


Figure 8.10.7 - Relation between dipper load weight and percent of hoist I readings > 1100 amps.

Figure 8.10.8 illustrates a previously explained relation between cut depth and the percent of hoist current readings >1100 amps. As cut depth (ratio) increases, the percent of hoist current readings >1100 amps decreases. It is impossible to directly compare load weight with cut ratio because the two could not be monitored simultaneously. It is possible however, to conclude based on the two previous observations, that as cut ratio decreases load weights increase. The logic behind this relation is as follows:

High load weights correspond to High % >1100 amps
whilst

High %>1100 amps correspond to Low cut ratios
therefore

High load weights correspond to Low cut ratios

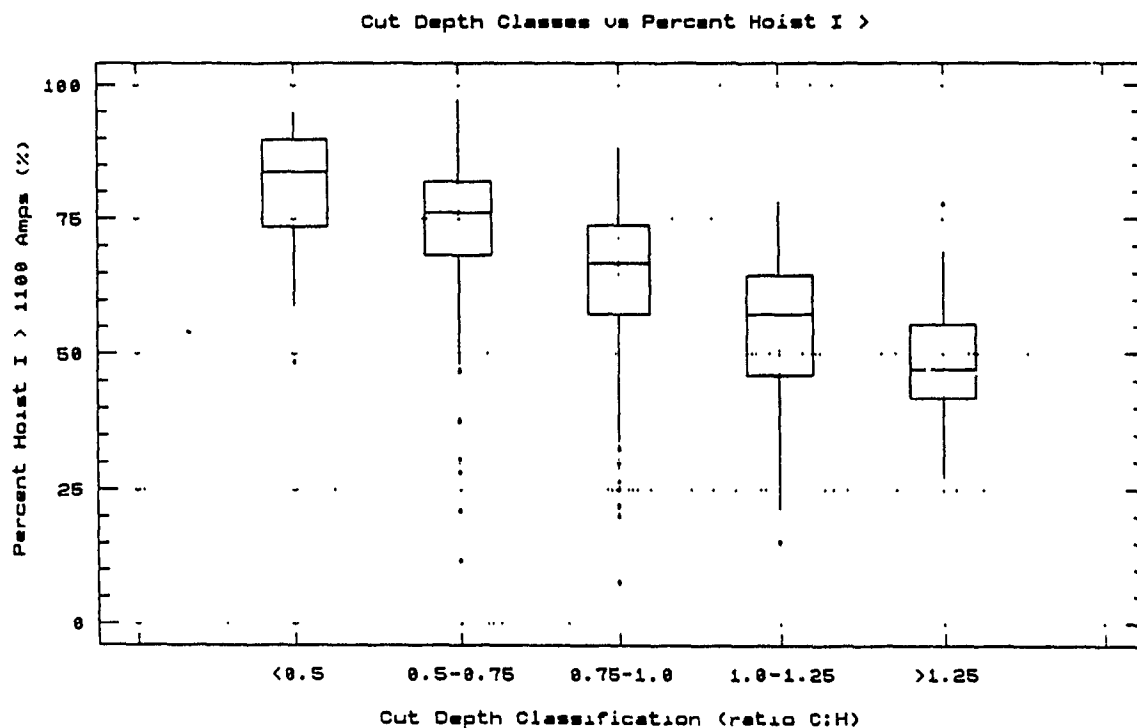


Figure 8.10.8 - Box and whisker plot, relation between cut ratio and percent of hoist I readings > 1100 amps.

The above reasoning finds support in the data of Figure 8.10.9, showing a plot of load weight classes against the fill RMS current per dig cycle. The fill RMS is the root mean square hoist current value for each dipper "fill" period, per load (recall that fill time is the interval of load cycle that hoist voltage is > 100 volts). This value is derived by the G.E. monitor while operating in a production statistic gathering mode for each load. Clearly, there is a strong relation between the hoist current response and the resultant dipper load per dig cycle. High average hoist currents are reflected in more efficient dipper loading. The percent of hoist current readings > 1100 amps is a reliable measure of both the average current value and degree of fluctuation (motor effort) in the signal during dipper filling. Importantly, this value is available for each load monitored by the Polycorder and G.E. monitor. Note that operator 4 has the highest average percentage of hoist

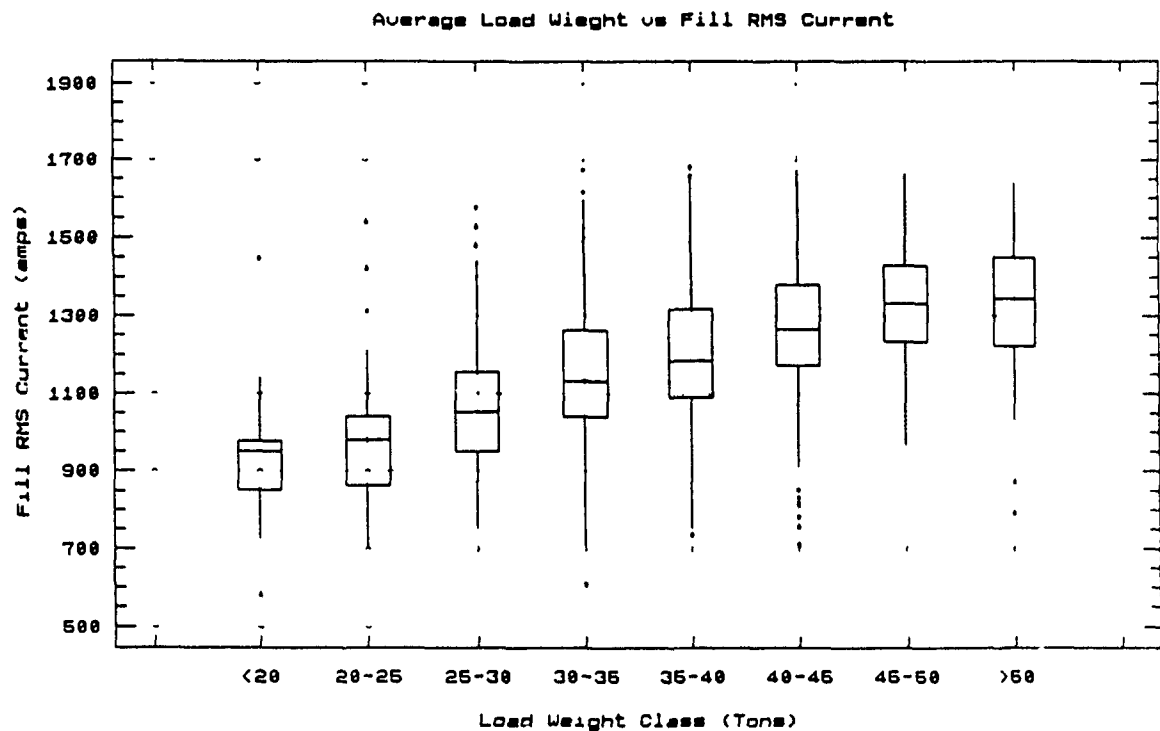


Figure 8.10.9 - Relation between dipper load weight and Fill RMS.

current readings >1100 amps - 71%, throughout the August and October monitoring periods. He was also seen to consistently have the highest average dipper load weights and productivities (Table 8.8.1).

It must be appreciated that dipper trajectory does not bias calculation of dipper load weights as they are determined during the swing to truck cycle. The interpretations are dependant upon the assumption that each operator maintains the shovel an equal distance from the toe of the muckpile. Since no differences in muckpile profile were observed; the position of the muckpile relative to shovel (and dipper) can be considered constant. It was not possible to take direct measures of shovel position. Daily visual estimates of the distance from the shovel crawler tracks to the muckpile toe indicated that all operators worked (dug) from an equivalent position. This observation finds support in the data of Table 8.10.3, which shows that all operators start their respective dig cycles from approximately the same position (i.e. equal dipper trajectory coordinates).

The data suggests that the dipper fills more efficiently if easy, shallow trajectories are employed. Such trajectories have been shown to be coincident with high percentages of hoist current readings in excess of 1100 amps. The mechanics behind this phenomena can only be speculated upon at this point. Perhaps high levels of dipper momentum developed with such trajectories, pack the dipper better through improved dipper filling. Laboratory research currently under way at McGill University, using a scale model of an electric shovel should be able to address issues of dipper trajectory and filling mechanics, as they pertain to muckpile looseness, fragmentation etc.. At the time of writing this thesis however, work in the laboratory had not progressed sufficiently to answer these questions (Hadjigeorgiou, pers. comm., 1989). Results presented in section 8.13.1 have identified a relation between diggability and fragmentation.

Equipment manufacturers have traditionally recommended cut depths that will permit dipper filling in trajectories equal to between 2 and 3 times the dipper length (Figure 8.10.10).

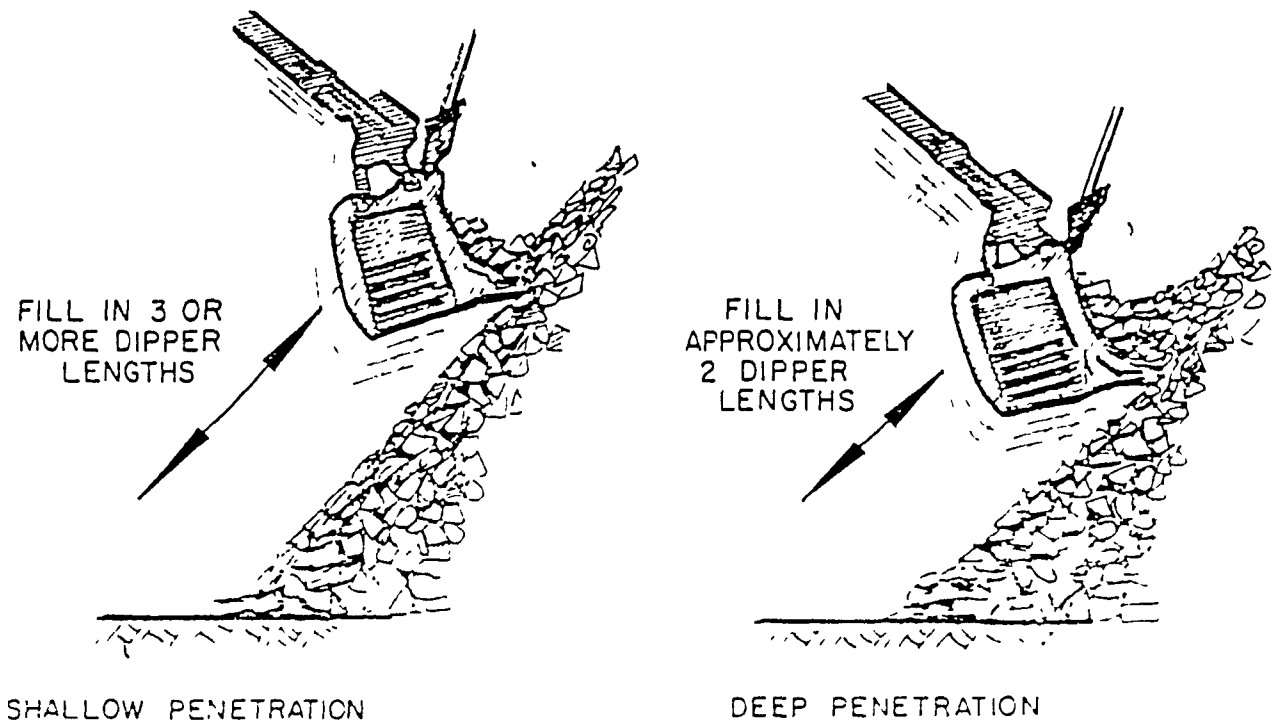


Figure 8.10.10 - Cut depth and dipper filling (Bucyrus Erie, 1979)

The dipper on the instrumented shovel was approximately 13 feet along it's base. With average recorded trajectories of between 34 and 43 feet, this rule of thumb appears to have been satisfied. It is to be stressed that, the performance of each operator on #15 shovel is good, all have a minimum of 15 years of operating experience and have proven themselves quite capable. The fact that one operator (number 4) appears to have better dipper fills and a characteristic range of motor responses is likely a consequence of minor idiosyncrasies associated with his digging practice - shallower cut depths.

The fact that the current monitoring technology has permitted identifying variations in the characteristic digging practices amongst so skilled a group of operators, suggests its potential future application in shovel operator training. This issue will be addressed in greater detail in Chapter 10 - Recommendations for Future Work.

8.11 TRAJECTORY AND OPERATOR ADJUSTED DIGGABILITY INDEX.

The results of the two preceding sections have demonstrated that the monitored shovel responses appear to be influenced by operating characteristics (trajectory). This section will propose an approach to incorporate the influence of varied trajectories into assessments of diggability through the hoist diggability index; DI. This is not to suggest that the average daily diggability indices are not realistic (i.e. heavily biased by operator variability), merely that expansion upon them, on the basis of elements of dipper trajectory, should enhance their significance and worth.

The problem was initially approached with the intent of incorporating an element or equation in the DI, describing the trajectory for each dig cycle. In this sense, the DI for each cycle would be corrected on the basis of deviations from a standard/idealized trajectory. The average cut ratio for all dig cycles (0.88) was taken as a base reference. The equations below demonstrate an attempt to correct DI's for deviations from the average cut ratio.

$$\text{If } CR > 0.88 \text{ then } DI = \left[DI_i - |CR - 0.88| \cdot DI_i \right] \quad (8.11.1)$$

$$\text{If } CR < 0.88 \text{ then } DI = \left[DI_i + |CR - 0.88| \cdot DI_i \right] \quad (8.11.2)$$

where; CR = Cut Ratio
DI = Corrected Diggability Index
DI_i = Initial Diggability Index

Equation 8.11.1 tries to correct high DI's that may have resulted from deep cut ratios. This is done by subtracting from the original DI for each load, a proportion of the DI equivalent to the amount of excess cut depth. As indicated by equation 8.11.2, the DI's for shallow trajectories are increased in proportion to the amount of cut ratio below the norm.

The above attempt to "standardize" the diggability index according to cut ratio failed. This was explained on the basis of variations in muckpile profile that exist on a per dig cycle basis. And on the assumption that muckpile characteristics are continuous throughout it's height.

When working a muckpile, it's profile is likely to change from one dig cycle to the next. If successive dipper passes are at the same approximate location, a bowl like recess will develop, and the distance between muckpile surface and the shovel will increase. As a result, progressively longer, deeper trajectories (**relative to the shovel**) will be used each time the dipper is loaded from this muckpile location. Since material characteristics have not changed, these deeper cuts (as indicated by dipper travel relative to shovel) will not be associated with a change in motor response or hence, DI's. Upon rilling, portions of, or the entire muckpile is replenished, and the muckpile surface is again at a different position relative to the shovel. It must be appreciated therefore, that the operator will load from several locations within the same muckpile during normal operation, each to some extent, distinct (i.e. with respect to muckpile profile, distance from shovel etc.) from the others. When considered over an entire day the effect of these differences in muckpile shape or profile are averaged out. Note in the above context, variations referred to in the distance between shovel and muckpile surface are not of a magnitude sufficient enough to warrant and advance of the shovel towards the muckpile; propelling. They refer to the more subtle variations in profile that occur as a consequence of normal digging actions between propel episodes.

Another concern is that material characteristics such as fragmentation and looseness are not continuous throughout the muckpile. Surface fragmentation is generally coarser than that found at greater depth, and compaction will obviously increase

with depth. Large blocks that have rilled on to the surface of the muckpile from the collared portion of the bench, will remain close to the toe of the muckpile. Smaller fragments will fall between the spaces of larger blocks. Vibration caused by dipper advance through the muckpile will further promote the migration of smaller blocks toward the bottom of the muckpile.

The preceding reasons discouraged attempts to reconcile individual DI's on the basis of individual trajectories. It is proposed that a final daily Shift Diggability Summary (SDS), be established, based upon the differences between the trajectories of cut ratio classes 2 (0.50-0.75), 3 (0.75-1.00) and 4 (1.00-1.25). In this manner, variations in the muckpile profile from one dipper pass to the next are minimized. Class 1 and 5 cut ratios have, for simplicity been omitted. This was justified on the basis of the relatively small portion of all dig cycles that fall within these two classes. The SDS is not a single value, rather it is a concise list of data that helps define both the degree of digging difficulty and nature of digging conditions which the shovel was involved in for each shift.

Table 8.11.1 defines the manner in which the Shift Diggability Summary (SDS) is established. Listed in it are the summary values of diggability indices per cut ratio class 2 to 4, per day - the SDS's. The values in Table 8.11.1 have been derived in the following way - using August 25th as an example:

class 2	No. cycles (43)	x avg. DI for class 2 (.719)	= 30.95
class 3	No. cycles (35)	x avg. DI for class 3 (.888)	= 31.08
class 4	<u>No. cycles (9)</u>	x avg. DI for class 4 (1.69)	= <u>15.21</u>
	total = 87		total = 77.24

	PERCENT OF TOTAL CYCLES	PERCENT OF TOTAL DI
class 2	43/87 x 100 = 49.4%	30.95/77.24 x 100 = 40.1%
3	35/87 x 100 = 40.2%	31.08/77.24 x 100 = 40.2%
4	9/87 x 100 = 10.3%	15.21/77.24 x 100 = 19.7%

the average daily diggability index DI is calculated based upon class 2, 3 and 4 only.

Table 8.11.1 - Shift Diggability Summaries - daily average diggability indices by class and degree of influence (%).

DATE AUG.	CUT RATIO CLASS	PERCENT OF TOTAL CYCLES	AVERAGE DI/CLASS	PERCENT OF TOTAL DI	DI
15th	2	20.0%	0.512	7.3%	1.41
	3	23.3%	1.233	20.4%	
	4	56.7%	1.797	72.3%	
17th	2	37.9%	0.546	23.4%	0.87
	3	43.1%	0.795	39.4%	
	4	18.9%	1.704	37.2%	
18th	2	13.7%	0.562	5.8%	1.32
	3	27.4%	1.430	29.6%	
	4	58.8%	1.451	64.6%	
19th	2	36.7%	0.574	26.1%	0.81
	3	44.9%	0.927	51.4%	
	4	18.4%	0.993	22.5%	
20th	2	25.7%	0.388	14.8%	0.67
	3	36.5%	0.725	39.4%	
	4	37.8%	0.813	45.8%	
21st	2	14.0%	0.822	9.3%	1.24
	3	68.0%	1.296	71.1%	
	4	18.0%	1.354	19.6%	
22nd	2	36.7%	0.588	20.6%	1.05
	3	48.5%	1.255	58.1%	
	4	14.7%	1.509	21.2%	
23rd	2	45.0%	0.248	23.8%	0.47
	3	38.7%	0.573	47.1%	
	4	16.3%	0.844	29.1%	
24th	2	45.2%	0.627	29.6%	0.95
	3	38.7%	1.025	43.3%	
	4	14.5%	1.784	27.1%	
25th	2	49.4%	0.719	40.1%	0.88
	3	40.2%	0.888	40.2%	
	4	10.3%	1.689	19.7%	

Structuring the data as per Table 8.11.1, presents the data in a manner that provides an appreciation of the average digging "effort" per day (on the basis of DI), in addition to an

indication of the "nature" of the digging (range of trajectories employed). The daily average DI by itself, gives no indication of potential reasons behind variation in the relative digging effort from one day to another. For example, consider the results for August the 20th and 21st: The average cut ratio for each is similar 0.93 and 0.96 respectively, the DI's are however, quite different 0.67 on the 20th versus 1.27 on the 21st. Although the average cut ratios are similar, the distribution of trajectories (cut ratios) is quite dissimilar. On the 20th, almost 46% of the diggability index is accounted for by class 4 cut ratios. This, coupled with the low average DI for cut ratio class 4 of only 0.813, indicates that the operator took advantage of a very well broken, loose toe. The same operator on the 21st encountered harder toe and an increase in fragmentation size (D_{50}) from 26 cm on the 20th to 52 cm. On the 21st 71% of the DI is accounted for by class 3 cut ratios. Clearly, while the average cut ratios for both days were very similar the nature of the digging for each day was quite distinct.

Structuring the data as per Table 8.11.1 also aids in accounting for any bias due to variations in operating practice. For example, it was demonstrated in section 8.10.2 that when operators 3 and 4 were presented with very easy digging conditions; August 20th and 23rd respectively, each exercised a unique digging practice (refer Figure 8.10.4). By inspecting the SDS's for the 20th and 23rd, it is clear that the digging practices were distinct. Operator 3 on the 20th elected to employ a greater percentage of deep class 4 cut ratios. The effort associated with these deep cuts accounted for 46% of the average DI for that day. On the other hand, operator 4 on the 23rd employed a higher percentage of shallower class 2 and 3 cut ratios.

The SDS for August 18th indicates that almost 60% of the dig cycles fell within class 4 cut ratios, accounting for almost 65% of the DI. The high percentage of class 4 ratios coupled with the

high average DI for these deep cuts (1.45) indicates a requirement to pursue these difficult trajectories. When operator 3 pursued frequent class 4 trajectories on the 20th, in easy digging, it was acceptable because the average DI associated with these deep cuts was very low (0.81). On the 18th however the average DI for class 4 cut ratios is very high. Clearly, the operator would not pursue such difficult trajectories were they not required in order to fill the dipper. The 18th was characterized by very coarse fragmentation; $D_{50} = 62$ cm. It has been shown that deep trajectories are demanded in order to force larger fragmentation into the dipper (this will be addressed in detail in 8.13 - Correlation of DI's with Blast Designs).

Much of the data of Table 8.11.1 is presented in a graphic format in Figures 8.11.1 and 8.11.2,. These illustrate the daily percent of dig cycles per cut ratio class (2, 3 and 4) and the percent influence each class has on the diggability index (DI) per day.

In summary, although dipper trajectories have been shown to significantly influence the monitored motor responses, their direct inclusion in an equation for the diggability index, is at present impractical. The diggability indices determined have proven sufficiently adequate to identify changes in muckpile diggability. The Shift Diggability Summary (SDS), is recommended as a means of enhancing the value of the monitored data, by incorporating a measure of the influence of dipper trajectory from one shift to another. In this fashion, variation in operating practice is identified with regard to changes in digging effort, as well as the muckpile characteristics that may necessitate these changes (ie hard toe, coarse fragmentation etc.).

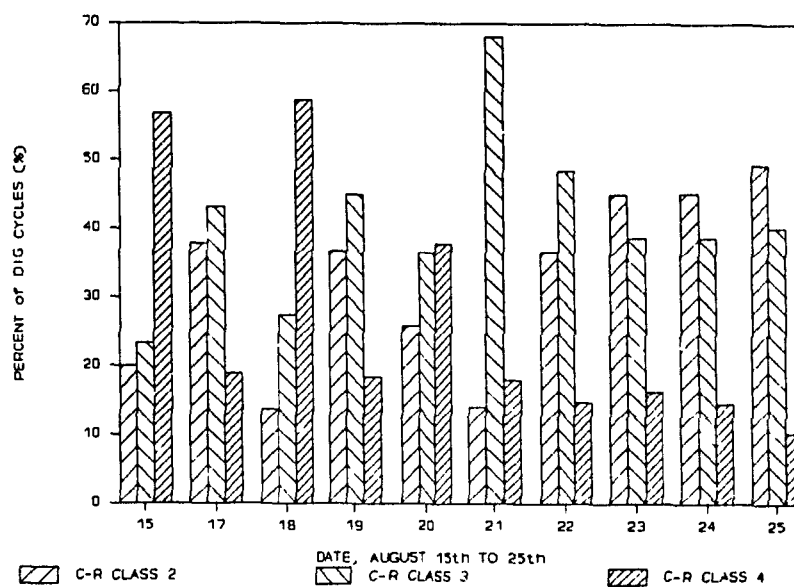


Figure 8.11.1 - Percent of daily dig cycles from cut ratio classes 2, 3 and 4.

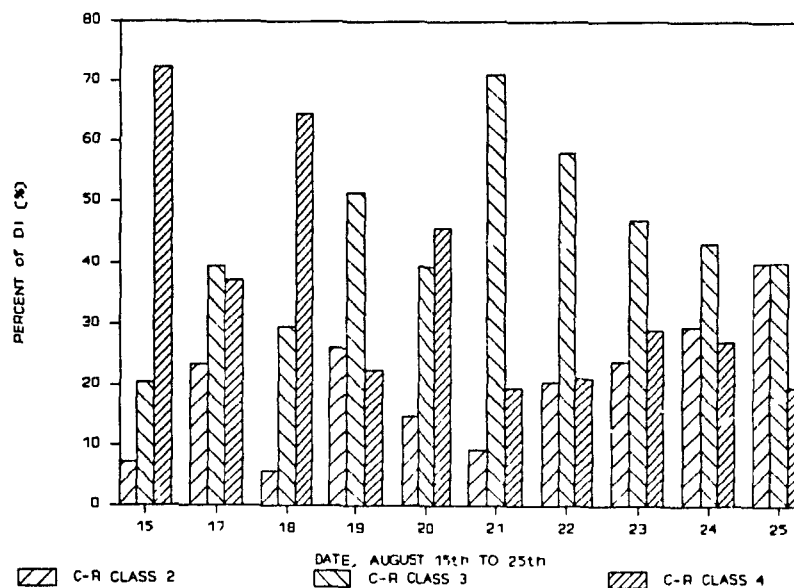


Figure 8.11.2 - Percent influence on daily DI from cut ratio classes 2, 3 and 4.

8.12 TIME STUDIES - VALIDITY AND RELATION TO DIGGABILITY

Time studies have traditionally been regarded as a quick and economical means of rating muckpile diggability. The technique assumes that digging rates are a function of digging effort; hard toe, tight muck and coarse fragmentation will be reflected by longer mean dig cycle times. It is also assumed that operators, with equivalent expertise, will perform equally in like conditions (Chiappetta, 1983). The current research, with data on dig cycle times and actual measures of digging effort has revealed:

- a). Dig cycle times are not specifically related to digging effort
- b). Operators of equivalent expertise, appear to maintain characteristic digging rates irrespective of digging conditions.

Operator Influence

Table 8.12.1 lists the average dig cycle times as recorded by the General Electric monitor, the Polycorder and as determined by a manual cycle time study, conducted on the video records per day of monitoring.

It is clear from Table 8.12.1 that operator number 1 consistently has the longest mean dig cycle times. The dig cycle times determined through the G.E. monitor are considered the most accurate since they are determined on the basis of dipper position. The polycorder dig cycle times were established based on the motor responses and are considered therefore marginally less accurate than the G.E. times. Finally, the cycle times established through review of the video tapes are considered the least accurate. Note the progressive increase of both the standard deviations and ranges from the G.E. data to the Polycorder and then the video tapes. Figure 8.12.1 is a box and

whisker plot of operator cycles times based on the G.E. data (Note operator 5 was a replacement on the shovel, as such his performance is not considered in further depth).

Table 8.12.1, Summary dig cycle times per operator as determined by each piece of instrumentation.

<u>Operator No.</u>	<u>Sample Size</u>	<u>Average dig time (sec.)</u>	<u>Median</u>	<u>Standard deviation</u>	<u>Range</u>

GENERAL ELECTRIC MONITOR

1	216	12.01	11.30	2.82	16.00
2	126	8.53	8.30	1.49	7.60
3	141	9.30	9.10	1.57	9.90
4	<u>167</u>	9.47	9.00	1.69	8.40
total	650				

POLYCORDER

1	569	10.82	10.66	2.69	25.42
2	214	8.86	8.20	2.16	13.94
3	262	9.21	9.02	1.94	16.00
4	<u>409</u>	9.27	9.02	1.97	13.94
total	1454				

VIDEO TAPES

1	1306	12.80	12.22	3.17	32.03
2	651	11.05	10.65	2.07	19.44
3	638	11.63	11.24	2.59	20.01
4	<u>809</u>	11.16	11.15	1.88	18.87
total	3404				

The true significance of video tape time studies is that they have allowed a comprehensive traditional time study, to be conducted in conjunction with shovel performance monitoring, through instrumentation. In this manner it has been possible to relate the average cycle times with measures of digging effort. Notably, the perspective afforded the analyst (person recording cycle times) from the shovel operators cab is extremely favourable. Most time studies are conducted from less than optimal locations in the pit, with a poor field of view, some distance from the shovel. Of additional importance is the fact that all 36 hours of video tape (18 tapes; 2 hours per day for

19 days) were analyzed by the **same** analyst. This, coupled with an ability to review a tape should a cycle be missed or miss interpreted, labels this one of the more accurate time studies undertaken in surface mining.

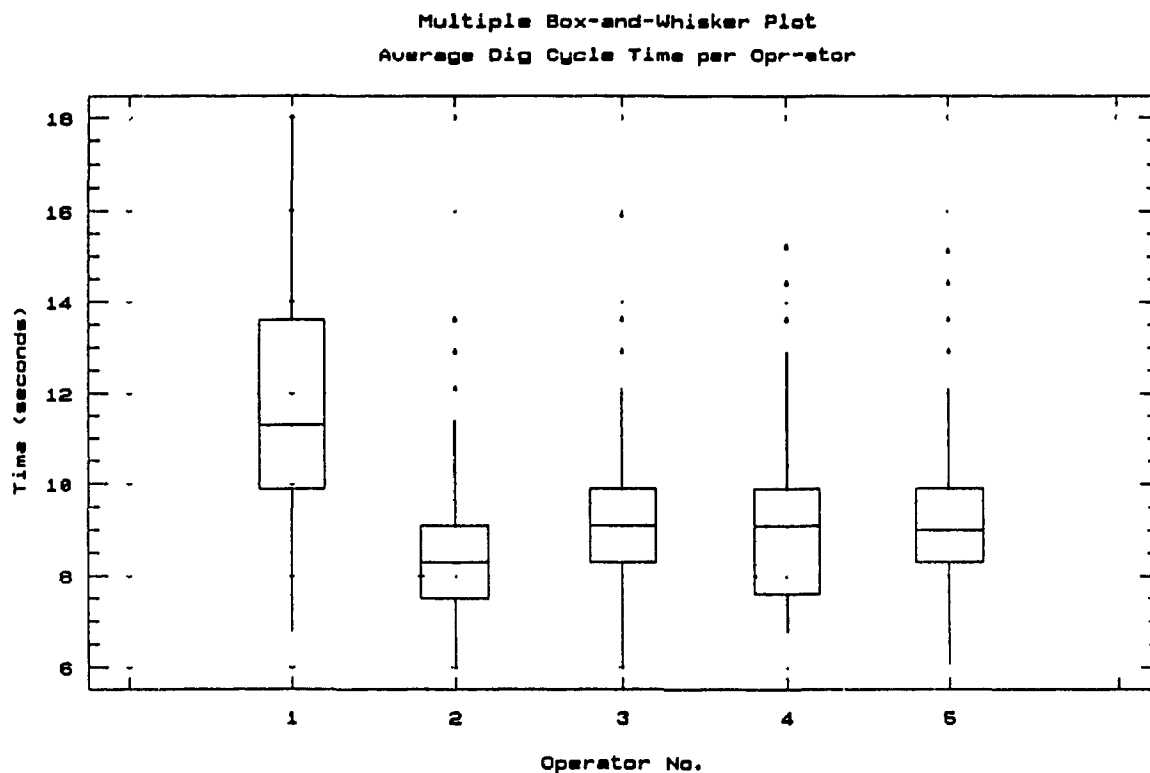


Figure 8.12.1 - Average dig cycle times per operator.

It has been concluded that manual time studies do reflect the actual dig cycle times established through shovel instrumentation. Their relative inaccuracy stems from human error in identifying the true start and end of the dig cycle. The position data used to derive the G.E. based dig times, provided a precise means with which to identify the duration of dig cycles. The manual time study has revealed operating practice to be of greater influence than changes in muckpile diggability.

A comprehensive tabulation of the results of the video tape based time studies, is presented in **Appendix H**.

Dig Cycle Times and Digging Effort

Table 8.12.2 documents the relation between the average dig cycle time and associated levels of digging effort as defined by the classes or ranges of diggability index DI. The same data is presented graphically in Figure 8.12.2 (for the G.E. derived dig cycle times) which shows that no changes mean dig cycle times are associated with increases in digging effort, as defined by the diggability index; DI. Figure 8.12.3 illustrates the lack of relation between dig cycle times as determined by the G.E. monitor, the Polycorder and the video based time studies and the diggability index DI per day of monitoring. Clearly, there exist no correlation between average dig times and levels of digging difficulty. The fact that average dig cycle times do not decrease with levels of digging difficulty is contrary to a popular assumption. Note, it is possible that variation in dig cycle times are associated with greater changes in digging difficulty than those observed within the current study. However, the range of digging conditions encountered are felt to have been sufficient to motivate a change in dig cycle times should they truly be reflective of diggability. The fact that dig times have not varied with known levels of digging difficulty appears explainable on the basis of previous observations in the current research.

The operator is aware of the digging conditions about to be pursued. In hard digging (ie coarse fragmentation, hard toe etc.) the muckpile is attacked with a lot of crowd and a deep cut in a short trajectory. The speed controlled hoist motor, attempts to maintain as high a velocity as possible. The deep cut however tends to slow dipper advance and results in a higher hoist motor torque. The operator, aware of the increased digging effort, appears to compensate by limiting the time in

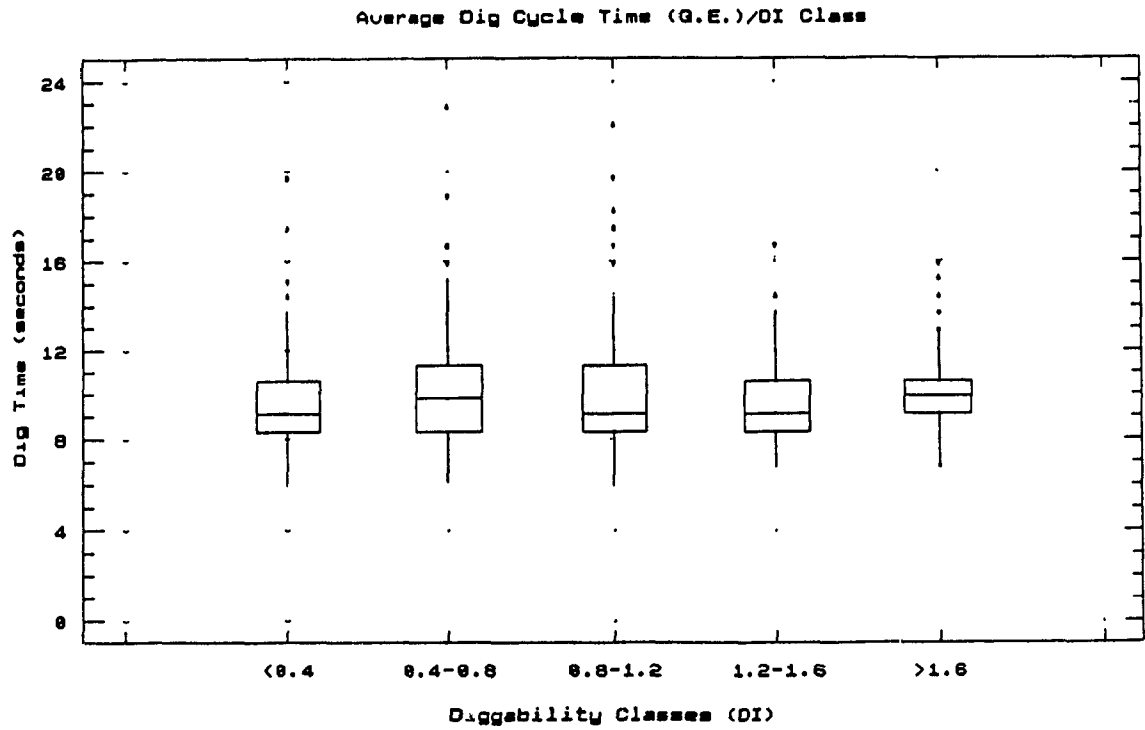


Figure 8.12.2 - Box and whisker plot - relation between average dig cycle times and diggability effort, DI

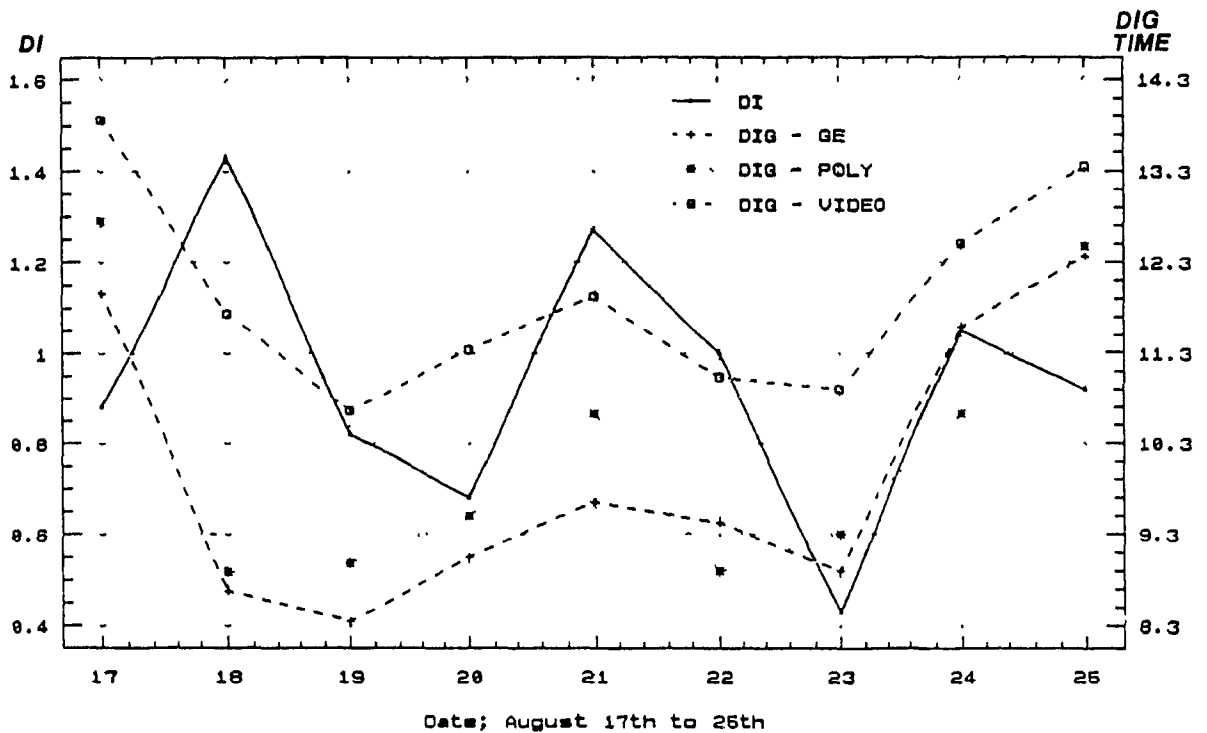


Figure 8.12.3 - Plot of average dig cycle times and diggability index, DI, per day of the study.

the bank. The net result is a slower dipper velocity for the dig cycle, with a shorter trajectory. The dig time however, approximately equivalent to the length of the short trajectory divided by the slow dipper velocity, equates to an average cycle time. Conversely, in easy digging, long shallow passes at higher velocity, equate to average dig cycle times

Table 8.12.3 demonstrates that differences in dig cycle time associated with increases in digging difficulty are only apparent within the performance of individual operators. Clearly however, these individual relations will be obscured when the performance of all operators are considered. Once again, operating practice is seen to dominate. It is important therefore, that time study based assessments of diggability, be based solely on the performance of a single operator.

Table 8.12.2, Average dig cycle times per diggability class G.E. and Polycorder data.

<u>DI Class</u> <u>(range)</u>	<u>Sample</u> <u>size</u>	<u>Average dig</u> <u>time (sec.)</u>	<u>Median</u>	<u>Standard</u> <u>deviation</u>	<u>Range</u>
GENERAL ELECTRIC MONITOR					
<0.4	135	9.67	9.10	2.40	13.70
0.4-0.8	233	10.11	9.80	2.61	16.70
0.8-1.2	143	10.06	9.10	2.97	16.00
1.2-1.6	72	9.83	9.10	2.21	9.90
>1.6	96	10.09	9.90	1.83	9.10
POLYCORDFT					
<0.1	180	10.34	9.84	2.37	12.30
0.1-0.2	582	10.05	9.84	2.40	15.18
0.2-0.3	386	9.43	9.02	2.15	13.94
0.3-0.4	204	9.45	9.02	2.46	17.22
>0.4	166	10.06	9.02	3.14	26.24

Table 8.12.3, Average dig times and diggability indices by operator per day.

Operator No.	Date	DI	Average dig time (sec.) G.E. based
1	Aug 17	0.88	11.96
1	Aug 24	1.05	11.58
1	Aug 25	0.92	12.36
2	Aug 18	1.43	8.67
2	Aug 19	0.82	8.34
3	Aug 20	0.68	9.05
3	Aug 21	1.27	9.64
4	Aug 22	1.00	9.64
4	Aug 23	0.43	8.90

It must be emphasized that the time studies have been based upon the performance of four of the mine's best shovel operators. All have a minimum of 15 years of experience. If clearly identifiable differences in operating practice exist amongst so select a group of operators, while operating on the same shovel, then it is almost certain that more pronounced operator variability exists elsewhere: For example, in third world nations with a less skilled labour pool, or in less refined studies involving the performance of more than one shovel, shovel model and operating crews.

Dig Cycle Times and Fragmentation.

Research by Bouden-Romdhane et al., 1987, reported upon a two year study in major quarries in France to assess the influence of several blast variables. A component of the research aimed to relate the loading rates of a front end loader with resultant fragmentation from test blasts. Although no data is

presented, the results identified loading rate as related to fragmentation and muckpile expansion. Results of the current research suggest however, that for a large cable shovel, there is no consistent correlation between dipper loading (dig time) and fragmentation. Figure 8.12.4 is a plot of dig cycle times as determined by the Polycorder and fragmentation D_{50} , for each of the 19 days of monitoring in August and October 1988.

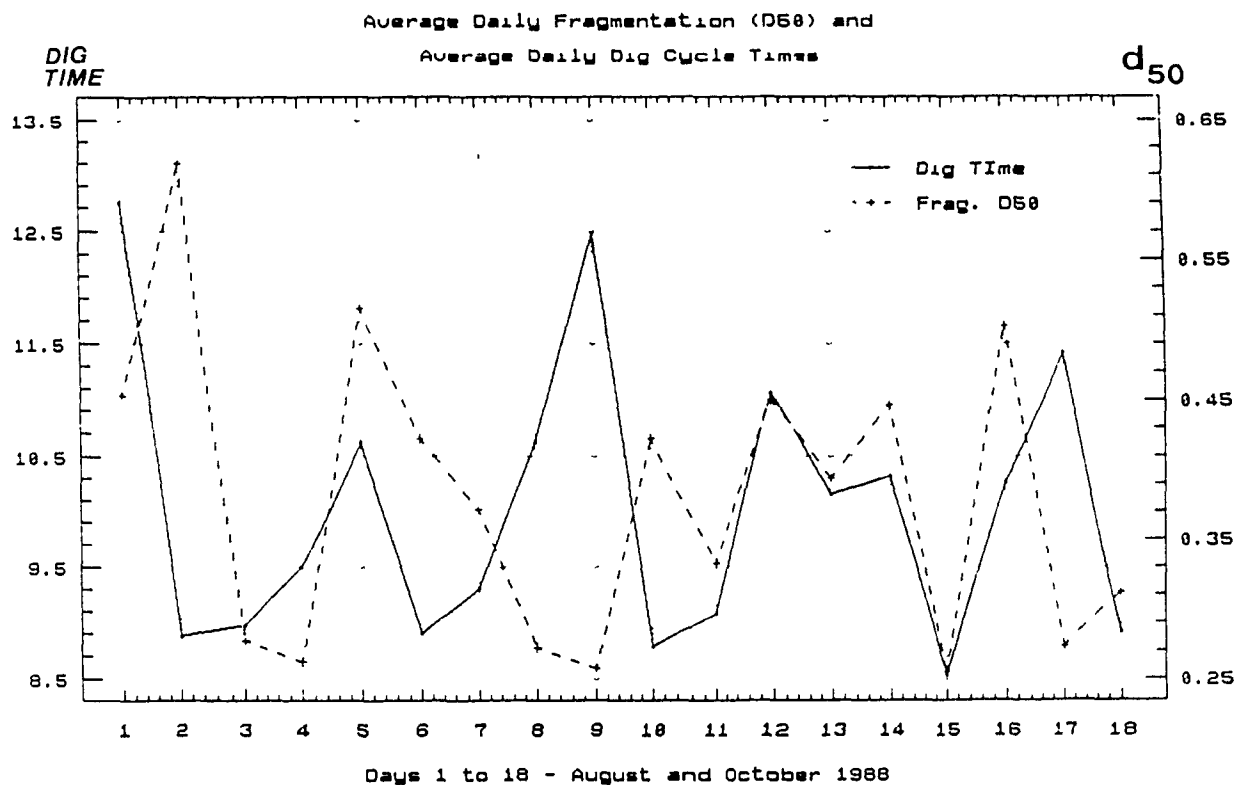


Figure 8.12.4 - Plot of average dig cycle times and mean fragmentation D_{50} per day of monitoring

8.13 CORRELATION OF DIGGABILITY WITH BLAST DESIGN ELEMENTS.

This section will, in advance of concluding the chapter, attempt to relate the assessments of diggability (DI) to elements of blast design employed in the study. In total, three blasts were evaluated, amongst which the following unique blast environments, or zones existed:

1. Square blast pattern zones in massive sandstones.
2. Square blast pattern zones in weaker bedded ground.
3. Back of a square blast pattern.
4. Equilateral blast pattern zone.
5. Heavily loaded zone in an equilateral pattern

It must be appreciated that the range of digging difficulty generated by the test blasts, was quite narrow. It was the opinion of all operators studied, that the worst digging conditions encountered within the test blasts, were merely equivalent to average diggability on a mine wide rating (digging conditions are often much worse than conditions generated by any of the test blasts). The fact that the calculated diggability indices have proven sensitive within so narrow a spectrum of digging conditions is encouraging. The narrow scope, and extent of variation in blast design criteria however, has made relating post blast conditions back to original blast design elements difficult. Typically, blast designs are compared with one another on the basis of the results of several entire bench blasts, not components of a single trial.

Fortunately, it has been possible to relate the most prominent variations in blast design and bench geology to the resultant diggability for such zones; the heavily loaded portion of blast EM#1, and the back portions of blast EZ#3 characterized by massive sandstones. Episodes of hard toe (poorly fractured ground at the base of the muckpile) encountered during the study

were not the product of poor blast design. As will be demonstrated, they represent a "localized failure" of a blast, which results from too much fill at the bottom of a blasthole, over drilling of designed blasthole depth, or are a consequence of an operator working below design grade. The fact that the diggability index reflected hard toe digging, gives further testament to it's diagnostic capability.

Relating shovel performance (diggability) to blast design has been based primarily upon the resultant fragmentation. This because, fragmentation was the most quantifiable aspect of muckpile characteristics. A relation has been identified between mean fragmentation size (D_{50}) and the average daily diggability indices (DI). This will be addressed in detail in section 8.13.1.

Muckpile looseness is considered to be another important consideration in determining blast efficiency and muckpile diggability. This aspect of the muckpile is however much more difficult to quantify. Traditional measures of looseness are based upon; muckpile profile, the size of the power trough at the back of the blast (ie how far has the blast laterally shifted the rockmass) and operator assessments of diggability. Of the three, operator feed back is regarded as the most valuable.

The blasts evaluated all developed good power troughs and muckpile profiles throughout were equivalent. Since no visual evidence was available to indicate changes in muckpile looseness, operator assessments were relied upon. Good correlation exists between the diggability index and operator ratings of looseness.

The following is a description of digging conditions as they pertain to elements of blast design and bench geology, for each day of monitoring with the General Electric monitor in blasts EZ#3 (square pattern) and EM#1 (equilateral pattern). It has been concluded, that the Polycorder-derived diggability indices for blast EM#4 (square pattern), are not sufficiently sensitive (due

to slow sampling rate) to justify a similar comparison of diggability with blast design elements.

August 15th-Hard Digging DI=1.41 Cut Ratio=1.04 D50=52 cm
August 17th-Avg. Digging DI=0.88 Cut Ratio=0.83 D50=45 cm

On August 15th the shovel was midway through blast EZ#3; an area characterized by a massive sandstone cap in the collared portion of the bench. The coarse fragmentation in this zone, from rilling of the poorly fragmented massive sandstone, significantly influenced diggability. Diggability was seen to improve at about three quarters through the same blast - August 17th. However, this was evidently a consequence of the operator's (operator 1) shallower average cut ratio, as fragmentation was seen to have improved only slightly.

August 18th-Hard Digging DI=1.43 Cut Ratio=1.11 D50=62 cm

On August the 18th the shovel was excavating the back of blast EZ#3. At this location hard digging is assumed to have resulted through diminished heave of the blast's final burdens. This, in addition to the influence of the weakly jointed massive sandstone, resulted in a very coarse blocky muckpile.

There exists evidence that fragmentation exerts a reasonable degree of influence over diggability. This topic will be addressed in detail in section 10.13.1, to follow.

August 19th-Avg. Digging DI=0.82 Cut Ratio=0.88 D50=28 cm
August 20th-Easy Digging DI=0.68 Cut Ratio=0.93 D50=26 cm

August 19th marked the entry of the shovel into blast EM#1. At this bench position, the extent and influence of the massive sandstone layer began to diminish. Furthermore, the shovel was at the blast initiation point, and hence benefited through good heave and theoretically improved looseness.

On the 20th, the shovel was clear of the massive sandstone layer and into that portion of the bench characterized by more

variable and weaker lithologies (inter-bedded mudstones, coaly-mudstones and siltstones). Fragmentation was very good and the operator commented on the excellent looseness of the muckpile, especially at the toe. There is no explanation for the improved looseness at this location. (The complexities associated with the interaction of explosive energy and an heterogenous rockmass are not always explainable on the basis of a single trial).

The diggability index is seen as especially well suited/sensitive to evaluating muckpile looseness. Since the fragment size distribution for August 20th is no different from the 19th, the improved ease of diggability (even with a higher average cut ratio) can only have resulted from increased looseness. Since no visual evidence of improved looseness was apparent or expected, the diggability index has demonstrated an ability to measure that which is otherwise not identifiable with traditional means (ie. muckpile profile).

August 21st-Hard Digging DI=1.27 Cut Ratio=0.96 D50=51 cm

On August 21st, the shovel was engaged in handling coarser fragmentation and digging hard toe. Hard toe conditions (poorly fractured ground) were confirmed at the end of the monitoring period, when surveyors informed the operator he was about two feet below designed bench grade.

August 22nd-Avg. Digging DI=1.00 Cut Ratio=0.79 D50=42 cm

On this day the shovel was working a muckpile of average looseness, with fragmentation a bit coarser than average for the study.

August 23rd-Easy Digging DI=0.47 Cut ratio=0.76 D50=37 cm

On the 23rd, the shovel was working that portion of blast EM#1 in which very high strength explosive columns were intentionally employed. The result was a marked improvement in muckpile looseness. It was originally theorized that the higher strength explosive, with their associated higher VOD and brisance

(shattering) energy, would have generated finer fragmentation. It appears, as discussed in chapter 5, that the influence of the rockmass structure (highly bedded and jointed) at this location dominated. The net effect appears to have been improved displacement of the structurally predefined, in-situ blocks.

August 24th-Avg. Digging DI=1.05 Cut Ratio=0.84 D50=27 cm
August 25th-Avg. Digging DI=0.92 Cut Ratio=0.79 D50=26 cm

On both of these days the shovel was working muckpiles characterized by very good fragmentation and average looseness.

8.13.1 Relation Between Fragmentation and Diggability Index DI.

Figure 8.13.1 is a plot of the diggability index (DI), and mean fragment size (D_{50}), determined for each day of monitoring. A relationship between the two appears to exist, and is accounted for, in part, on the basis of dipper trajectories adopted by operators when confronted with coarse fragmentation.

Figure 8.13.2 presents plots of average trajectory ranges recorded from August 18th (D_{50} =62 cm) and August 19th (D_{50} =28 cm). Note that same operator was present for both days. It is clear that a change in his digging practice is prompted by coarser fragmentation. Figure 8.13.3 is a plot of the trajectory ranges from the entire ten days of shovel monitoring from August 15th to 25th (note the ranges are based upon the most frequent trajectories per day). Indicated within this global range, are the trajectories associated with August 15th (D_{50} =52 cm), 17th (D_{50} =45 cm), 18th (D_{50} =62 cm) and 21st (D_{50} =52 cm), days upon which the coarsest fragmentation was encountered.

It is believed that deeper cuts are required when handling coarser fragmentation, in order to manoeuvre (ie. bulldoze) these larger blocks into the dipper. This does not mean that higher DI's associated with coarser fragmentation are solely the product

of deeper cuts through the muckpile. When average cut ratios per day are compared with fragmentation (D50) a weaker relation exists.

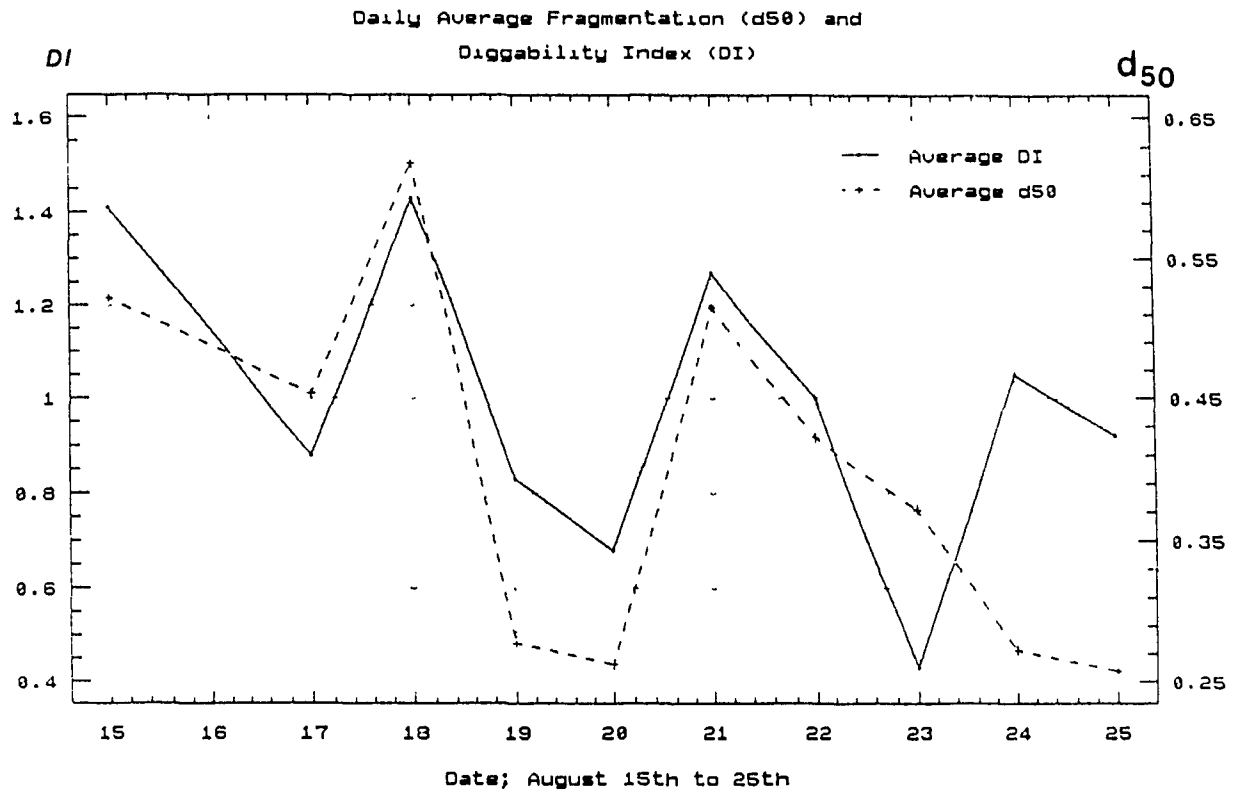


Figure 8.13.1 - Plot of diggability index, DI, and mean fragmentation D_{50} .

This is believed due to: a) The average cut ratio by itself has been shown to be ambiguous. Inspection of the relative proportions of cut ratio ranges (classes 2,3 and 4) provide greater insight into the true nature of shovel activity. b) The coarser fragmentation exerts greater resistance to dipper travel due to higher strength interlocking of blocks.

As mentioned in section 6.3, the fragmentation shape will exert, an as yet undetermined degree of influence over muckpile diggability. Due to variations in the packing densities

associated with different shape fragments, two identical size distributions characterized by different fragment shapes would theoretically offer different levels of diggability. For example, tabular fragments, which may typically result from layered rock units, would almost certainly assume a "packing density" and interlocking, distinct from that associated with a more cubic fragmentation, say from massive units. The extent to which measures of diggability would vary from one characteristic fragment shape to another, can not be assessed on the basis of results from the current research, since fragmentation shape was not seen to vary noticeably.

It must be assumed however, that although shapes may vary from one mine site to another, the characteristic shape of fragments within each site will remain fairly constant due to regional structural controls (e.g. no "significant" variations in fragment shape were observed through out the mine site). And regardless of shape, an increase in the size of the material will result in an increase in the required digging effort.

The relationship identified therefore, between fragmentation size distribution and diggability, should ultimately be tempered by the introduction of some shape factor, to account for the variations in packing density and digging resistance, offered by different shape fragments ie. cubes, parallelepipeds, discs etc.

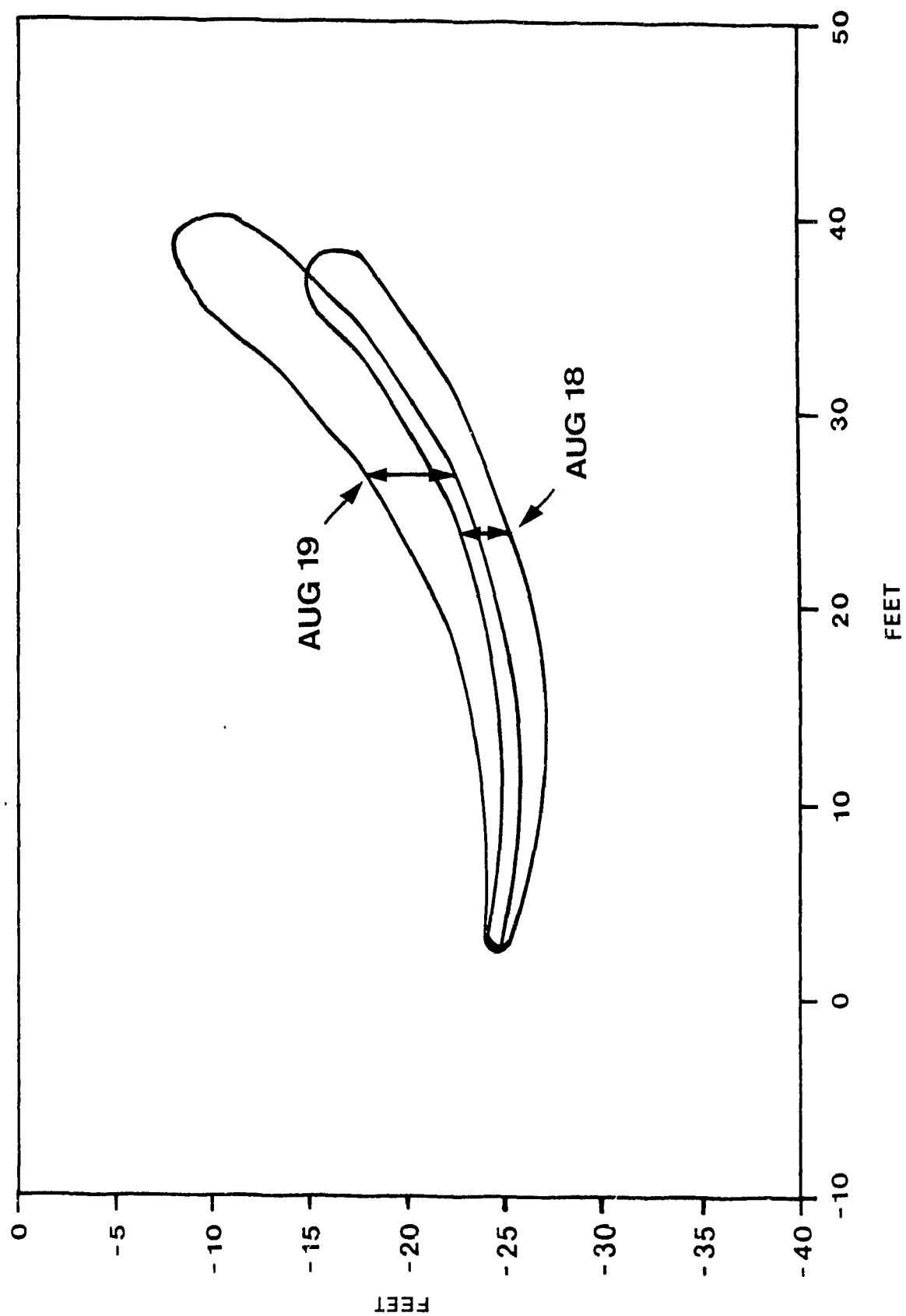


Figure 8.13.2 - Dipper trajectory ranges, August 18th and 19th.

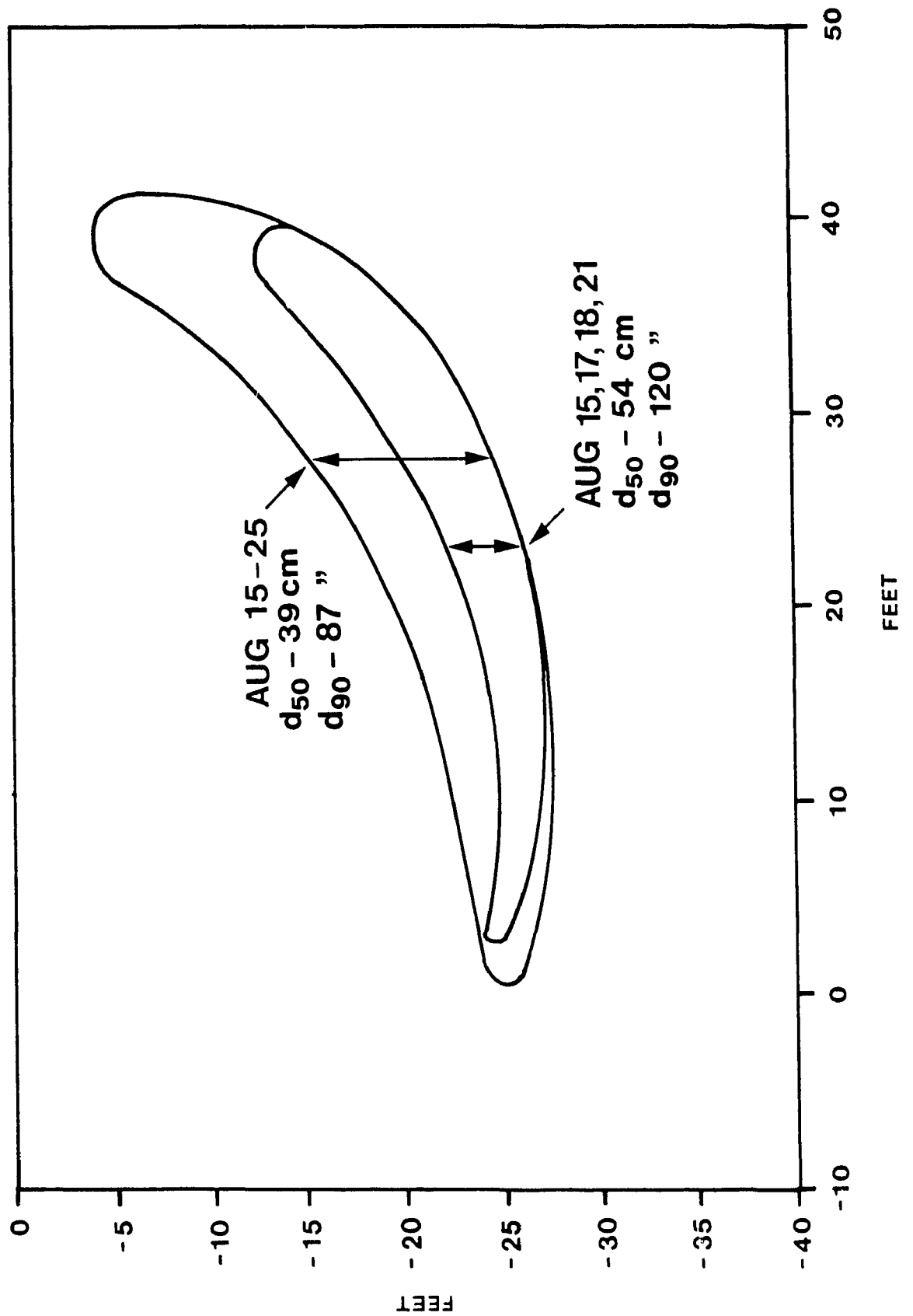


Figure 8.13.3 - Isolation of dipper trajectory ranges associated with coarse fragmentation; August 15th, 17th, 18th and 21st.

Pandey (1974), in describing blasting in an iron ore quarry, cited the following formula to determine the maximum size fragments a shovel should handle, without their becoming blocked in the bucket (dipper).

$$X = 0.75 \times \sqrt[3]{e} \quad \text{..... (8.13.1)}$$

X = size of ore

e = capacity of bucket

The monitored shovel (P&H 2800XP) was equipped with a 30 cubic yard (23 cu.m) dipper, the above formula, suggests it could handle fragments less than 2.33 yards or 2.13 metres in diameter. During the course of the study, the coarsest fragmentation encountered (D_{90}) was less than 1.4 metres in diameter. It is assumed therefore, that the shovel was never asked to perform outside of it's design criteria as far as fragmentation is concerned.

8.14 SUMMARY.

A diggability index (DI) based on hoist voltage and current responses has proven a reliable indicator of muckpile diggability. Ratings of diggability based on the DI have correlated very well with operator based ratings of diggability. Data on dipper trajectories have been interpreted and found to exert a significant degree of control over shovel motor responses. The path followed through the muckpiles experienced had a more pronounced influence over motor response than did the characteristics of the muckpile itself. Diggability can therefore not be interpreted without due regard to aspects of dipper trajectory.

The influence of dipper trajectory (and hence operator variability) has been incorporated in assessments of diggability through Shift Diggability Summaries (SDS's), determined for each day of the study period. The SDS provides a concise breakdown of the range and frequency of trajectories employed for the shift. This coupled with ratings of effort or diggability (DI's) for each trajectory range, provides insight into both the degree of digging effort and reasons behind particular digging practices adopted for the shift.

A diggability index proposed by prior researchers, based on crowd motor response (CVDI) has been not proven valid for this study environment. The CVDI displayed an inverse relation with respect to known levels of digging difficulty and appeared sensitive to individual operating characteristics.

Operator variability, dismissed by prior researchers as insignificant, would appear to have been due to design of the field experiments. It has been determined through the current research that variations in operating practice do exist and can, if unaccounted for, bias assessments of diggability. Issues of

variation in operator practice can only be addressed through knowledge of dipper trajectory. Importantly, if operator variability was identifiable amongst so experienced a group of operators, working on the same shovel, then it is certain that the influence of operator variability in other, less efficient operations would be greater.

Time studies, considered to be fast and economical means of rating diggability, have proven unreliable to this end. Within the current study, there exists no relation between average dig cycle times and calculated levels of digging difficulty. Rather, cycle times have proven an operator dependant parameter. An operator's mean dig cycle time will vary only slightly between easy and hard digging situations.

A relationship between fragmentation size distribution and shovel performance has been identified. It suggests that coarser fragmentation demands greater digging effort. The increased digging effort being a product of deeper trajectories required to load the larger fragments, as well as the influence of greater interlocking of such blocks, which result in increased resistance to dipper travel. The effects on diggability of variations in fragment shape, have not yet been fully investigated due to the lack of variability in said parameter during the course of the study. Still, the influence of fragment shape has been identified as an area of concern to future endeavors, as shape will influence material packing densities and hence muckpile looseness and ultimately; diggability.

9.0 CONCLUSIONS

It has been demonstrated that the monitored shovel performance parameters are responsive to variations in muckpile diggability. Allied investigations into the influence on diggability of bench geology, blast design criteria and fragmentation size distribution, has lent support to this conclusion. The current research has succeeded in the development of instrumentation, monitoring and interpretive procedures appropriate to the use of an electric mining shovel as a diagnostic tool in rating muckpile diggability. In this sense, the shovel can be adapted to provide continuous, realistic and much valued feedback on the relative success of individual blasts, as part of ongoing blast optimization programs.

This thesis has aimed to advance the following areas:

1. The development of micro-processor based instrumentation appropriate to the performance monitoring of electric mining shovels.
2. The interpretation and analysis of monitored shovel performance parameters to relate their responses to changes in muckpile diggability.
3. The understanding of the influence of muckpile fragmentation and looseness on diggability and shovel performance.
4. The application of shovel performance monitoring technology towards optimization of the drilling, blasting and loading processes in surface mining operations.

A commercial shovel production monitoring system (General Electric's SPM-8000) was adapted with the cooperation of the manufacturer, to provide real time data on key shovel performance parameters. Hoist motor rpm (armature voltage), torque (armature current), hoist rope position and crowd arm extension were recorded continuously at a 100 ms sampling interval during monitored periods. The data afforded through this system has proven of critical benefit. It has been shown that a 100 ms sampling interval was required to record hoist motor response of sufficient resolution to permit identifying changes in diggability. The position transducers installed on the hoist motor drum and crowd motor have proven sufficiently robust to withstand the rigors of the physical environment. They have also provided the much needed data on dipper position that has been instrumental in confirming several of the conclusions drawn from the current study.

In addition, the G.E. system was used in its conventional capacity; as a production monitor, to record data on dipper load weights and the various shovel cycle times. The data on dipper load weights has been of use in characterizing the operating practices of the shovel operators investigated. A relationship has also been established between the response of the hoist motor current and associated dipper load weights.

Secondary instrumentation included the adaptation of a commercial (micro-processor based) data-logger whose primary purpose was to record the performance of the shovel's crowd motor during monitoring. The maximum sampling rate of 820 ms of this instrument did not prove to be adequate.

A suitable monitoring strategy was established that permitted acquisition of a complete performance parameter database during normal mine production operations. Appropriate software has been designed to permit processing, interpretation, analysis and presentation of the extensive data base accrued during field

monitoring.

The current research permitted isolating several of the experimental variables that may have significantly biased prior research (Williamson et al., 1983 and Mol et al., 1987). The experimental design effectively enabled monitoring the performance of a single shovel and set of operators, in a controlled environment, during excavation of more than 500 000 bank cubic yards of waste. Drill monitored performance parameters from an instrumented blasthole drill (Peck, 1989) were successfully used in conjunction with geophysical logging of blastholes, to derive detailed cross sections of bench geology. With this data the extent of lateral variation in bench rock mass characteristics was identified, towards predicting its influence over blast results. A series of different blast designs fired in the test bench, generated muckpiles with unique characteristics against which to compare the records of shovel performance.

It has in the past been suggested that variations in operating practice are insignificant (Williamson et al., 1983 and Mol et al., 1987). This has not been confirmed by the current research. Through monitoring the shovel's hoist rope position and crowd arm extension it has been possible to compute the dipper trajectories for each shovel dig cycle. Only in this manner was it possible to investigate the characteristic digging practices or tactics employed by each operator. Characteristic digging practices have been shown to exist amongst a group of very experienced operators. Furthermore, variations in dipper trajectory have been demonstrated to exert a greater influence over the response of monitored motor parameters than do the characteristics of the muckpile being handled. It is concluded as unreasonable therefore, to establish measures of muckpile diggability without due consideration to dipper trajectory.

The hoist motor was found to be more sensitive to variations in muckpile diggability than the crowd motor. Prior

investigations (Williamson, 1983 and Mol, 1987) have used crowd motor responses to rate muckpile diggability. However, the current research, with available data on dipper trajectory, has demonstrated the crowd motor to be relatively insensitive to variation in muckpile diggability. It was concluded that the response of the crowd motor appeared heavily influenced by operating practices, namely; the amount and rate of crowd.

Numeric ratings of diggability based on the monitored responses of the shovel's hoist armature voltage and current signals have been derived. A diggability index has been established which shows good correlation with digging conditions as observed, and corroborated by operating personnel during field performance monitoring of the shovel. The significance of the diggability index has been improved upon by including and accounting for the influence of dipper trajectory. Variations in dipper trajectory (as they pertain to digging tactics) have been shown to relate to changes in muckpile characteristics such as fragmentation. Therefore, by noting changes in trajectory it has been possible to relate such changes not only to variations in digging effort but also to potential differences in the muckpile characteristics which may have motivated a change in digging tactics.

A comprehensive photographic survey of muckpile fragmentation has established the size distributions of fragmentation handled by the shovel during each day of the study. Records of fragmentation size distribution (mean fragment size D_{50}) have been shown to relate well with the monitored records of shovel performance and ratings of diggability. The influence of fragment shape however, has not been explored within the confines of the current research. This because, the structural regime of the test bench area consistently generated the same shape fragments - orthogonally defined, cubic blocks. Variation in fragment shape may however, exert a significant influence over muckpile diggability. It is recognized that different shape fragments will

assume unique packing densities and interlocking frameworks, hence, influencing muckpile looseness and resistance to dipper travel - diggability. It is intended that future research address the influence of fragment shape on muckpile diggability, in appropriate detail, see Chapter 10.

The design of the experiment sought to intentionally vary blast design elements, in order to generate muckpiles which would offer varying levels of diggability. This, in order to examine how the responses of the monitored performance parameters varied from easy to difficult digging conditions. It was not the central intent of the research to establish "firm" relationships between measures of shovel performance and elements of blast design. This aspect is viewed as the logical next step in the research, having defined within the current investigation, a monitoring strategy which permits identifying variations in diggability. Still however, use was made of the opportunity to "attempt" a correlation between shovel performance and blast design. To this end, rock mass characteristics (geology and structure) throughout the test bench were explored, and blast designs carefully documented (patterns, loading instructions etc.) in order to isolate and assess, as much as possible, the influence of changes in design parameters from each blast. Unfortunately the limited scope of the experiment, when viewed in this regard, made it difficult to establish relationships between records of shovel performance as defined by the diggability index - DI, and elements of blast design.

The third blast excavated, EM#4, was drilled in a square pattern of sufficient extent to cover the bench areas defined within blasts EZ#3 (square) and EM#1 (equilateral). In this manner it would have been possible to evaluate the same blast pattern in different rock mass settings; i.e. the massive sandstone units towards the east of the bench in EZ#3 and the weaker, more variable geology towards the west of the bench in blast EM#1. Additionally, the latter half of EM#4 (square), occupying the western extreme of the test bench area, would have

permitted evaluation of square and equilateral patterns (EM#1) in equivalent rock mass settings. Unfortunately, only the Polycorder (data-logger) was available during the excavation of blast EM#4. The sampling rate of this instrument, only 820 ms, proved inadequate to derive measures of diggability.

It has however, been possible to relate the diggability indices determined by the G.E. instrumentation, with the broadest changes in bench characteristics and blast design, between blasts EZ#3 and EM#1: The most difficult digging and highest associated DI's, were encountered towards the back of blast EZ#3 in massive sandstone units, where the coarsest fragmentation was seen to result. The easiest digging conditions (lowest DI) resulted in that portion of blast EM#1 in which uncommonly high strength explosive columns were employed.

While this component of the research has not succeeded in establishing a definitive relationship between the diggability index and elements of blast design, the DI has proven responsive to broad changes in elements of geology and blast design. It appears logical to conclude therefore, that in a more expansive deployment of the established shovel monitoring technology, relationships between the calculated DI's and blast design would be confirmed (see chapter 10 - Recommendations for Future Work). Values of the DI may then be used to establish the relative efficiencies of the various blast designs evaluated, during blast optimization programs of realistic scope.

The DI is viewed as an indicator of changes in digging effort associated with the excavation of muckpiles. It is known that fragmentation size and looseness influence diggability (the influence of fragment shape has yet to be fully explored). The diggability index will only respond to, or indicate a change in digging effort. Whether this be a result of changes in looseness, fragmentation size, or hard toe for example, must be evaluated in order to affect appropriate changes to blast design. It is in determining what type of digging conditions existed, not merely the digging effort, that the shift diggability summaries could

prove beneficial. For example, if fragment size and trajectory ranges remained equivalent from one day to another, any change in the DI may be attributable to a variation in muckpile looseness. Operator feedback would of course be of added benefit in establishing the nature of digging conditions.

Finally, a shovel cycle time study has been conducted, based on over 36 hours of video-taped records of shovel performance throughout the integrated bench study. Although manual cycle time studies conducted in a preliminary phase of the research proved inconclusive, the opportunity existed to conduct a similar study within the confines of a more controlled and comprehensive experiment. To this end it has been shown that, even under optimal experimental conditions, shovel dig cycle times appear to be unrelated to actual measured levels of digging difficulty. It was also concluded that individual shovel operators maintained characteristic dig cycle times, irrespective of digging conditions. While each operator had an associated range of dig cycle times for easy to hard digging conditions, the differences in average dig cycle times between operators were greater than the individual ranges.

10.0 RECOMMENDATIONS FOR FUTURE WORK.

The central hypothesis of the thesis has been validated by establishing relationships between records of monitored shovel performance and levels of digging difficulty. The shovel is therefore seen as amenable to use as a diagnostic tool in indicating the efficiency of ground preparation practices. Future work is suggested that would involve a wider application of the current shovel monitoring technology in the following specific areas:

1. As a practical tool for use in the surface mining industry in a continual blast optimization program.
2. As an investigative tool to advance the state of science surrounding the fragmentation of rock by blasting.

In addition to the above "immediate" applications of the shovel monitoring technology, it is suggested that research be continued in terms of assessing the applications toward operator training, equipment maintenance and automation.

It is also recommended, as discussed within the thesis, that research be undertaken to assess the influence of fragment shape and interlocking upon muckpile diggability.

Blast Optimization in Surface Mining.

As mentioned in the thesis, a parallel investigation concerning the performance monitoring of rotary blasthole drills was conducted by Peck, 1989. The integration of both an instrumented drill and shovel on a working mine bench would enable gains to be realised over the entire production cycle.

Drilling performance data collected through the instrumentation and monitoring of rotary blasthole drills has proven capable of providing detailed information on bench geology. Cross sections of bench geology, based on interpreted drill monitored data may then be used to define optimal placement of explosive columns in mine benches. Performance monitored data collected through shovel instrumentation can then be applied towards rating the success of blast designs by assessing the relative diggability of the ensuing muckpiles.

Upon determining ratings of diggability indicative of difficult digging, for example, then an examination of the bench geology and the explosives loading instructions would follow. Subsequent blasts on the same bench level would then employ modified designs (e.g. explosive type, charge configuration, delay interval etc.) to yield better digging conditions. If successful, then shovel performance within the redesigned blasting strategy could be correspondingly improved.

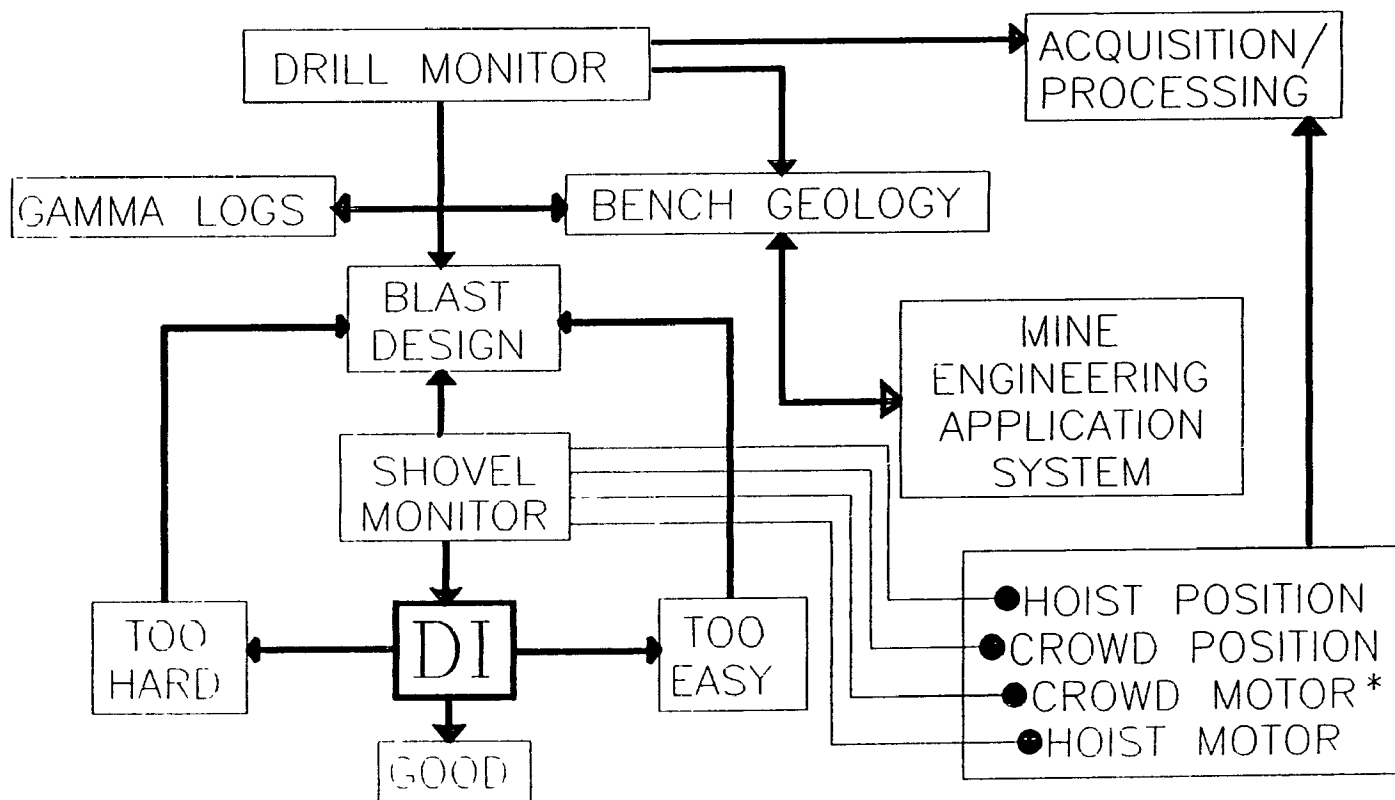
The concept of digging effort is by nature associated with levels of digging "difficulty" rather than digging "ease". It is possible however, that after broad use of shovel performance monitoring, the situation will arise in which consistently easy digging conditions have been indicated. In such instances the mine would have to evaluate whether the shovel is being worked below its operating design threshold. Further, if improved shovel productivity is not associated with very low levels of digging difficulty, it may be concluded that the cost of producing such easy digging conditions, is unwarranted. In such instances a cut back in powder factor may be indicated. Similarly, the same approach may be applied towards evaluating the benefits of high and low strength explosives. For example, in certain environments, the added costs of higher strength "high brisance" explosives with associated higher VOD's may be unwarranted. Such environments may benefit more from explosives that maximize the heave to strain energy ratio and produce better swell of the

rockmass. According to Hagan, 1974: "For the same degree of fragmentation, low and high swell factors produce poor and good digging conditions respectively", where swell is regarded as a measure of muckpile looseness.

The technique of blast optimization proposed therefore is an iterative one. Improving digging conditions through modifications to blast design may take several trials. However, both the direct and indirect benefits (production statistics, loading instructions, maintenance records etc.) that would accrue from equipment performance monitoring, outweigh the time and effort necessary to implement such a system. Figure 10.1 is a schematic of how the above integrated approach to blast optimization is perceived.

Work has already commenced on the design of a prototype shovel monitor that would satisfy the requirements of the above monitoring strategy. The results of the current research have indicated instrumentation of the following shovel operating parameters to be of prime concern: hoist armature current and voltage, crowd arm extension and hoist rope position. Design of the system will accommodate instrumentation of the crowd motor, should some future application require data on its behaviour. Although the current research has not succeeded in establishing any relations between the response of this motor and measures of diggability, it is possible that some relationship between the two exists. The issue should be further explored in the future with more rapid sampling intervals (i.e. 50 ms) and more elaborate inspection of the crowd motor signals to search for motor transients which may reflect changes in muckpile diggability. It is intended that appropriate algorithms and software be developed, to handle the efficient processing of the instrumented parameters, and generate end of shift diggability summaries.

Figure 10.1 - Integrated shovel and drill monitoring system.



The new monitoring system will be installed on a single mine shovel and used in conjunction with an instrumented drill, to create an integrated production reporting and control system, as well as a means of blast optimization.

Application to Blasting Science.

The preceding has described what is viewed a practical industrial application of the current research. The monitored shovel data is, however, also seen as capable of providing broader insights into the actual physics of what has been termed "muckpile diggability". Future, more refined controlled research is needed to advance the state of knowledge surrounding which characteristics of a muckpile exert the dominant influences over diggability such as; intact rock strength, muck shear strength, fragmentation size, fragment shape and interlocking, and compaction.

There currently exist no quantified means by which muckpile looseness can be directly measured. This is viewed as significant to the explosives industry, which currently is attempting to use such methods as computer modelling and high speed photography to gain insight into the dynamics of muckpile formation, evaluating such factors as heave, swell, profile and looseness. The monitored shovel parameters have proven very responsive to variation in looseness. It is suggested therefore that future research be conducted in tandem with research by the explosives industry. Attention should be paid towards developing a portable shovel monitoring unit, that could be moved about to the several mines at which the explosives industry annually conducts trial blasts, in optimization programs. Such a combined effort would provide actual measures of muckpile "looseness" (as inferred through diggability) with which to advance and validate the accuracy of the computer models, currently used to predict looseness.

Additionally, it is recommended that in future monitoring trials attempts be made to account for variations in muckpile profile from one dig cycle to the next. It is proposed that an EDM (electronic distance measuring) device be fastened on the shovel boom with a view towards the muckpile surface.

At the onset of the dig cycle this instrumentation would immediately scan a line segment from bottom to top of the muckpile (Figure 10.2). This data would then represent the muckpile surface profile for each dig cycle. Knowledge of the surface profile would accurately define the true cut depth in the actual material, and would permit advancing the already established relations between diggability and depth of cut.

Future research is also recommended in the area of determining muckpile fragmentation size distributions. Peripheral work should be initiated that would lead to real time feedback on fragmentation size distribution for each monitored dig cycle. It is ultimately envisaged that a video camera, also positioned on the underside of the shovel boom, would upon initiation of a dig cycle, recorded a few frames of the muckpile size distribution. It is hoped that future advances in automated image analysis would allow for the efficient, timely processing of the video images.

Operator training.

Although not specifically addressed within the confines of the current research, the present monitoring technology could find application in the assessment and training of shovel operators.

Presently, shovel operator trainers have no way of assessing operating practices other than via a physical presence to observe shovel activity. Observations tend to be restricted due to time limitations and the number of operators to be surveyed. Importantly, a knowledge of the trainers presence may motivate a

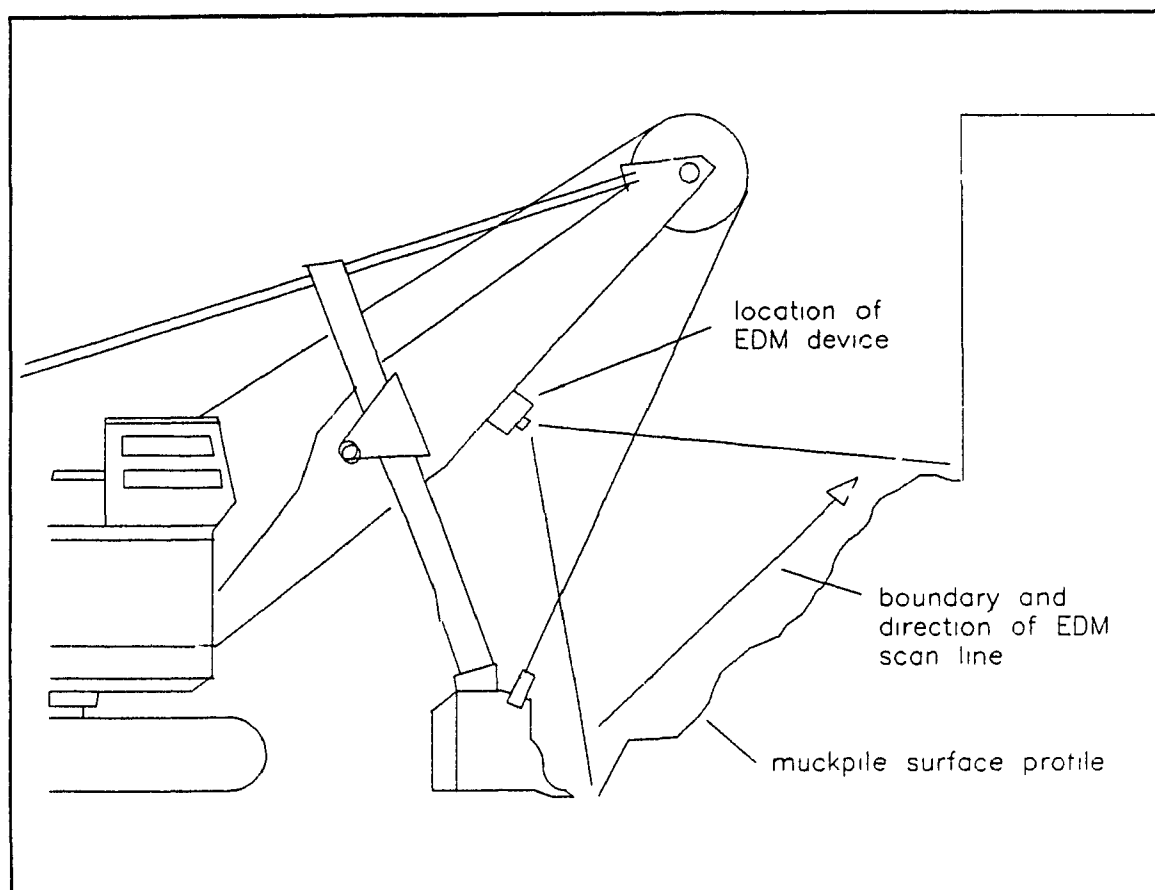


Figure 10.2 - EDM scanning of muckpile profile.

temporary change in the shovel operator's practice to satisfy the trainer that the operator is following the prescribed operating practice. The trainer has no continuous, quantified or permanent record of activity over which to track, compare and contrast operating practices.

The dipper trajectories recorded during the current research provide what is considered the most reliable and efficient means of determining and assessing operating practices. Continuous records of dipper trajectories provide insight into several facets of shovel operation: In addition to their obvious application in determining digging practices, trajectories can be employed to document the operating practices during the remaining load cycle elements; swinging and dumping. The swing and dump cycles represent, on average, 75% of the complete load cycle time. Optimization of these elements would therefore

contribute significantly towards the overall operating efficiency of the shovel. To illustrate one possible application; the value of the crowd arm extension signal upon dipper trip activation (dumping), is an accurate measure of the distance the shovel operator has positioned the haulage trucks from the shovel (spotting). If the trucks are positioned too far from the shovel the operator will have to use near maximum crowd travel to reach them, leaving little latitude for positioning of the dipper above the truck's box during dumping. Final loads on the trucks often require careful positioning to insure that large fragments do not fall from the truck during the drive to the spoil, potentially blocking haulage routes.

It is suggested that in order to gain a knowledge of dipper position in three dimensions, swing angles be monitored for future applications. Such data would broaden insights into operating practice by establishing records of swing distances (swing angles).

Equipment maintenance.

Regarding equipment maintenance, the present monitoring technology could be applied towards establishing historical trends on the performance of the various shovel work motion motors. Currently, only the crowd and hoist motors have been considered, it is recommended however that for the purpose of equipment maintenance, the shovel's swing motors be instrumented as well. It is not seen as necessary, for purposes of tracking motor wear, to maintain continuous records of motor performance. It would suffice to establish, on a continuous basis, the incidence of threshold overloads for each work motion motor. Such thresholds, tripping mechanisms, set point or limits could be adjusted on the basis of historical data to allow a realistic range of operating effort. Records of motor overloads could also be related to motor wear and the frequency of unscheduled maintenance. Since motor responses are a product of both digging conditions and operating practice it may be necessary to

undertake scheduled, controlled tests to track motor wear. This could be accomplished by having each operator conduct a series of imaginary dig cycles at the start of each shift. This data could serve as a calibration of the motors state of health from day to day.

The data collected through the above approach may ultimately lead to maintenance schedules established on the basis of the motor performance (overloads) records. Unscheduled maintenance episodes arise in part as result of the assumption that during equal intervals (i.e. scheduled maintenance intervals) the shovel was engaged in equally demanding digging conditions. The proposed approach to assist in shovel maintenance would provide mine maintenance with data which could permit scheduling maintenance periods on the basis of a documented need, as established by histories of digging effort, rather than just by chronology. It may also be possible to relate such records of motor performance to other maintenance concerns such as hoist rope wear.

Automation.

The complete or even partial automation of an electric mining shovel offers significant challenges. Recent work in Australia (Godfrey et al. 1989) has involved the partial automation of a dragline feeding a hopper of fixed position. The researchers acknowledged the difficulty associated with the automation of loading equipment: "The wide ranging variability of the bucket filling operation, digging position, dump position and surrounding topography does not readily lend itself to automation, a technology which favours more predictable repeatable processes". Realizing that partial, staged automation of individual cycle elements was the only way in which to proceed, it was elected (and successfully undertaken), to automate the swing to the hopper cycle. Mining shovels however, offer a yet greater challenge towards automation, as even the swing to truck cycle is non repeatable since haulage trucks, unlike a hopper, are not always at a fixed position.

The significance of the current data in regard to automation, lies in its contribution towards a better understanding of how the human operator undertakes operation of the machine. This understanding constitutes a developing knowledge base from which to design appropriate control strategies - the first step towards automation of any form of equipment. The current insights have helped define how operators modify equipment usage to accommodate the various digging circumstances encountered during normal operation. It is suggested that future research on automation dedicate itself initially towards applying the expanded knowledge of operating practices and equipment behaviour through both; physical models and computer based models/simulations of shovel operation in varied digging conditions.

Influence of fragment shape.

It is recognized that fragment shape will impart an as yet undetermined degree of influence over muckpile diggability. Different shape fragments will assume varied packing densities and hence offer varying degrees of interlocking resistance, affecting muckpile looseness and thus diggability.

Typically, each mine environment will, due to local structural regimes offer only one characteristic fragment shape. It will only be possible to assess the influence of different fragment shapes in the field, through shovel monitoring in a range of mine environments i.e. layered vs massive rock units.

It is recommend that in addition to the possible empirical relations derived through field monitoring, that laboratory investigation be undertaken. A similarly instrumented scale model shovel could be employed to conduct diggability experiments in controlled, specifically engineered muckpile conditions. The diggability of different shaped fragments under varying degrees of compaction could thus be explored.

Work is also recommended towards establishing a shape factor to describe or account for the influence of fragment shape on photographic surveys of fragmentation size distribution and in assessments of diggability. For example, the image analysis software employed in the current investigation could be modified in the future to accommodate conversion of fragment areas to a more suitable "universal" shape other than a sphere. If for example, jointing and bedding in a rockmass described tabular blocks (rather than cubic), then the "Z" dimension assumed by the image analysis software could be defined as 1/2 the diameter of a circle, thus describing a discus rather than a sphere, which assumes equal X, Y and Z dimensions.

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APPENDIX - A

POLYCORDER PROGRAM TO MONITOR HOIST AND CROWD ARMATURE VOLTAGES AND CURRENTS.

PROGRAM: SHOVEL 1

1. 31 OPN TESTDAT Open data file named "TESTDAT".
2. 26 CON Load number 1 into register A
 initialization of pointer and counter.
3. 58 SLC Instructs system to store registers A,B
 and C as the column line and page
 pointers respectively.
4. 53 AON Turns analog section on, must be done
 before any analog readings can be taken.
5. 11 DLY 5 Executes a 5 millisecond delay for warmup
 before first analog reading taken.
6. 23 SCN 25,1,2,3,4 Instructs system to scan analog
 input channels 1,2,3 and 4 (hoist
 voltage and current, and crowd
 voltage and current) for a 25
 millisecond integration period.
7. 66 TIM Records time for system clock at which
 above readings were taken.
8. 14 STO 4 Store constant of register A in user
 register 4.
9. 74 XAB Exchange the contents of registers A and
 B.
10. 14 STO 5 Store constant in user register 5.
11. 26 CON 0 Pointer to register containing first
 reading.
12. 14 STO 10 Store constant in user register 10.
13. 26 CON 6 Counter for number of readings ie 4
 analog and 2 for time.
14. 14 STO 11 Store constant in user register 11.

15. 17 RCI 10	Recall contents of user register 10 and store value in register A.
16. 77 STF	Store first reading in first column of file TESTDAT.
17. 19 DCR 11	Decrement user register 11 ie. decrement the counter by 1.
18. 18 ICR 10	Increment user register 10 ie. increment the pointer by 1.
19. 61 ICP	Increment to the next column of the data file TESTDAT.
20. 17 RCI 10	Recall the next reading using the pointer in register 10.
21. 77 STF	Store next reading in next column of the data file TESTDAT.
22. 10 DJN 11,18	Decrement the counter in user register 11 and jump to line 18 if not 0.
23. 1 JMP 8	Jump to line 6 and scan the 4 analog inputs again.
24. 0 END 11,18	End of program.
25. 0 END 8	
26. 0 END	

APPENDIX B - PRELIMINARY INVESTIGATIONS:
MANUAL TIME STUDIES - SUMMER 1987.

B.1 INTRODUCTION.

This appendix details the nature of work done in advance of the more comprehensive shovel instrumentation studies, reported on in the remainder of this thesis. The results of this preliminary research were significant in that they motivated and defined the required approach undertaken during the more sophisticated/comprehensive investigation of the second field period - the integrated bench study.

B.2 METHODOLOGY - MANUAL TIME STUDIES.

The time studies were undertaken by two analysts (the author, in addition to by Mr. H. Casanova and/or Mr. J Davidson both of McGill University) stationed in a truck, typically situated some 30 to 60 meters behind or to the side of the operating shovel being studied. The analysts would alternate recording the following shovel cycles times:

1. Dig time, the time spent in the muckpile filling the dipper.
2. Swing time to truck.
3. Dump time, the time to unload the dipper in the truck.
4. Swing time back to the bank (muckpile).

In addition, delays associated with each of the above cycles such as face preparation, sifting the muckpile, waiting for or spotting a truck, propelling etc., were also recorded. While one analyst was recording the above times on his stopwatch, the other analyst would be involved in transferring recorded times to a form designed for this purpose. In this manner a continual record

of shovel cycle times could be established. Typically, an average of 65 complete cycles would be recorded per study, and as many as seven studies would be conducted per shovel per day.

B.3 ANALYSIS OF TIME STUDY DATA.

A preliminary analysis of the data revealed that of the four operating parameters monitored (dig time, swing times and dump time), the dig cycle times warranted the priority analysis. The remaining parameters, while of real interest, are felt to be of no significance to the development of a diggability index based on material and site characteristics. It was the opinion that the derivation of a diggability index should first be a reflection of the physical responses of the shovel (dipper) to the characteristics of the material being excavated, which in turn are considered a function of the original bench geology and blast design.

In light of the above, it was decided to investigate in depth the relationships that may exist between mean dig cycle times of the shovel in a given mine environment and the geologic and blast design criteria that characterize that environment.

The conclusions drawn on the data of a preliminary field period were considered with caution. The methodology employed during the summer 1987 field period was regarded as exploratory, and based purely upon shovel time studies. However, the equipment and resources employed during the integrated study (summer 1988) have provided the depth of data considered necessary to evaluate with confidence, the role and effect on diggability of mine environment, i.e. geology, blast design and operating practice.

B.4 SITE GEOLOGY AND BLAST DESIGNS.

In order to fully interpret and explain the shovel performance statistics compiled from the summer 1987 time

studies, the geology and blast designs for each of the sites and digging conditions studied have been analyzed.

Geology.

Individual sites selected for study have been identified first by pit name, i.e. Taylor Pit or Eagle III, secondly by the blast being excavated and finally by the location of the shovel in the blast pattern during the monitoring period of a particular study.

The geology for each site/condition studied is presented in the form of a stratigraphic column. The column is representative of the geology encountered by the shovel during specific studies in each pattern. The stratigraphy has been derived from gamma logs of production and exploration holes taken within +/- one or two units of burden along strike of the shovels position. In this fashion it was considered possible to develop a reasonable estimate of the muckpile composition during individual studies.

Blast Designs.

For each of the studies the associated blast pattern has been drafted in plan view. These illustrations serve to identify the approximate location of the shovel in the blast pattern during individual studies, in addition to providing the fundamental blast design parameters for each of the sites examined; hole size (i.e. 60-R drill= 12 1/4", 45-R= 10 5/8"), burden and spacing, type of downlines and bench number.

The loading instructions are, for ease of interpretation, presented in the form of a single explosive column representing the charge distributions issued to correspond with the stratigraphy encountered by the shovel during any particular study. The "explosives column" is intended to provide an indication of the overall scheme adopted by the blasting engineer, in designing a blast to meet the strength characteristics of rock units in any given pattern / environment.

The charge distributions have been determined by a survey of the loading instructions issued for at least two or three units of burden, behind and in advance of the shovel's position during the study period. It is felt that in this manner, while not accounting for all deviations in loading instructions from the idealized explosive column, a clear and concise picture of the blast design for each study location is evolved.

B.4.1 Site Characteristics.

The following are descriptions of the geology and blast designs for each of the mine sites in which the cycle times of shovels were monitored in 1987. All of the patterns were drilled with 12 1/4" diameter blastholes in 15 metre high benches, and all blasts, with the exception of one (which used prilled AN/FO), employed emulsion based explosives - Heavy AN/FO.

Studies A1-A12:

Studies A1 to A12 were conducted on Eagle Stage III, bench 2210, while #14 shovel was digging compacted rehandle. Since neither the effects of original geology or blast design can be resolved for this digging condition, no stratigraphic or explosive columns are provided for this situation. This series of studies has served as a "base case", representing the easiest digging conditions, against which other studies can be compared.

Studies B1-B2:

Studies B1 and B2 were conducted on Eagle Stage IV, bench 2300, while #15 shovel was excavating the back of blast EF #1 fired in a series of very competent flat lying massive siltstones and sandstones. Detailed information on the design of the blast was unavailable.

Studies C1-C6, G1-G3 and I1:

Each of these studies was conducted in Taylor Pit, bench 1977, while #12 shovel was excavating blast TH #1. All studies, with the exception of G2 (digging downdip), were conducted while

digging along strike, as illustrated in Figure 1. which represents the upper portion of the blast. This part of the blast pattern was through straight waste above number 7 coal seam. The waste units encountered were predominately silt and sandstones (Figure 2), which dipped to the northeast at an average of 18 degrees.

Design specifications for the blast were; 12 1/4" blast holes drilled with burdens and spacings (predominantly) equal to 10 m. Being through straight waste, the loading instructions for this portion of the blast were fairly uniform; 65 foot holes backfilled to 57 feet followed by a single 30 foot explosive column of SAN 2495 and a 27 foot collar. The downlines employed were Scuf-Flex, with standard 35 and 50 ms delays.

Studies D1-D5:

Studies D1 to D5 were conducted in Taylor Pit, bench 1992, while #11 shovel excavated the final portions of blast TE #2, fired December 3, 1986. All studies were conducted while digging the footwall of 11 seam in a downdip direction perpendicular to strike. The materials being excavated during the studies were a 10 m thick, hard massive sandstone overlain by a 5 m thick unit of siltstone (Figure 3), which dipped to the northeast at 17 degrees.

This is the only study conducted in a blast that employed aluminized ANFO explosive blends. The blast's design parameters were; 12 1/4" blast holes drilled in a pattern with burdens and spacings equal to 10.5 m. The loading instructions depicted for this blast in Figure 4 were; 64 foot holes backfilled to 59 feet followed by 10% AL ANFO to 49.5 feet, 5% AL ANFO to 30 feet, ANFO to a 23.5 foot collar. Downlines were Scuf-Flex with standard delays of 35 and 50 ms, employed in a "V1" firing sequence.

Studies E1-E6 and F1-F5:

Studies E1 to E6 and F1 to F5 were conducted on Eagle Stage III, bench 2210, while #15 shovel excavated blast EA #1. While all of these studies were in the same blast, the difference in geology/stratigraphy between "E" and "F" studies is sufficient to qualify the two as distinct situations, with loading instructions unique to each environment (Figures 6 and 7).

The fundamental design parameters were 12 1/4" holes, burdens and spacings of 10 m and Scuf-Flex downlines.

Studies E1 to E6 (Figure 5): #15 shovel was digging west and downdip through the shallow dipping hanging wall waste of number 15 coal seam. The waste units encountered by the shovel during the study were a massive sandstone unit at the toe of the muckpile, followed by the 1 meter thick 17 seam and a series of alternating silt and mudstone units, all dipping at 8 degrees to the west (Figure 6).

The explosive column at this location in the pattern had been tailored to meet the needs of the hard toe. A 65 foot blast hole backfilled to 58 feet, followed by; SAN 2510 to 48 feet, SAN 2505 to 38 feet and SAN 2495 to a standard 23 foot collar. Despite heavier loading at the toe, difficult digging conditions in the toe region were observed during the monitoring of "E" studies, ie. the shovel was observed to be over 4 feet above intended grade. It was assumed that the operator could not excavate to design grade.

Studies F1-F5 were conducted approximately 100 meters to the east of the "E" studies. Here at the toe of 15 seam, #15 shovel handled hangingwall dozed material during studies F1-F3, and dug along strike at the base of the 2210 highwall during studies F4 and F5. The waste being excavated is similar to that described above for the "E" studies, with the exception that stratigraphy then placed the competent massive sandstone unit

(previously in the toe of "E" study) in the centre of the bench (Figure 7).

The loading instructions had been modified as follows: Increased standoff, 64 foot hole backfilled to 56 feet, SAN 2495 to 49 feet, SAN 2505 to 40 feet, SAN 2495 to 30 feet and SAN 2510 to a standard 23 foot collar.

Studies H1-H2:

Studies H1 and H2 were conducted on Eagle Stage III on the west side of the syncline axis, bench 2210, while #15 shovel excavated blast EF #1 by digging west and updip to expose the toe of 15 seam. Referring to Figure 8, study H1 was conducted while the shovel dug a half-height bench, working to expose 15 seam, study H2 was conducted while digging a full height bench along strike. The bench stratigraphy being excavated during the study was essentially a single thick, very competent massive silty sandstone (Figure 9).

Of all the sites investigated during the preliminary research, this study was in the mine location characterized by the most competent rock units - the synclinal axis of Eagle Mountain. This is also one of the two blasts studied in which NONEL downlines and DTH delays were employed to meet the demands of the massive units typical of bench areas bordering the syncline axis. Again, 12 1/4" blast holes were drilled in a pattern with spacings and burdens equal to 10 m.

The loading instructions for the study location were as per Figure 9; 75 foot holes backfilled to 61 feet, a very heavy toe charge of SAN 2510 to 43 feet, SAN 2505 to 26 feet, deck to 19 feet and SAN 2505 to a short collar of only 14 feet, with standard tie-in and NONEL delays of 50 and 75 ms.

Studies J1-J2 and K1-K6:

Studies J1-J2 and K1-K6 were conducted in Taylor Pit, bench 1977, in blast TD #5, after #11 shovel completed a ramp-down from 1992 and worked to expose the toe of 9 seam (Figure 10). Excavation for these studies was both along strike and updip through the waste above number 9 coal seam. The waste units were an alternating sequence of sandstones, siltstones and mudstones dipping to the northeast at approximately 11 degrees.

Again, 12 1/4" blast holes were drilled in a pattern with spacings and burdens of 10 m. The explosive column designed for the stratigraphy encountered during the studies is presented in Figure 11; 65 foot holes were backfilled to 48 feet (depth to 9 seam was approx. 53'), SAN 2505 to 38 feet, SAN 2510 to 30 and SAN 2495 was loaded to a standard 23 foot collar. The downlines employed in the blast were Scuf-Flex with standard delays of 35 and 50 ms, in a "V1" firing sequence.

Figure 1 - Blast design for studies C1-C6, G1-G3 and I1.

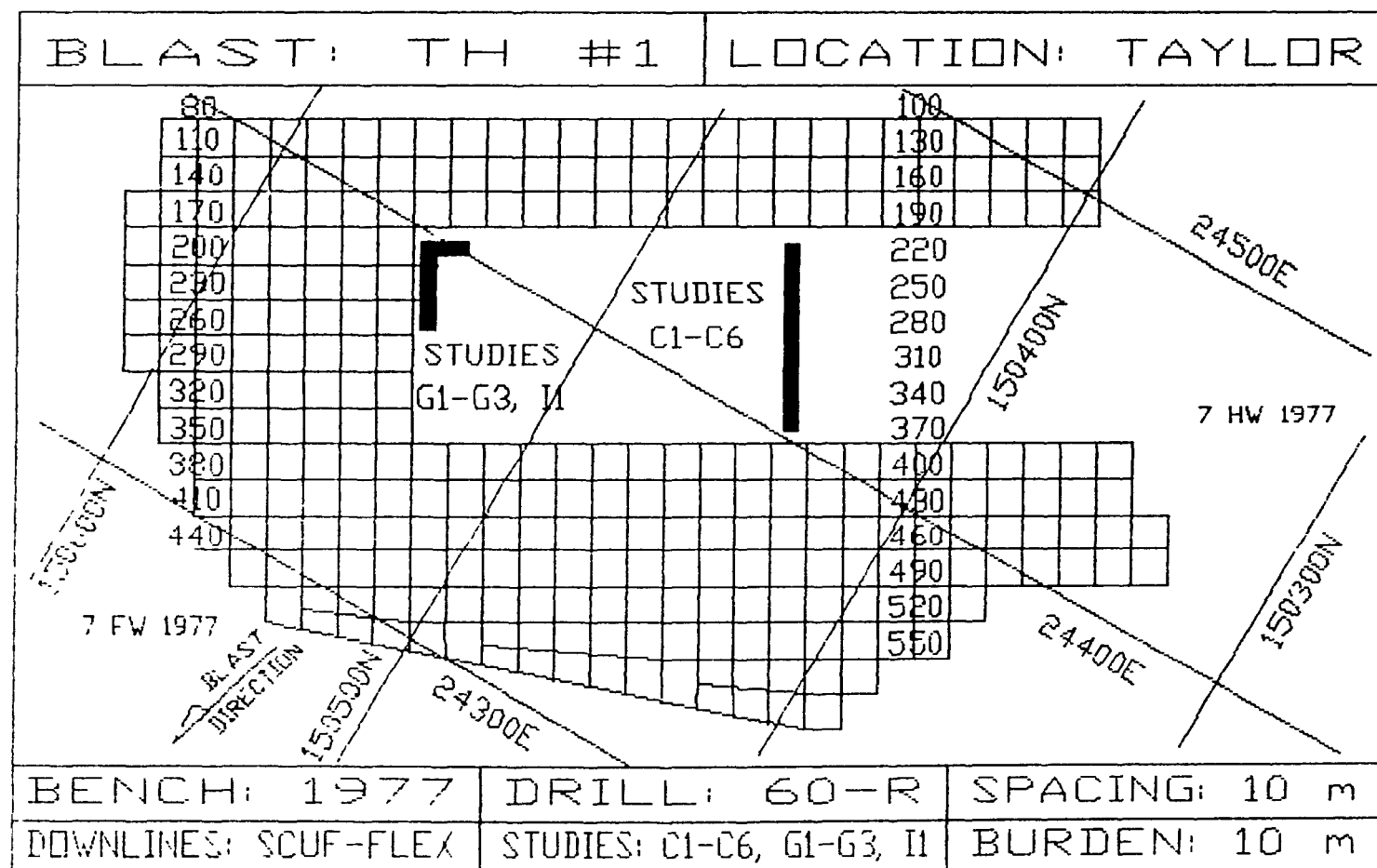


Figure 2 - Stratigraphy and loading instructions for studies C1-C6, G1-G3 and I1.

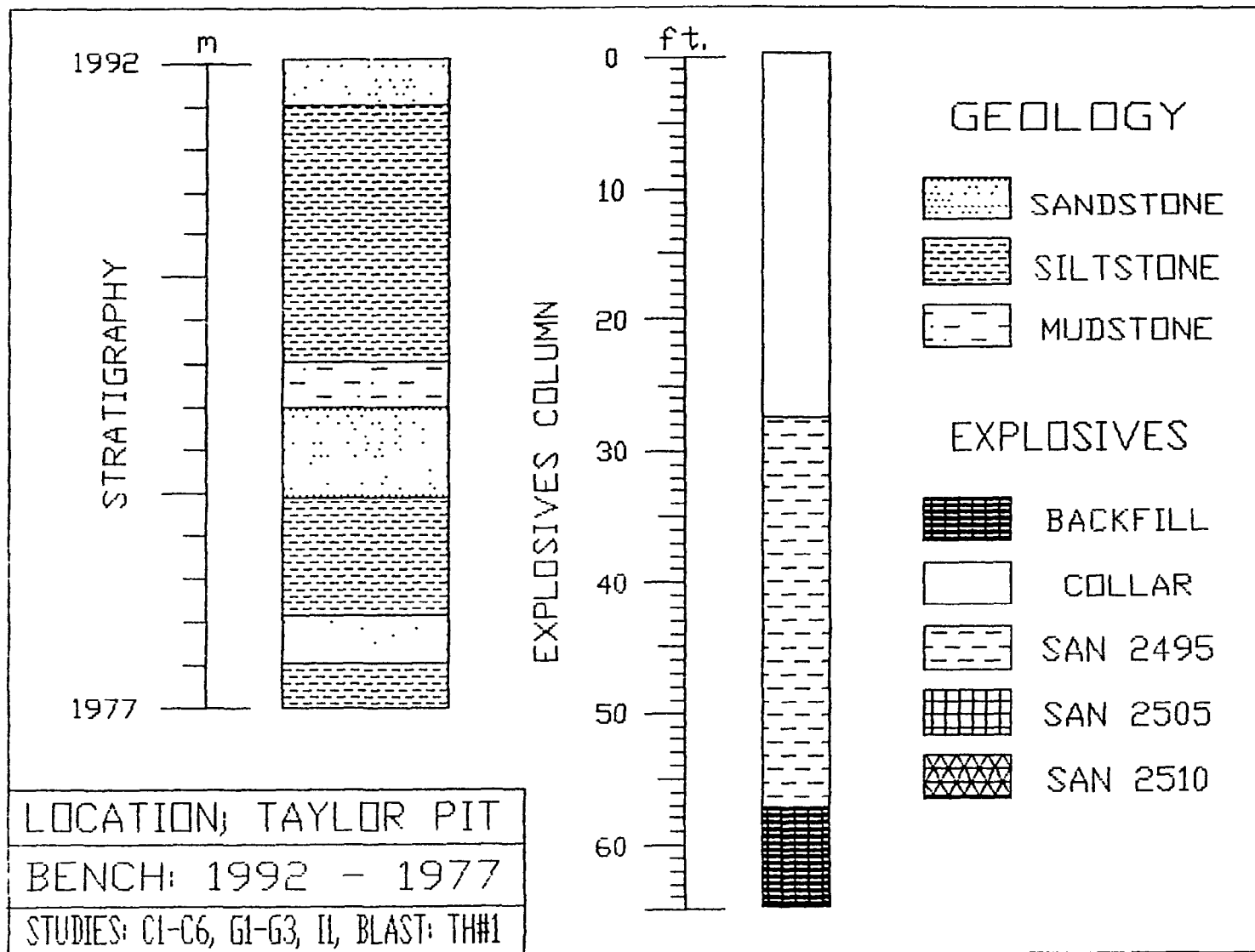


Figure 3 - Blast Design for Studies D1-D5.

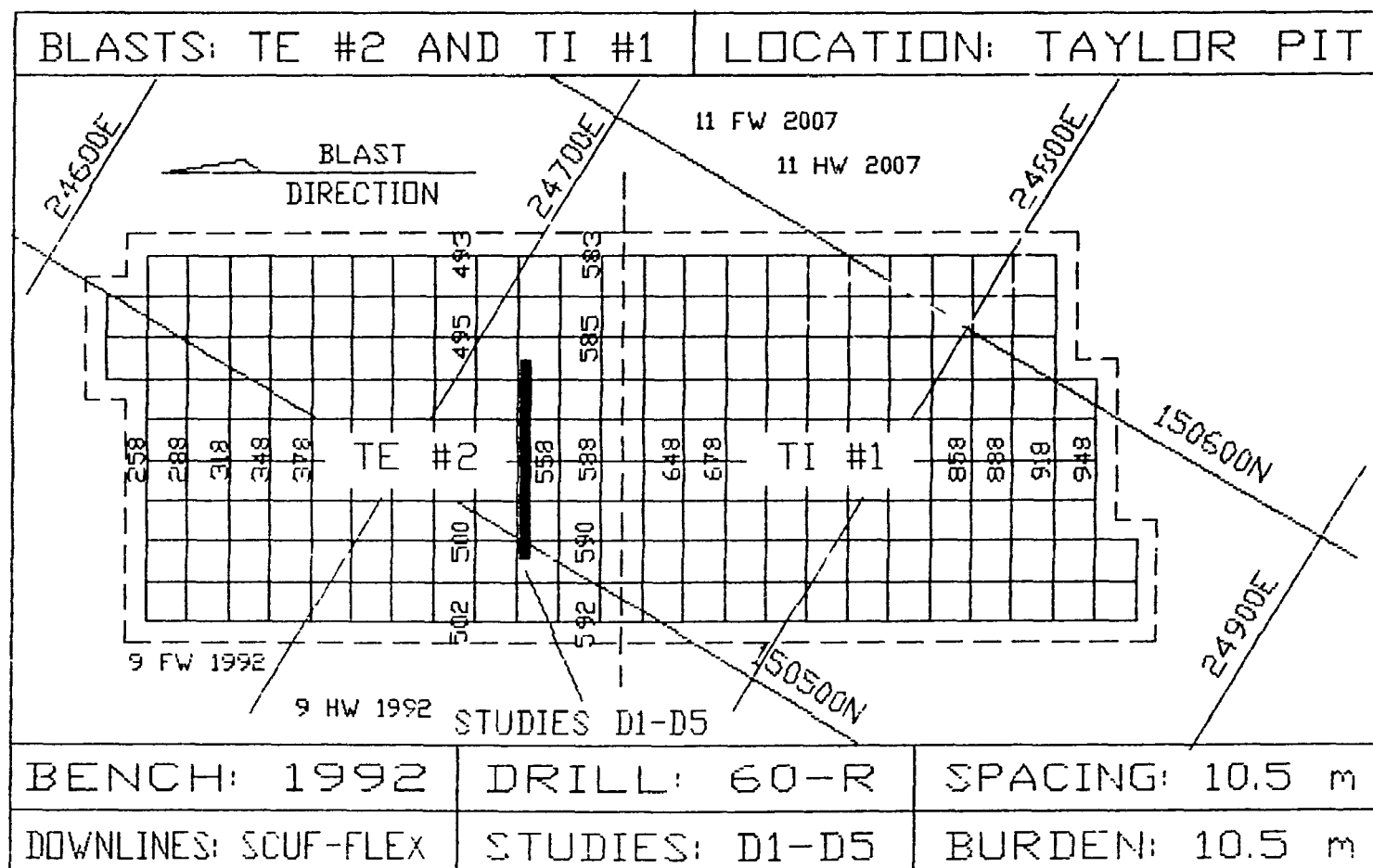


Figure 4 - Stratigraphy and loading instructions for studies D1-D5.

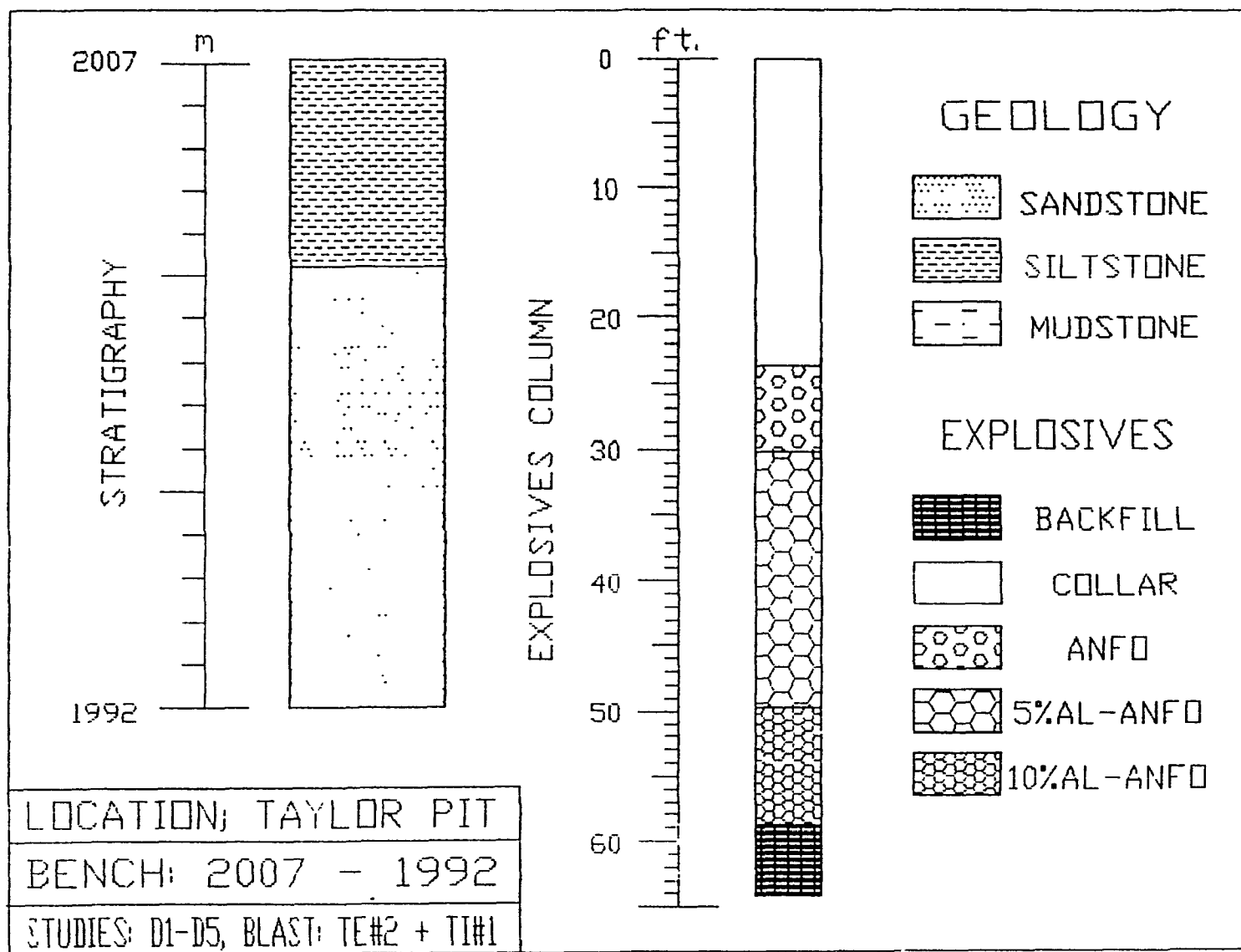
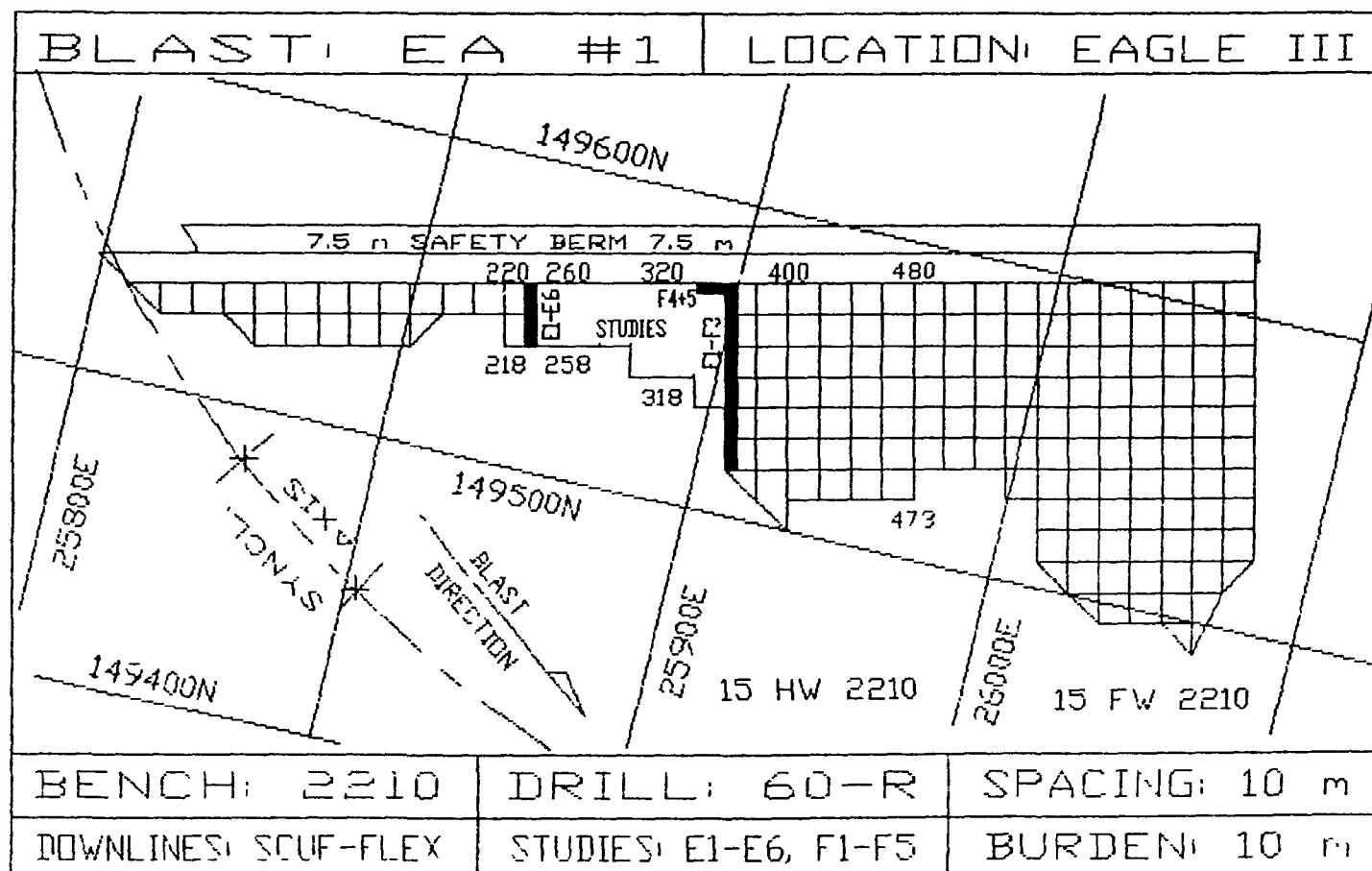


Figure 5 - Blast design for studies E1-E6 and F1-F5.



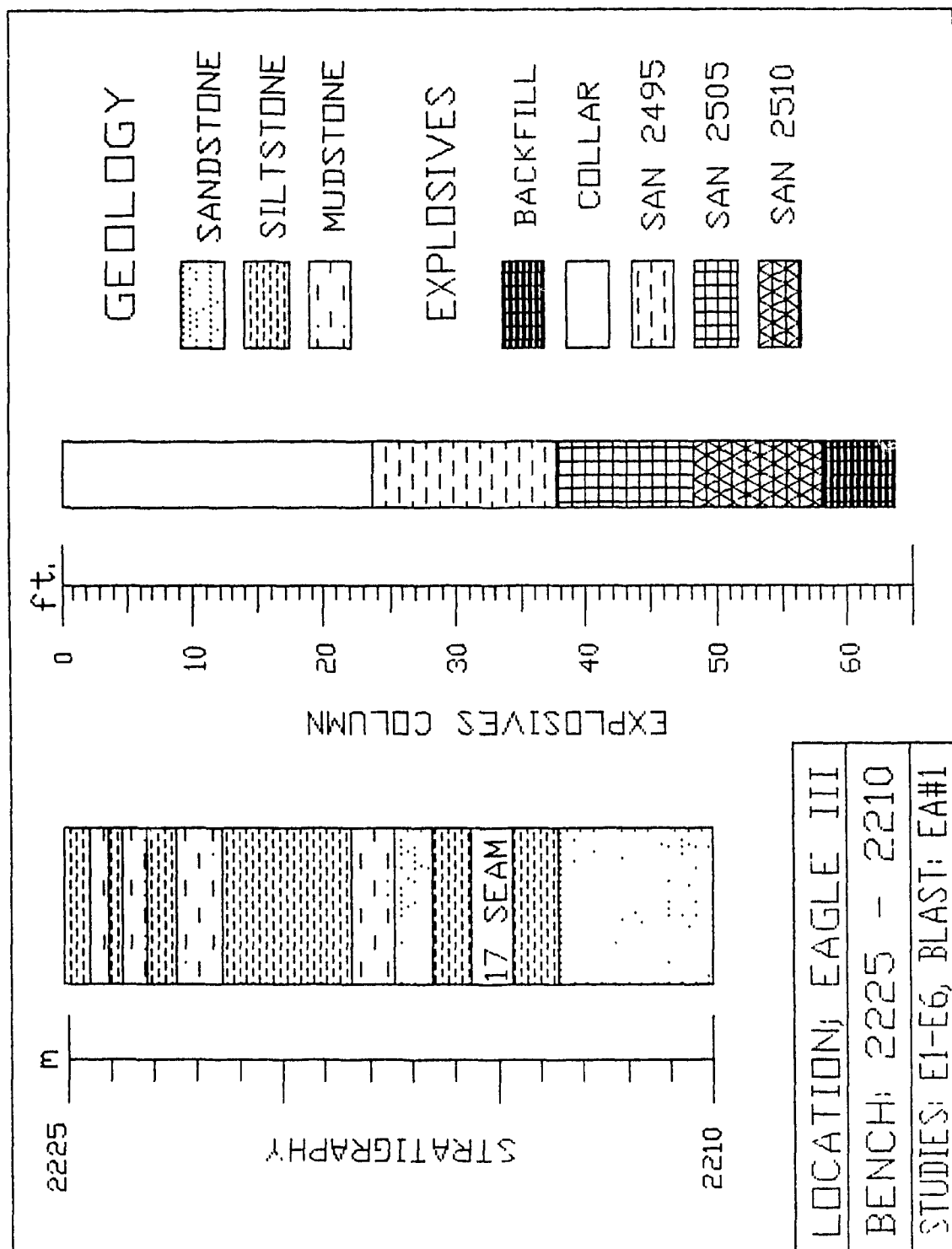


Figure 6 - Stratigraphy and loading instructions for studies E1-E6.

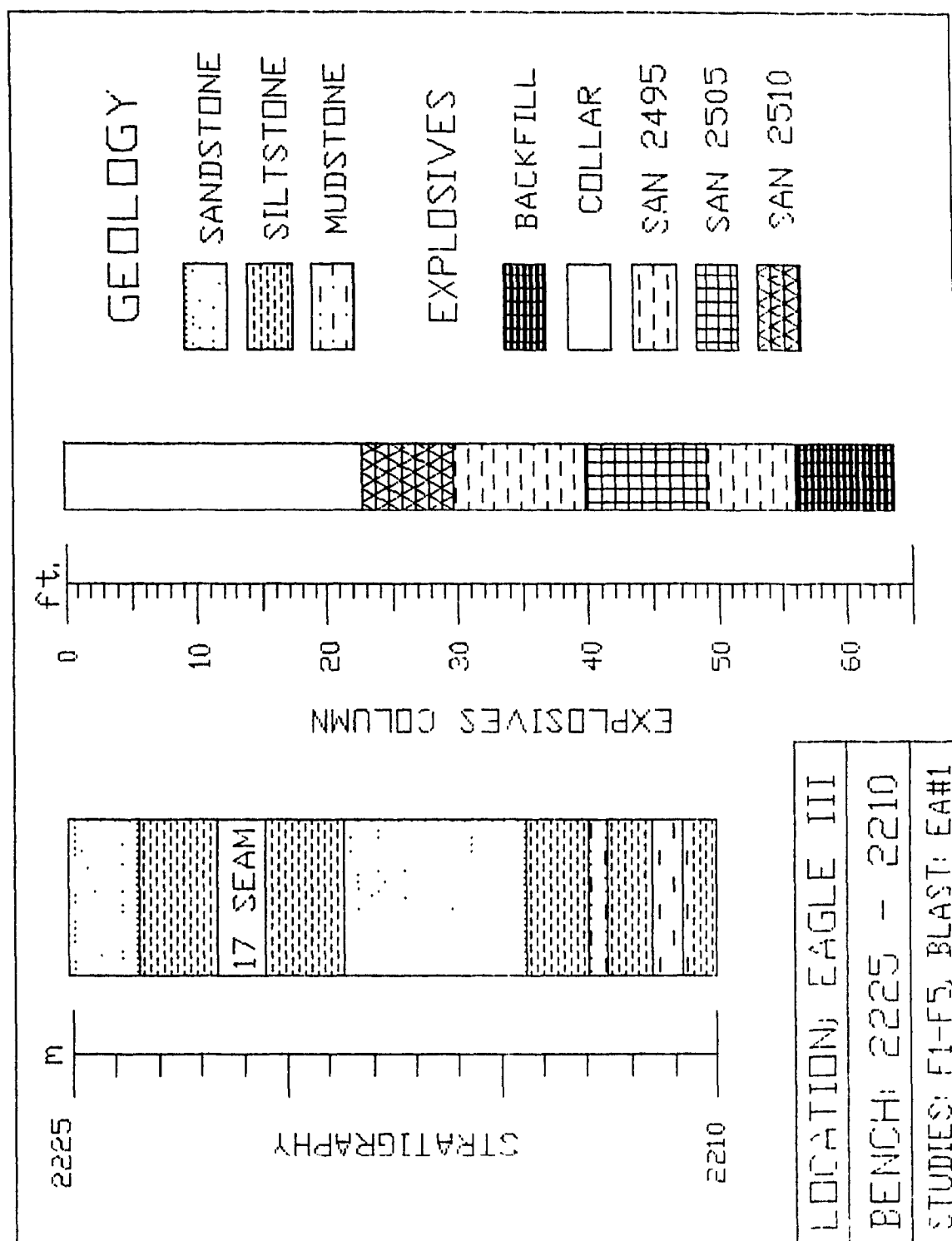


Figure 7 - Stratigraphy and loading instructions for studies F1-F5.

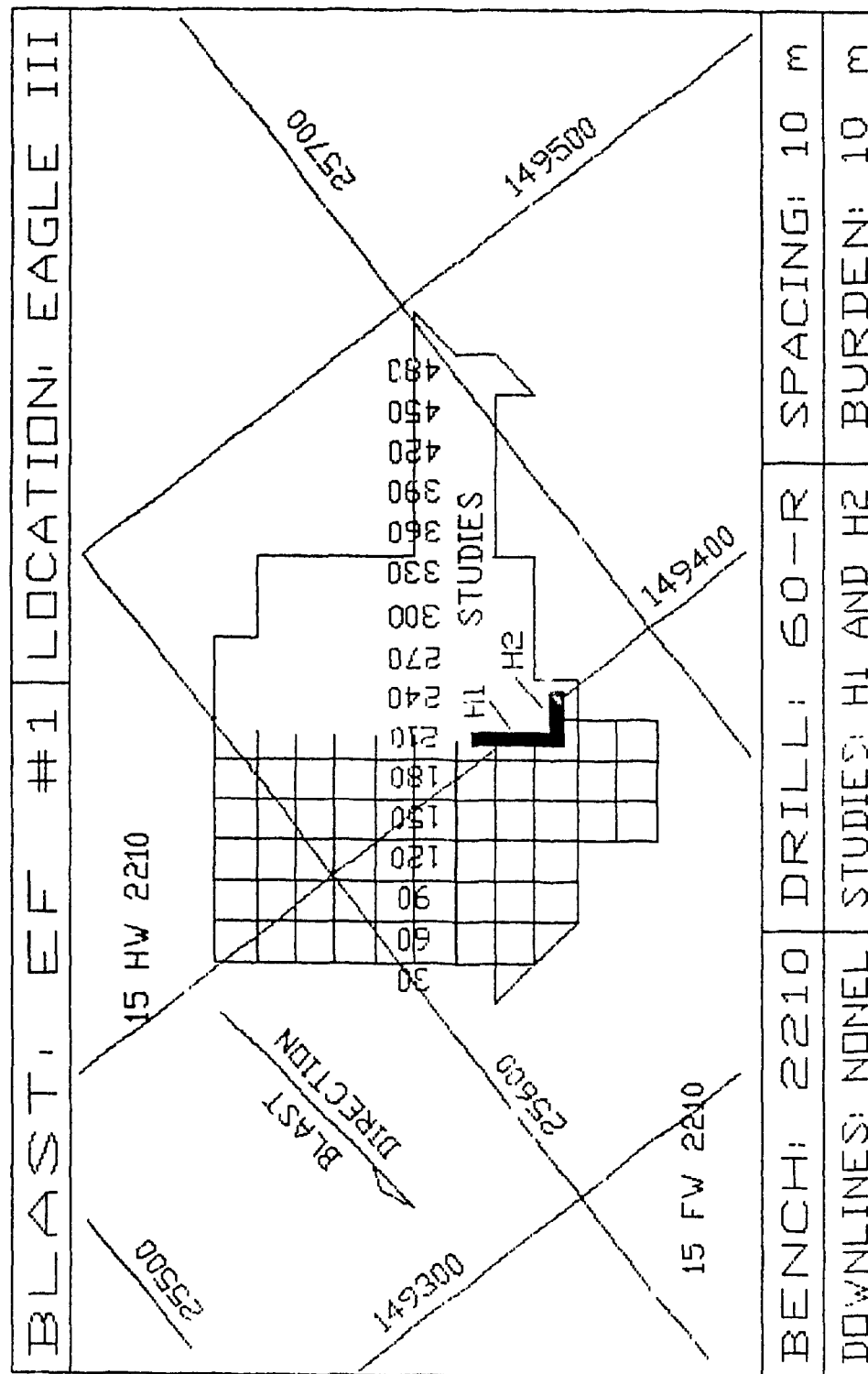


Figure 8 - Blast design for studies H1 and H2.

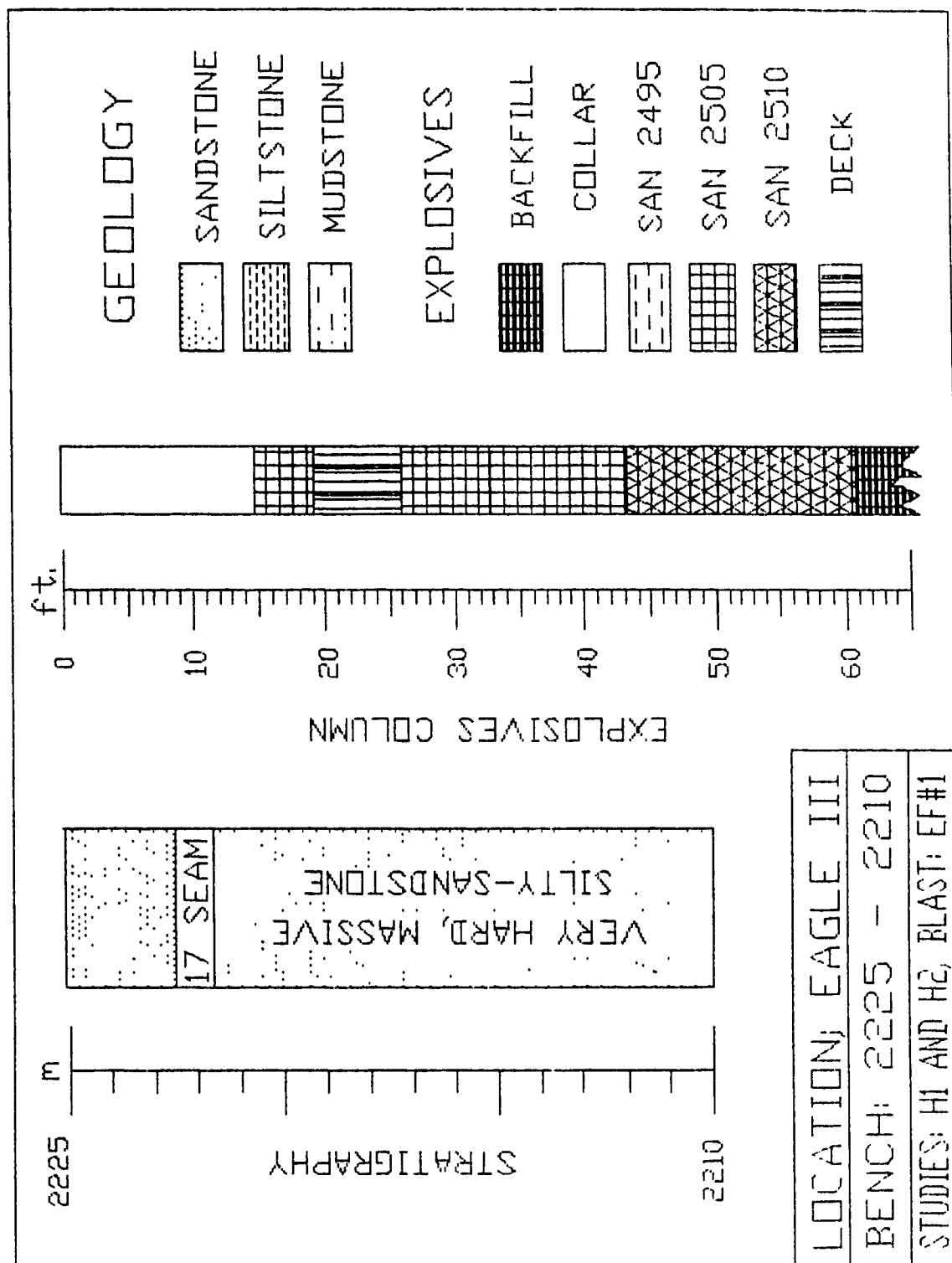


Figure 9 - Stratigraphy and loading instructions for studies H1-H2.

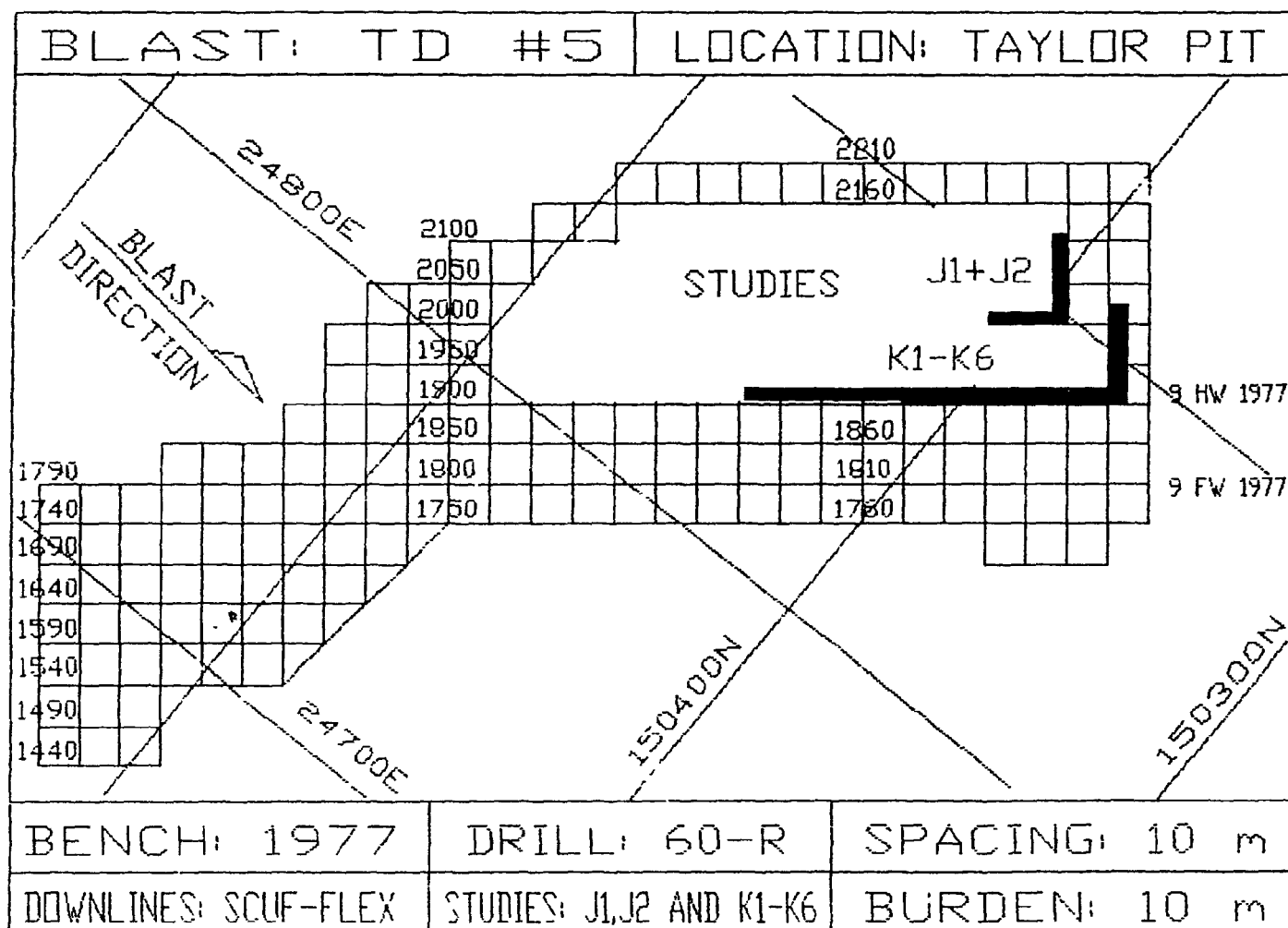


Figure 10 - Blast design for studies J1-J2 and K1-K6.

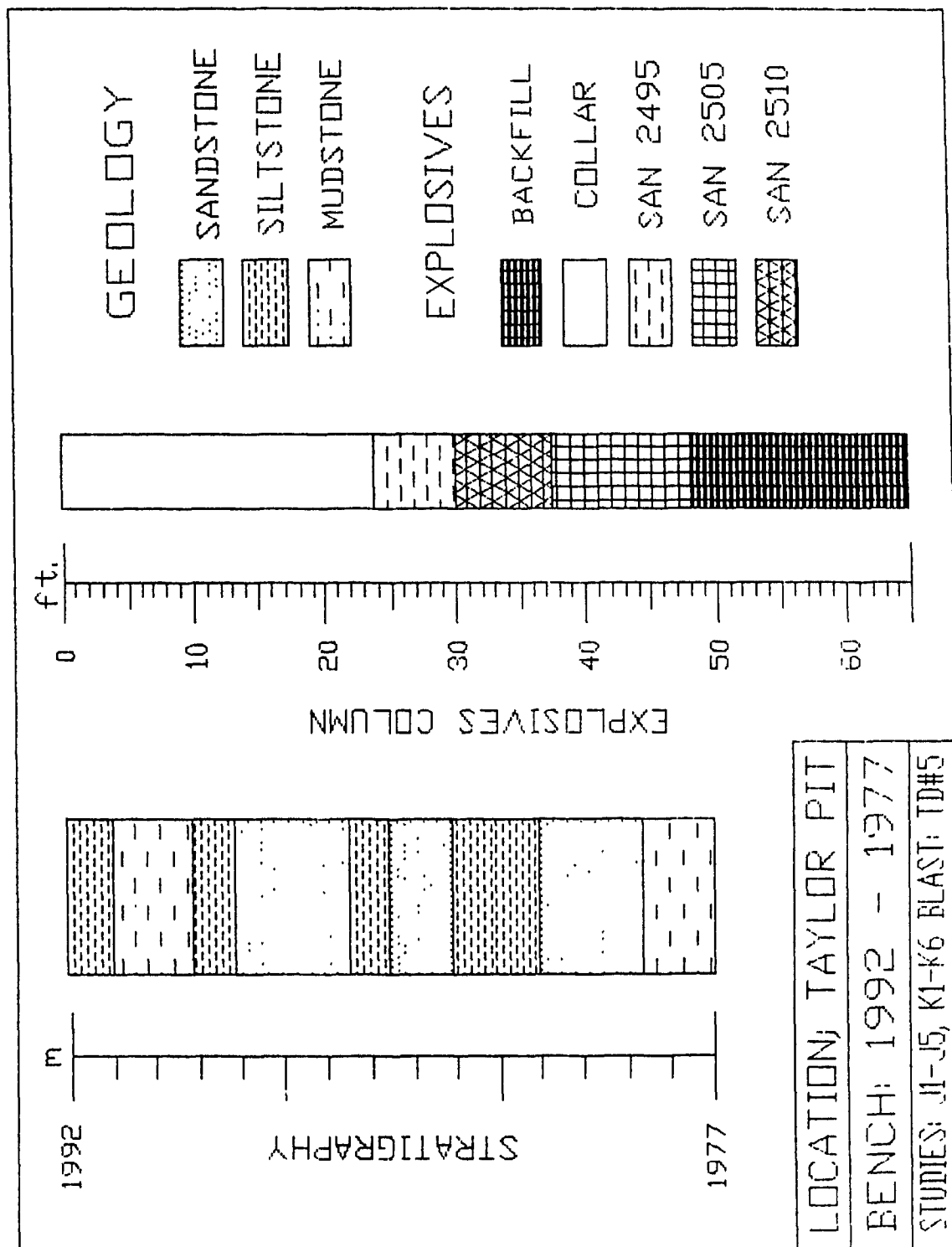


Figure 11 - Stratigraphy and loading instructions for studies J1-J2 and K1-K6.

B.5 DERIVATION OF SITE INDICES.

For each of the sites or study locations described in the previous section, an index, based on the location's geology and blast design parameters has been determined. The site index was aimed to permit simplified comparisons of sites and provide the basis for attributing variations in dig cycle time from site to site.

The index is defined as the ratio of the total bulk explosive strength of the explosive column employed at the location of any given study, to the total strength of the geologic column being excavated:

$$\frac{\text{Total Energy Units/litre}}{\text{Total MPa-m}}$$

All calculations were based on the average rock compressive strengths¹ as determined by Golder & Assoc. (Golder, 78) for Fording Coal Ltd., these are:

- Sandstones 136 MPa
- Siltstones 113 MPa
- Mudstones 55 MPa
- Coal 25 MPa

The total strength of explosive columns have been determined through the following data, supplied by Fording Coal Ltd., on the properties of the various explosive blends used at the mine.

¹ While tensile strength is a better indicator of rock blastability, data on tensile strength was limited and the test to determine it (Brazilian test) is less reliable than for compressive strength testing.

	<u>RBS (E.U.)/l.</u>	<u>Density (gm/cc)</u>
SuperAN 2495	109	0.97
SuperAN 2505	120	1.06
SuperAN 2510	125	1.15
ANFO	99	0.84
5% Al ANFO	109	0.87
10% Al ANFO	128	0.89

The following is a sample calculation of a site index using the data obtained for studies E1 to E6, blast EA #1, Eagle Stage III, Figures 5 and 6.

The total strength of the stratigraphic column is determined by calculating the combined length of each rock type in the column, and multiplying each of these by their associated compressive strengths in MPa. For the above example study the break-down is as follows:

	Total length of similar units (m)	Average Strength MPa.	Total unit strength MPa-m
Sandstone	4.5	136	612
Siltstone	7.0	113	791
Mudstone	3.5	55	192.5
Coal	1.0	25	<u>25</u>

Total strength of rock column: 1620.5

The total strength of the explosive column is determined in like fashion. Since all studies were conducted in patterns drilled with 12 1/4" blast holes, distributions of explosives in the column have been converted from feet to a standard volume of 28.18 litres of explosive per foot of explosive column.

	Total volume of explosive (l)	RBS energy units/l	Total energy RBS x l
SuperAN 2495	422.7	109	46074.3
SuperAN 2505	281.8	120	33816.0
SuperAN 2510	281.8	125	<u>35225.0</u>

Total strength of explosive column: 115115.3

The site index for this study would therefore be calculated by dividing the total strength of the explosive column by the combined strengths of the rock units:

$$\frac{115115.3}{1620.5} = 71$$

The units associated with the index value are E.U./MPa-m.

Proceeding as above for the remaining studies, the indices documented in Table 1 were established:

Table 1, Site indices for study environments

	Total rock strength	Total explosive strength	INDEX
- Eagle Stage III, studies A1-A12	n/a	n/a	REHANDLE
- Eagle Stage IV, studies B1-B2	unavailable		
- Taylor Pit, studies C1-C6, G1-G3, I1	1729	92148.60	53.3
- Taylor Pit, studies D1-D5	1930.75	112029.59	58.0
- Eagle Stage III, studies E1-E6	1620.50	115115.30	71.0
- Eagle Stage III, studies F1-F5	1681.25	109070.69	64.9
- Eagle Stage III, studies H1-H2	1796.75	130896.10	72.9
- Taylor Pit, studies J1-J2 and K1-K6	1621.25	85272.68	52.6
AVERAGE VALUES:	1729.92	107422.16	62.1

B.6 ANALYSIS OF SHOVEL PERFORMANCE TIME STUDIES.

A principal objective of the preliminary field study was to attempt to identify variations in shovel digging performance in a variety of mine environments and digging conditions, and to analyze the significance of these differences. To that end a hierarchical data structure was delineated, Figure 12.

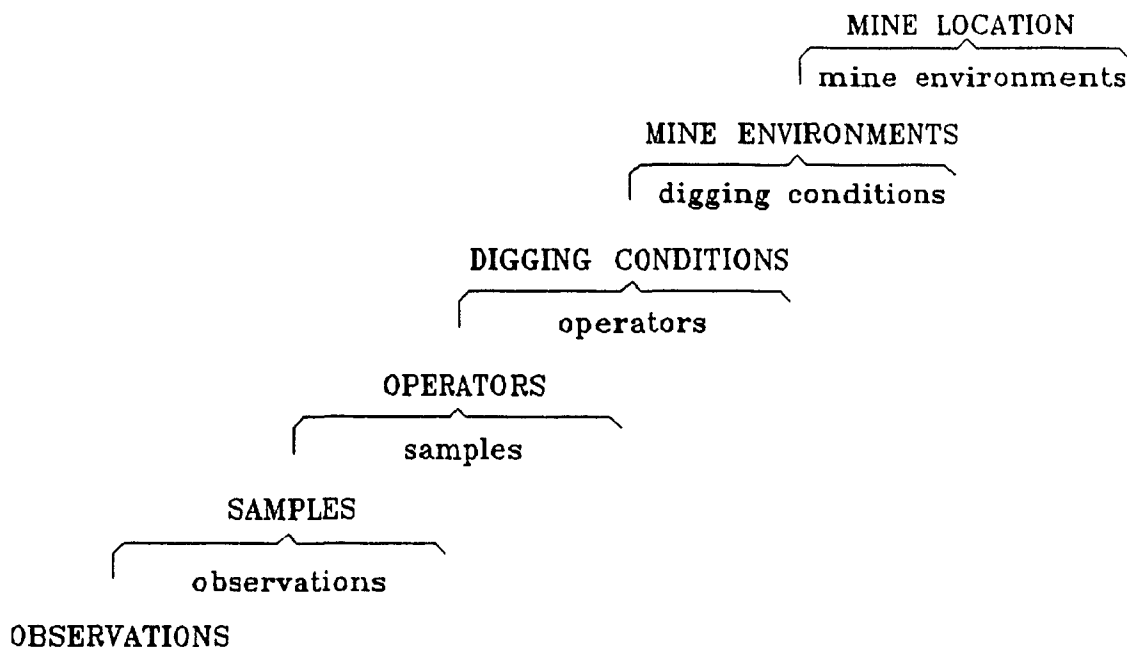


Figure 12 - Hierarchical structure of data.

This resembles a pyramid, starting with individual observations (records) at the base, aggregated into single sample sets of an individual operator (samples). These samples sets taken together approximate the performance of an individual operator on a given shovel in a given place on a given shift. If more than one operator was observed in a given location, the operator data sets taken together form a basis for describing the shovel performance in a given location (studies). Upon examination of the underlying geology and observable digging conditions, these "location" studies can sometimes be further combined into descriptions of particular environmental situations

or zones which appear to exist at the mine (environments). This decision will be discussed in greater detail below.

The final level of aggregation was the shovel type, i.e. 2800 or 2800XP (the 2800XP is equipped with a boom, 7 feet longer than on the 2800 model, also, the 2800XP has dual propel motors). These shovel types effectively divided the total database into two distinct data sets - one describing Taylor Pit in which 2800's are employed and the other the top of Eagle Mountain - 2800XP's.

Data Analysis

The approach taken to statistical analysis of the data involved the following series of steps.

1. Summarize the underlying characteristics of the data, including measures of central tendency location dispersion and skewness.
2. Establish the normality or non-normality of the data distributions.
3. Develop and test hypotheses regarding the equivalence/differences between data sets.

The means and standard deviations of the dig cycle times at the various study levels are summarized in Table 2. Note that the base case easy digging scenario, Study Ab (compared full bench rehandle) did indeed have the shortest average dig time and smallest variation around the average.

Table 2 - Summary of dig cycle time by environment.

ENVIRONMENT	STUDY SERIES	MEAN TIME FOR STUDY	OPERATOR	MEAN TIME/ OPERATOR
I	Aa	10.65 (2.84)	#2	10.94 (2.69)
			#12	9.05 (1.61)
	Aa+Ab	9.64 (2.24)		
	Ab	9.12 (1.80)	#3 #12	8.95 (1.76) 9.44 (1.61)
II	B	11.73 (2.99)	#6	11.76 (3.30)
			#1	11.69 (2.51)
III	E	10.49 (3.03)	#3	10.60 (3.36)
			#8	10.29 (2.35)
IVa	Fa	-----	#2	11.09 (2.56)
	Fa+Fb	11.13 (2.52)		
	Fb	-----	#12	11.19 (2.47)
IVb	H	-----	#3	10.63 (2.26)
V	C	9.88 (2.20)	#1	10.14 (2.24)
			#7	9.65 (2.15)
	C+G	10.25 (2.50)		
	G	11.02 (2.88)	#5 #10	11.12 (2.46) 10.96 (3.10)
VI	D	10.15 (2.17)	#4	10.01 (2.03)
			#14	10.31 (2.32)
VII	J	-----	#9	11.35 (3.83)
	J+K	10.14 (2.75)		
	K	9.68 (2.02)	#4 #13	9.53 (1.57) 9.74 (2.20)

A summary of the bench characteristics associated with each of the environments listed in Table 2, is given in Table 3 below.

Table 3 - Shovel Study Bench Environments

ENVIRONMENT	DIGGING CONDITIONS
Ia	- Compacted, rehandled waste rock half height bench.
Ib	- " " " " full " "
II	- Excavating ends of blast in very hard massive sand and siltstones.
III	- Excavating central portion of a blast in rock units of modest strength.
IV	- Shovel working at the toe of a coal seam handling hangingwall waste rock dozed to the shovel along the top of the coal seam.
V	- Working the central portion of a blast in very high strength rocks.
VI	- Excavating straight waste digging along strike, parallel to coal seam.
VII	- Digging downdip in the footwall waste of a coal seam.
VIII	- Excavating rocks of modest strength while exposing the toe of a coal seam, at times handling hangingwall dozed material.

B.6.1 Relationship Between Dig Cycle Time and Site Index.

Having established a site index for each of the environments studied, it was possible to examine the degree to which mean dig cycle times correlate with their associated environments. Eight distinct environments were defined and are presented below with their associated site index values and mean dig cycle times. A correlation analysis of site index values and associated mean dig cycle times for each environment, revealed poor correlation existed between the two.

ENVIRONMENT/STUDY	SITE INDEX	MEAN DIG TIMES	STD
I - A1-A12, Eagle III Rehandle	n/a	9.64	(2.24)
II - B1-B2, Eagle IV	unavailable	11.73	(2.99)
III - E1-E6, Eagle III	71.0	10.49	(3.03)
IVa - F1-F5, Eagle III	64.9	11.13	(2.52)
IVb - H1-H2, Eagle III	72.9	10.63	(2.26)
V - C1-C6, G1-G3, Taylor	53.3	10.25	(2.50)
VI - D1-D5, Taylor	58.0	10.15	(2.17)
VII - J1-J2, K1-K6, Taylor	52.6	10.14	(2.75)

B.7 SUMMARY.

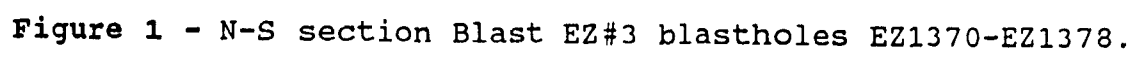
Statistical analysis of the time study data proved confounding. It was not possible to establish any sound and consistent relations between dig cycles times and environment characteristics. Analysis of the data was plagued by a multitude of variables, each of which could not be accurately parametrized and accounted for (i.e. several shovels each with a unique set of operators, working in multiple environments)

Key deficiencies of the preliminary research have been identified as:

- 1). There existed no direct means of relating mean dig cycle times with actual measures of digging effort. A key assumption of the preliminary research was that shovel performance, affected by varying degrees of digging difficulty, will be reflected through changes in mean dig cycle times. Longer dig cycle being indicative of more difficult digging conditions. This has since been proven false (on the basis of conclusions drawn from the integrated bench study employing shovel instrumentation), mean dig times show little variation between easy and hard digging conditions.
- 2). There was no way of assessing variations in the operating practices amongst operators. It was necessary to assume that all operators perform equally in equivalent digging conditions. This too, has since been proven false.
- 3). Dig cycle times alone revealed little about actual shovel productivity. Dig cycle times may have varied as a consequence of an operators attempts to increase dipper fill factors. However, it was possible that a shovel working a particular environment may have a longer mean dig cycle time than in another environment, where it may have been more productive.

APPENDIX - C

CROSS SECTIONS OF BENCH GEOLOGY.



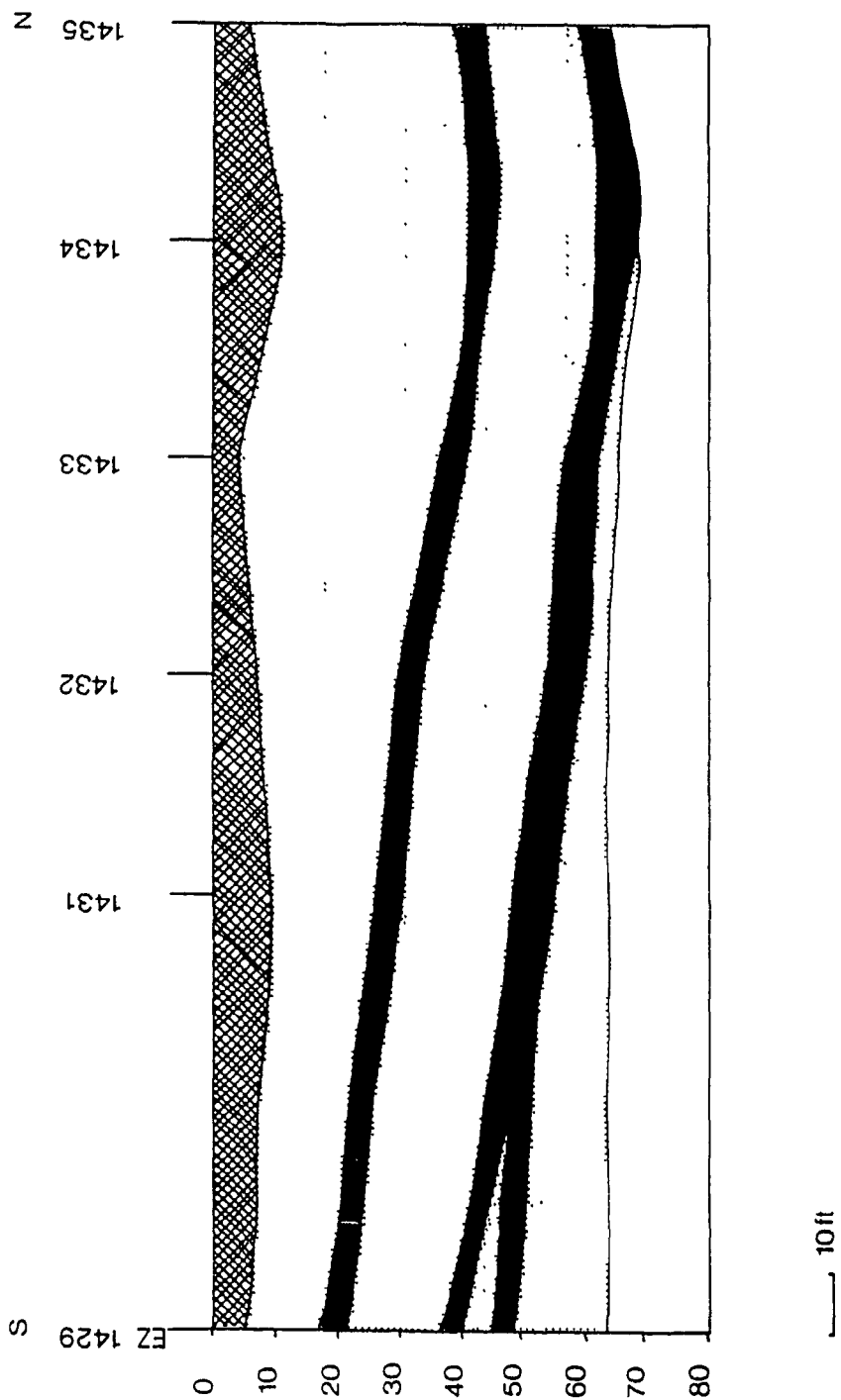


Figure 2 - N-S section Blast EZ#3 blastholes EZ1429-EZ1435.

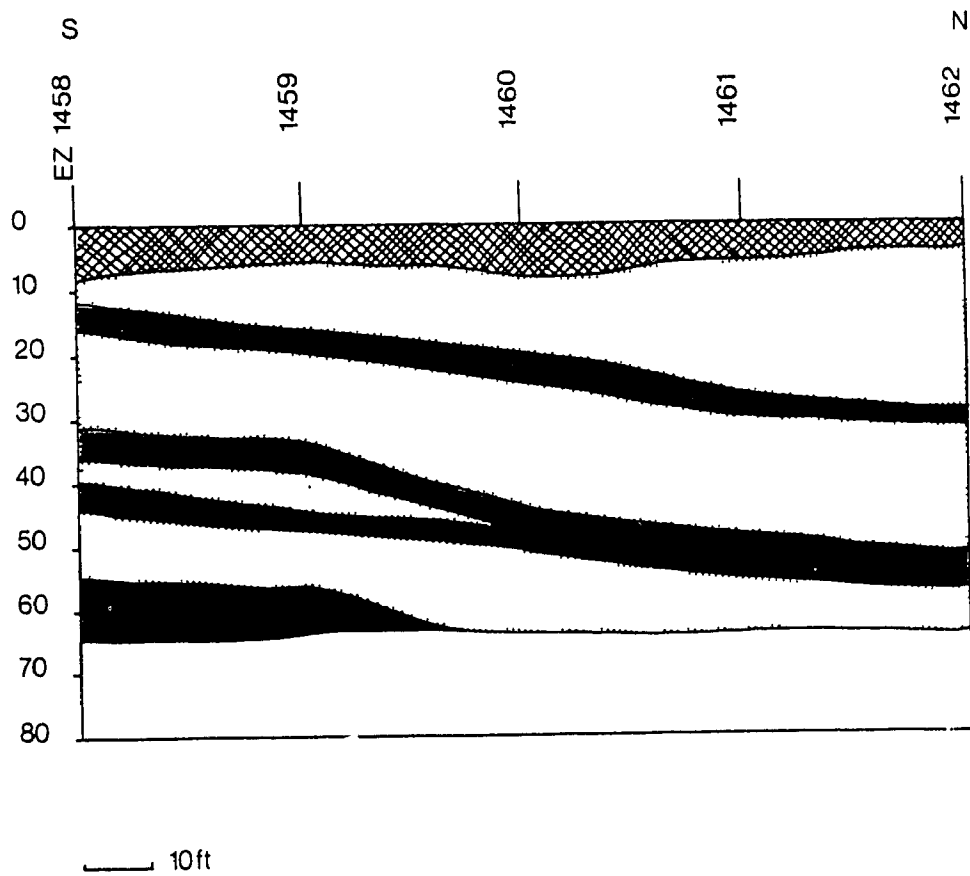


Figure 3 - N-S section Blast EZ#3 blastholes EZ1429-EZ1435.

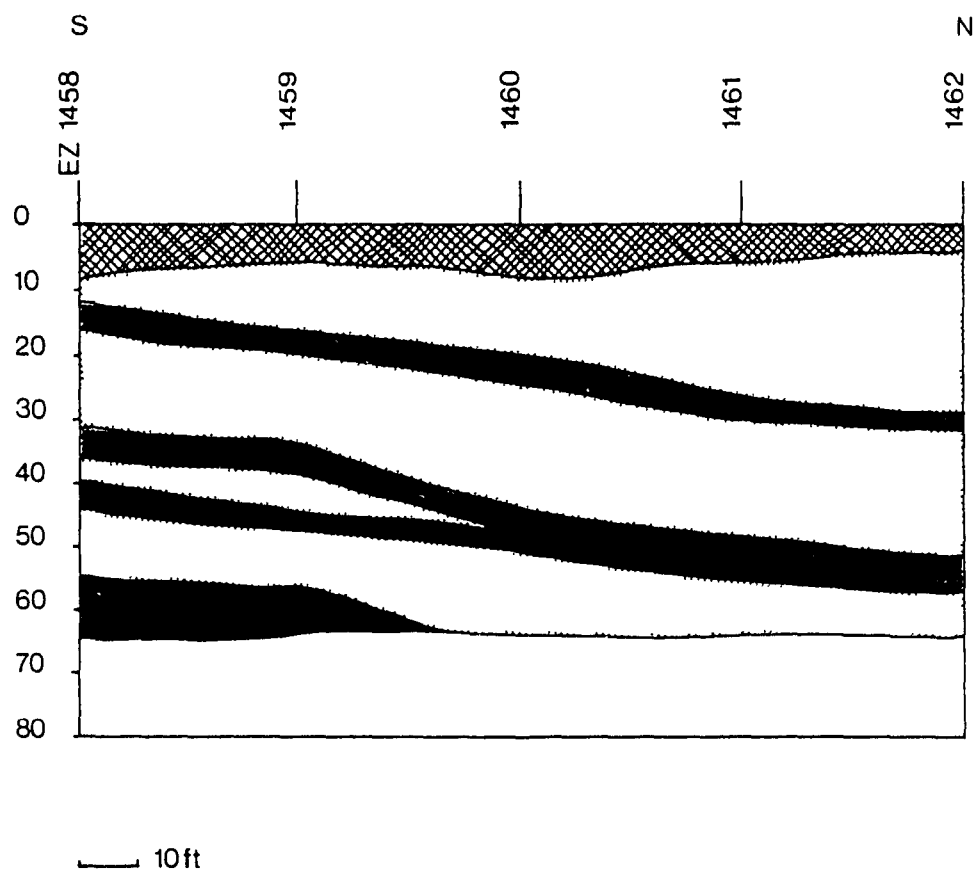


Figure 4 - N-S section Blast EZ#3 blastholes EZ1458-EZ1462.

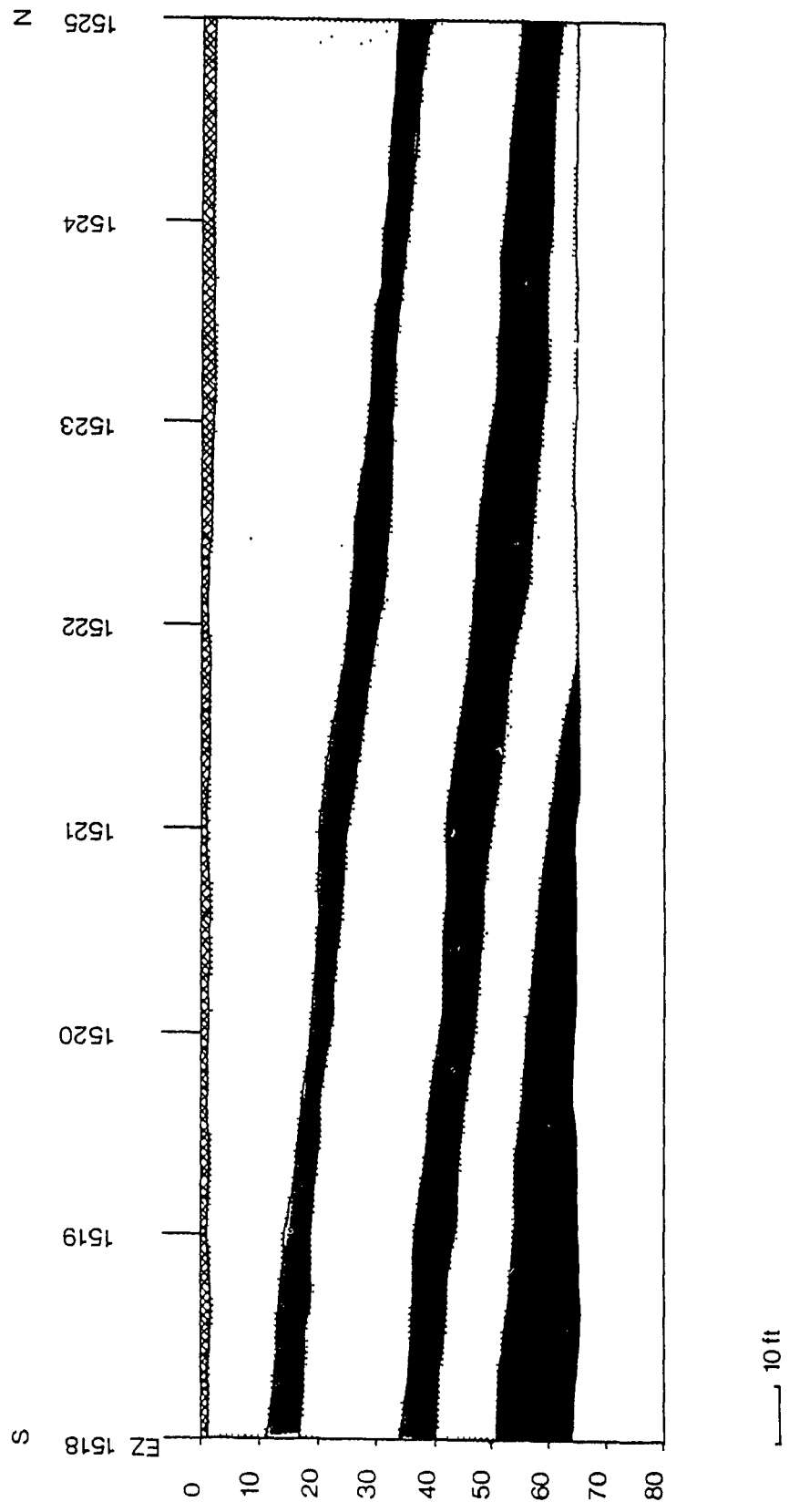


Figure 5 - N-S section Blast EZ#3 blastholes EZ1518-EZ1525.

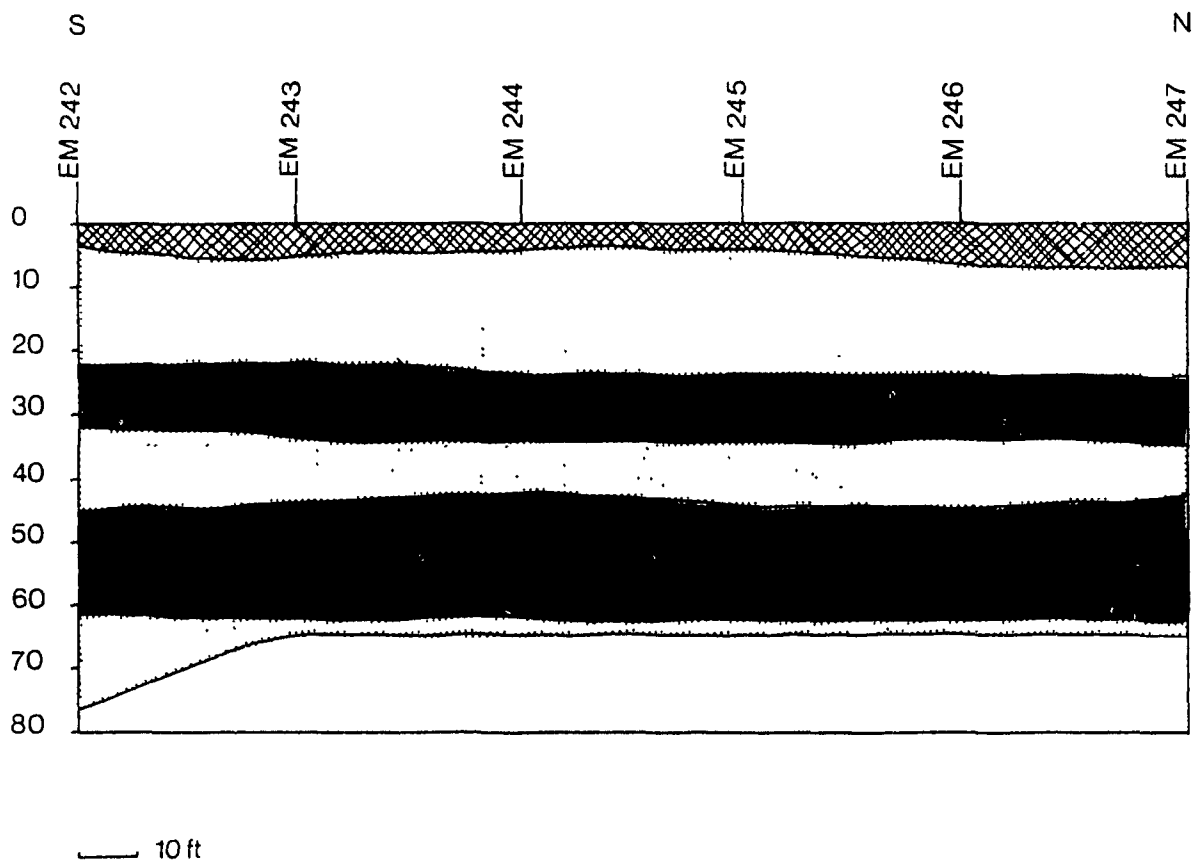


Figure 6 - N-S section Blast EM#1 blastholes EM242-EM247.

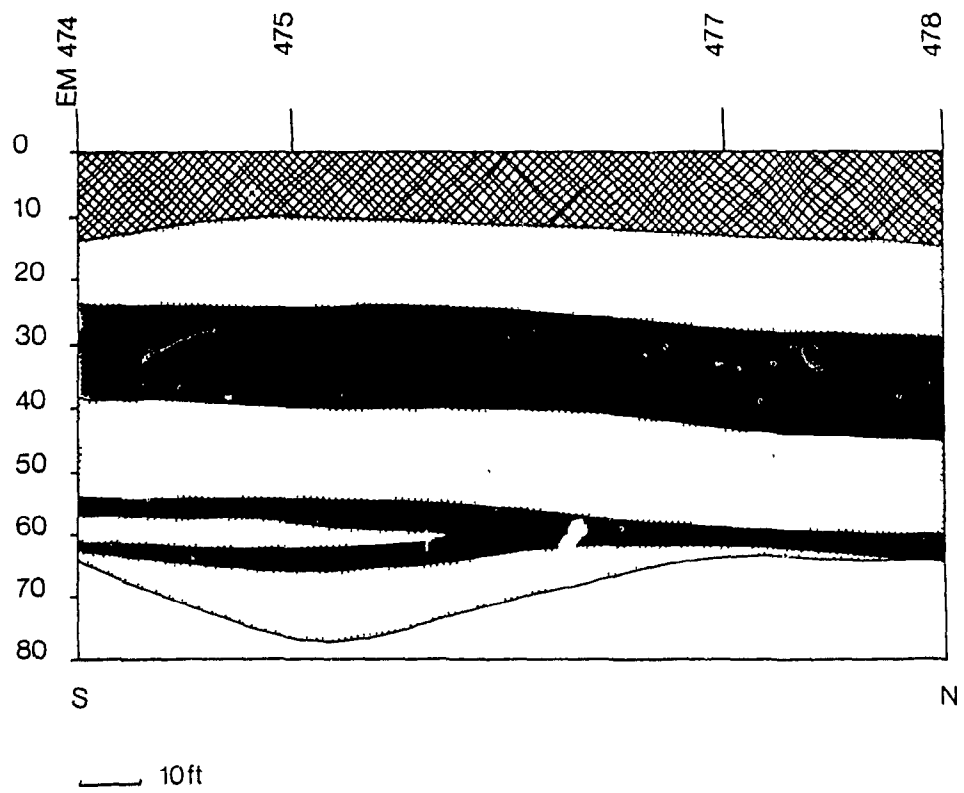


Figure 7 - N-S section Blast EM#1 blastholes EM474-EM478.

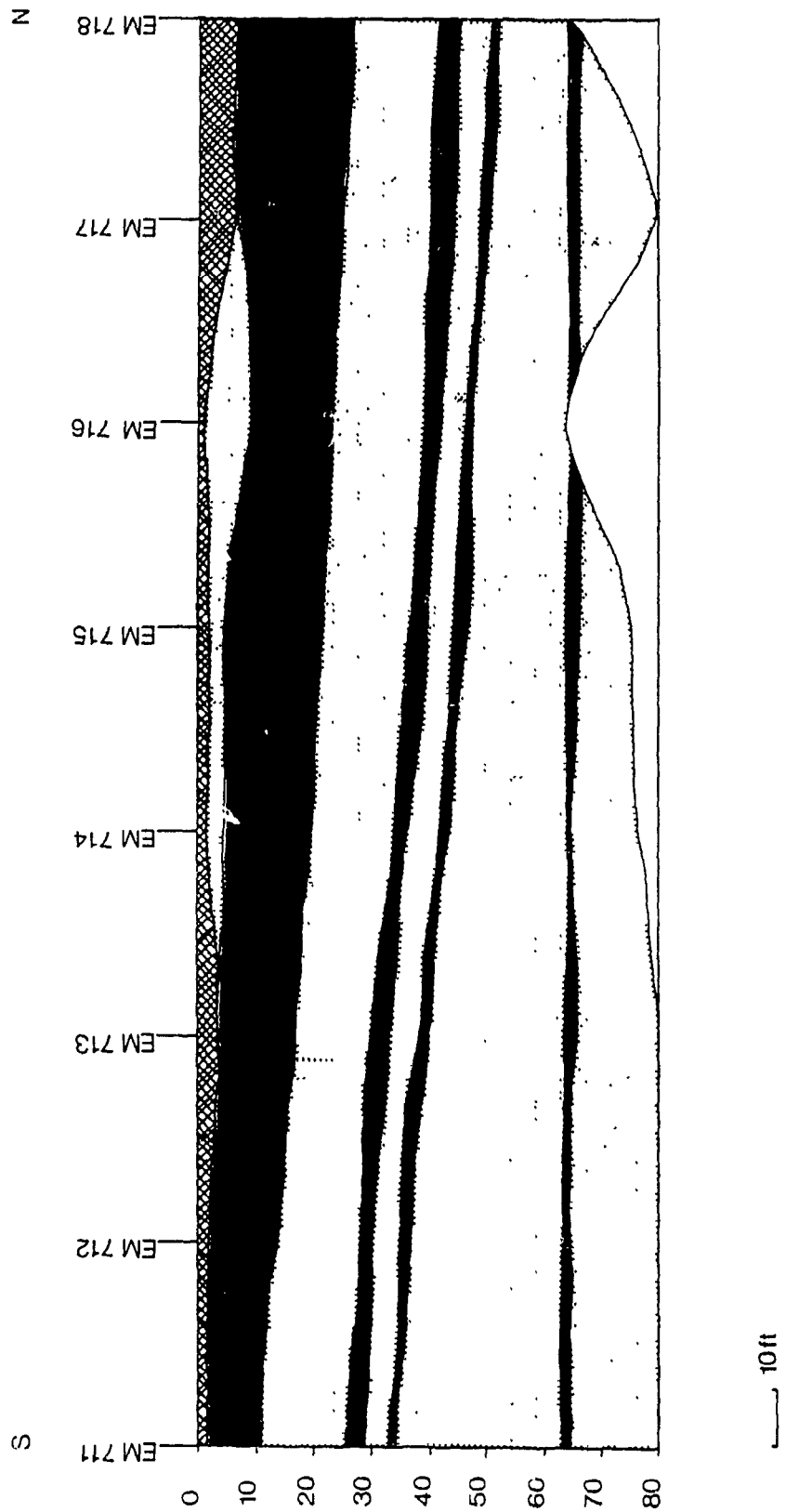


Figure 8 - N-S section Blast EM#1 blastholes EM711-EM718.

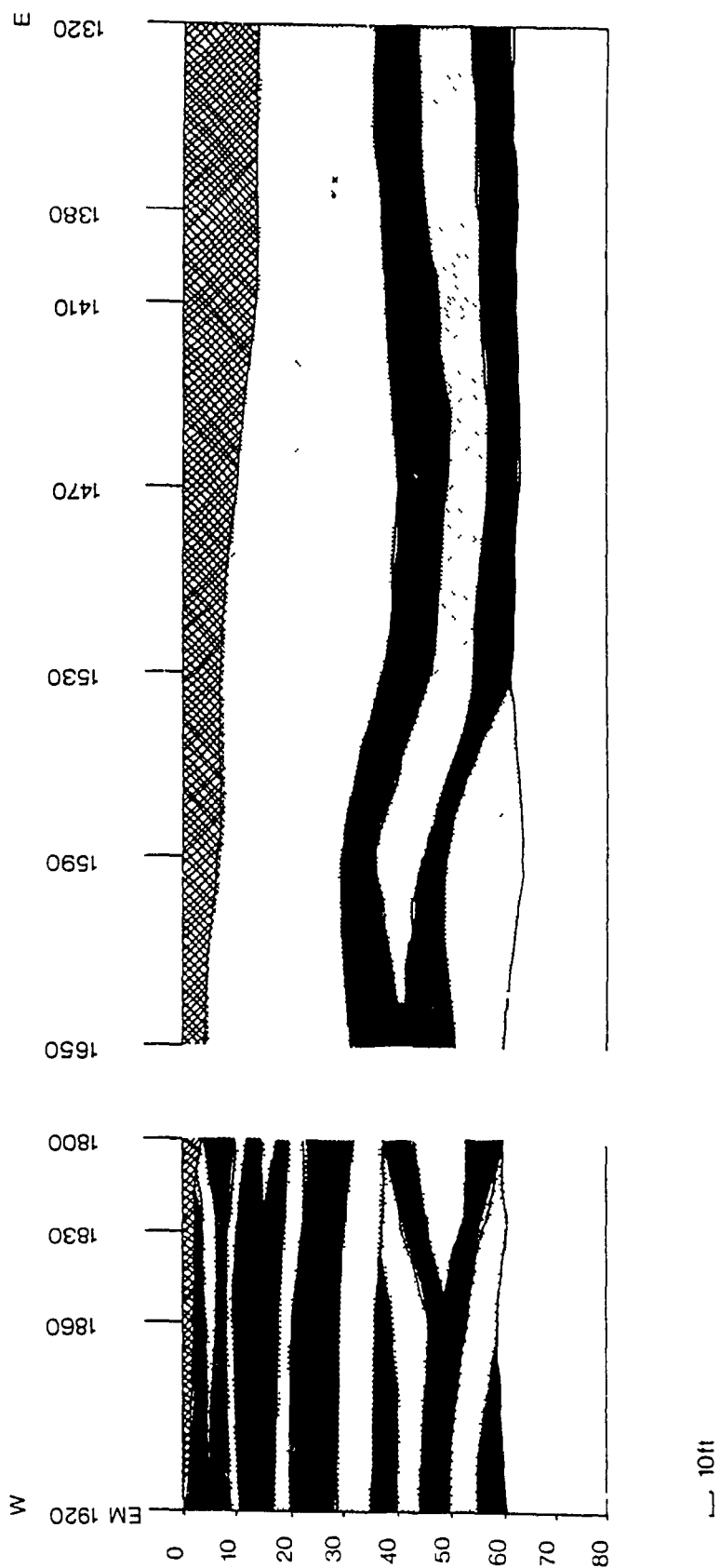


Figure 9 - E-W section Blast EM#4 along EZ1320 to EM1920.

APPENDIX D - ROCK FRAGMENTATION BY BLASTING: INFLUENCE OF BLAST DESIGN ELEMENTS AND ROCK MASS CHARACTERISTICS.

This Appendix will address modern blasting principles that relate to the design aspects of the various blasts investigated during the course of the thesis research.

Several elements combine to influence the process of rock fragmentation by blasting. There are those variables that the blasting engineer has control over such as the basic blast geometry (burden, spacing, hole diameter etc.), strengths of explosives used, delays and firing sequences. Then there are those variables over which the engineer has no control, such as the rock mass, which can vary in terms of its composition ie. stratigraphy and its structure, i.e. bedding and jointing - each of which can have a pronounced affect on the blasting process. These topic areas are addressed.

D.1 CONTROLLABLE VARIABLES - BLAST DESIGN ELEMENTS.

As defined, controllable variables refer to those elements of a blast that are readily altered by the blasting engineer to improve the blasting process. The following is a list of the main controllable variables that govern blast efficiency (Hagan 86):

1. Blast hole diameter.
2. Blast hole inclination.
3. Bench height.
4. Explosive type.
5. Shape of face.
6. Drilling pattern.
7. initiation sequence.
8. Delay timing.

For the purposes of the current research the effects of the first three variables may be negated. The same drill was used throughout the integrated bench study. All blastholes were 12 1/4" (310 mm) diameter and drilled vertically in benches of uniform height 50 feet (15 m). This section will elaborate on the remaining six controllable variables, and how the effects of their variation pertain to the current research.

D.1.1 Explosive Type - Strength.

Explosives must develop sufficient detonation and explosion pressures to cause rock displacement and failure strains; minimum threshold pressure requirements. Following this, adequate energy and sustained pressures are required for desired fragmentation and displacements to be executed.

High initial, or detonation pressures, are generally related to denser explosives of higher detonation velocities. Such explosives are more costly than those of lower density and detonation pressure.

High strength, competent, rock masses little modified by structural elements (bedding, jointing etc.), typically require "high brisance" explosives to effect initiation of blast fragmentation. High initial pressures commonly cause crushing of the borehole wall. Excessive crushing in rocks not requiring high initial detonation pressures is indicative of unwarranted expenditure of money for high brisance effect.

Having employed explosives of suitable threshold pressure

characteristics, an important event involving sustained explosion or borehole pressures follows. Fragmentation is a time consuming stage in which crack growth events, ground displacement and rock mass heave all occur. Each of which influences muckpile diggability and fragmentation size.

Many rocks have modified mass strength due to the presence of joints, fissures, bedding planes and other structural features. Such rockmasses are often blasted to desirable muckpiles through heave, with simple parting of the inherent structures resulting in good fragmentation sizing. Explosives with good energy content and sustained explosion histories, are usually more effective in generating desirable blast results than high brisance explosives, and are less costly.

The bulk handling and explosives blending systems employed at Fording, enable the loading of explosives into blastholes to develop a range of detonation velocities (and pressures). This permits matching of explosives characteristics to blasting site requirements.

The actual strength value of an explosive can be expressed through a variety of means. This often results in confusion and makes relating just how much stronger one explosive is than another a difficult parameter to communicate (Lownds, 1986). The simplest, and most commonly accepted means of relating the strength and hence expected performance of explosives is on the basis of their relative weight or bulk strengths (Lang, 1972).

For example, a borehole may be loaded with prilled AN/FO with an average density of 0.88 gm/cc, and a calculated thermochemical energy of 760 cal/gm. The same hole may also be loaded with an emulsion based explosive of density equal to 1.40 gm/cc and a relative weight strength of 80. The relative bulk strengths of these two explosives would be 100 for the freely poured AN/FO and 112 for the emulsion.

The available range of bulk energies available will be administered to the confining rock over a considerable range of detonation and borehole pressure delivery mechanisms. The emulsion detonates at 5-6 km/sec and AN/FO at 4.2 km/sec. Final selection and matching of explosives to ground characteristics requires monitoring and evaluated performance comparisons, since the theoretical analytical basis is inadequate for complete application in practice.

D.1.2 Shape of free faces.

A component of the current research is involved in attempting to evaluating the efficiency of two types of blast patterns; square, and staggered equilateral which in theory provides for a more optimal distribution of explosive energy in two dimensions.

It is important that each blasthole be afforded as much free space to break and expand into as is possible. This is best achieved according to Hagan (1986) by creating a situation in which "The angle subtended at the axis of a blasthole by that part of a face which is reasonably near should be as large as possible". The conditions defined by optimal face shape are shown diagrammatically in Figure 1. If a charge is not provided with a sufficient free face, the expansion of the rock mass around the blasthole will be constrained and the resulting muckpile tight and difficult to dig.

Optimal, highly effective (ie. near and extensive) free faces are most readily developed by employing staggered V1 blasts and shallow V-type blast patterns (Figures 1 and 2). The free faces developed by these patterns allow individual blastholes to shoot to a body of rock which approximates a semi-cylindrical protrusion from the face (Figure 4). Figure 1 illustrates the concavely biplanar faces to which charges fire in a V1-type staggered (equilateral) pattern. Fragmentation generated by these types

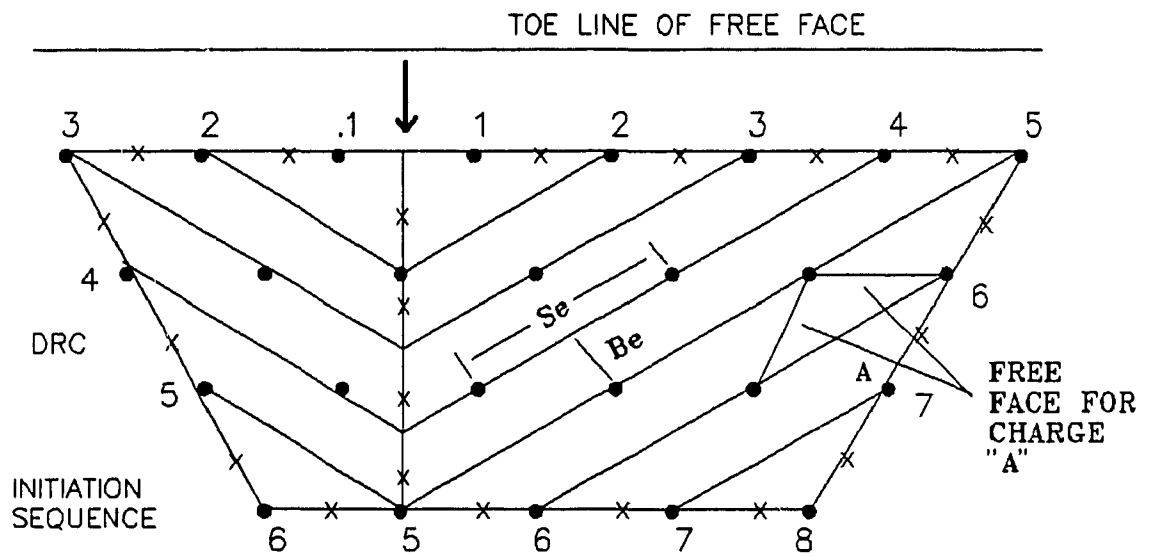


Figure 1 - Staggered "V1" blast pattern (after Hagan, 1986).

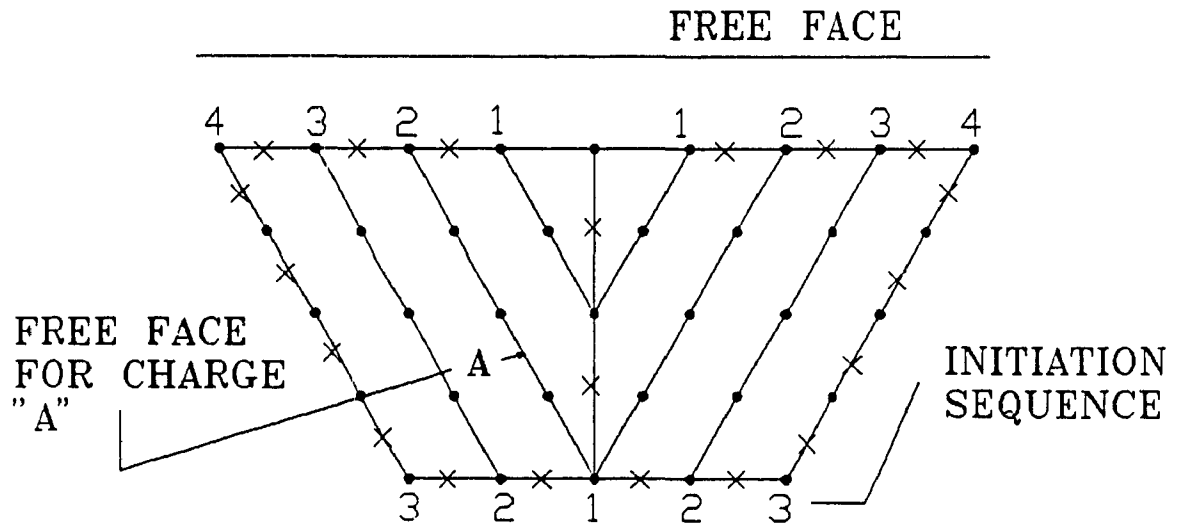


Figure 2 - Staggered "V" blast pattern (after Hagan, 1986).

of blasts tend to be superior to that produced by the more commonly applied square V patterns. Figure 3 shows that in a square V sequence individual charges shoot to planar faces which do not represent as optimal a free face as those developed in a staggered V1 sequence.

The formation of "effective" concavely biplanar faces do not develop until the conventional value of effective blasthole spacing: effective burden distance, $Se:Be$, exceeds 2.6 (Hagan, 86). The spacing to burden ratio for a square V pattern is 2.0, that for a V1 staggered pattern drilled on an equilateral grid is 3.5. The qualifying term "effective" refers to burden and spacing values actually experienced during the blast, they are a consequence of the way in which charges are tied in with one another - the initiation sequence.

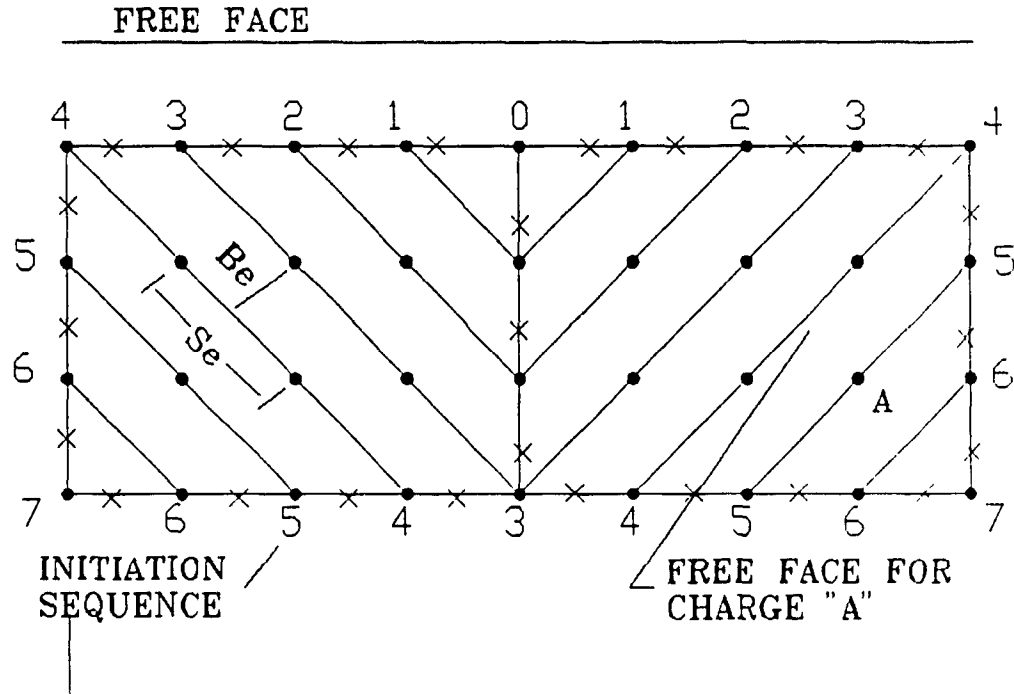


Figure 3 - Square "V" blast pattern (Golder, 1984)

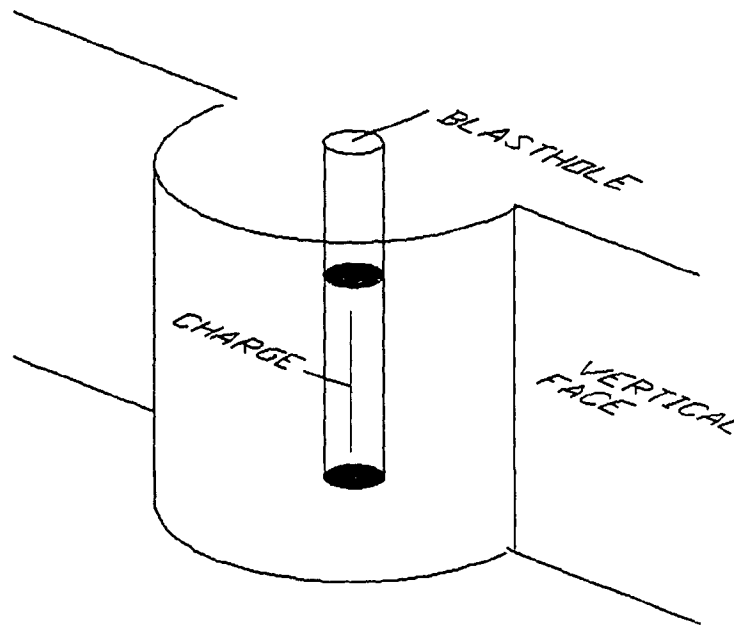


Figure 4 - Ideal shape of free face (after Hagan, 1984).

Figure 5 illustrates how principles of initiation sequence, drill pattern and measures of effective burden and spacing apply to the drill patterns investigated in the test bench.

D.1.3 Blasthole Drilling Pattern.

There are three basic blasthole drilling patterns in use in most surface operations; (1) square, (2) rectangular and (3) staggered. In a square pattern the burden B is equal to the spacing S , and holes in each row are aligned directly behind holes in the front row.

In a rectangular pattern the burden B is less than the spacing S , again holes in each row are aligned directly behind holes in the front row.

Staggered patterns may have burdens equal to spacings but are usually employed with burdens less than spacings ($B < S$). The staggered patterns employed by the mine are equilateral triangular patterns with burdens = 9.3 metres and spacings = 10.7

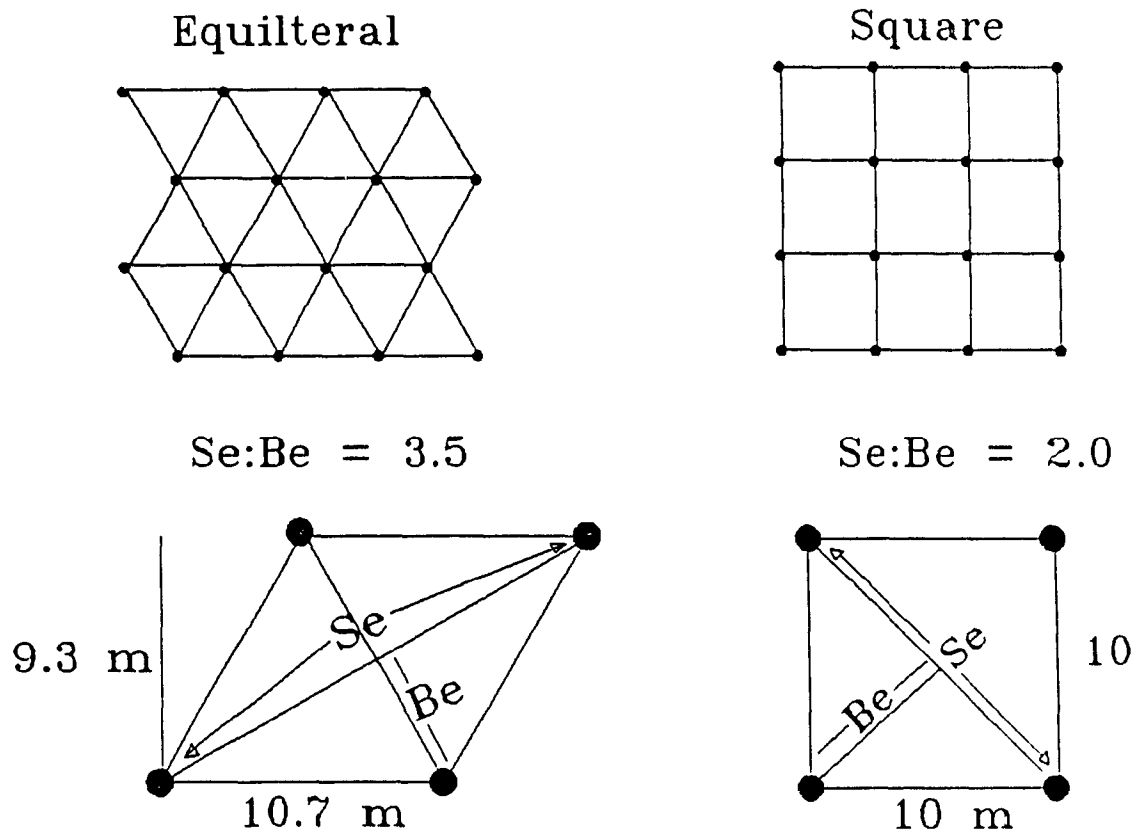
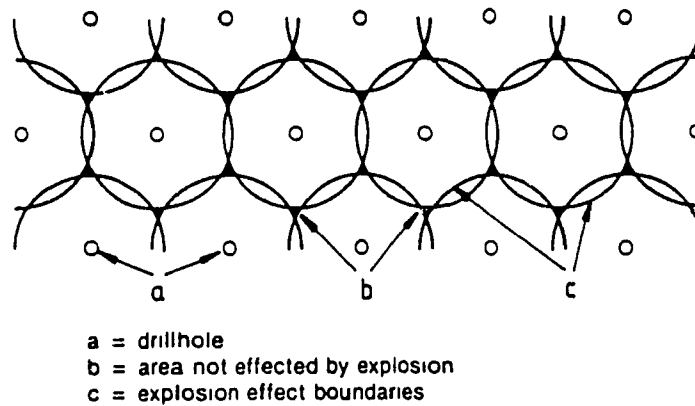


Figure 5 - Blast tie-ins; Square and equilateral patterns.

metres. In such patterns the blasthole spacings are equal to 1.15 times the burden distance. An equilateral pattern allows a high degree of flexibility in the initiation sequence, which determines direction of firing.

Errors in drill hole position in a staggered equilateral pattern can have pronounced effects on the efficiency of the blast. Figure 6 shows that although these patterns do provide optimal distribution of explosive energy, an error of only 10% in hole alignment can result in large zones in which poor rock breakage has occurred.



Triangular pattern and explosion energy distribution

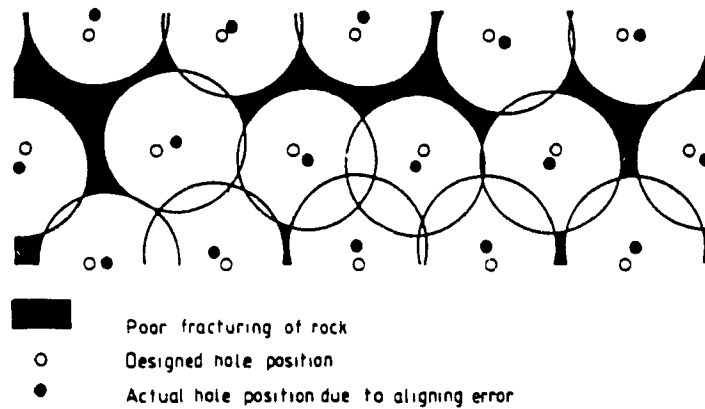


Figure 6 - Distribution of explosion energy in equilateral pattern (Tamrock, 1984)

D.1.4 Initiation Sequence.

The initiation sequence of a blast represents the manner in which individual blasthole charges are linked with one another (blast tie-in), and hence, the order in which they fire. This is a critical consideration in the selection of blast pattern and the overall blast design since it has considerable control over the final values of Be and/or Se (effective burden and spacing). An initiation sequence should (where possible) satisfy the following criteria:

1. Individual charges should shoot to optimally developed free faces.
2. The ratio of effective spacing to burden be 3.5 or more.

3. The blastholes are "effectively" (actual staggering as a consequence of the blast tie-in) staggered with good balance.

The above conditions are most readily accomplished by employing staggered V or V1 type patterns superimposed on an equilateral triangular grid (Hagan, 86), Figures 1 and 2. A "V" type pattern is one in which the tightest deepest firing of blastholes is accomplished (Figure 2). As the angle within the V increases the resulting changes in initiation sequence give way to the development of V1 to V2 to V3 firing sequences.

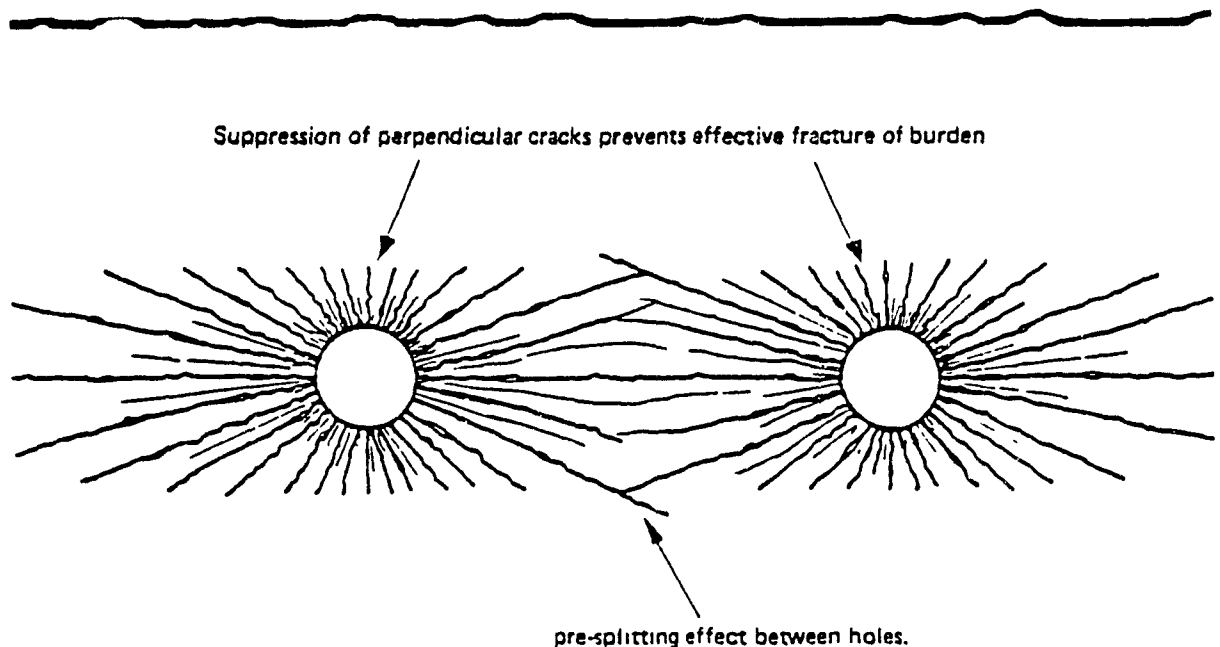


Figure 7 - Planer inter-blasthole split.

Figure 2 shows a V pattern in which the distances between simultaneously detonating charges are too short. In this pattern effective burden and spacing (Se, Be) are similar in dimension. This creates a tendency for cracks (radiating from a blasthole upon explosive detonation) to run preferentially between holes before the full burden has been properly fragmented (Anon, 78), Figure 7. They create a truly planer inter-blasthole split. Conversely, referring to Figure 8, in which optimal biplanar free

faces generated by a V1 pattern control and direct explosive energy outward into the actual burden without interference from simultaneous detonation adjacent blastholes. In a V1 pattern simultaneously detonating charges in a row are too far apart to permit planar inter-blasthole splits. These distances are of course a function of Se:Be ratios.

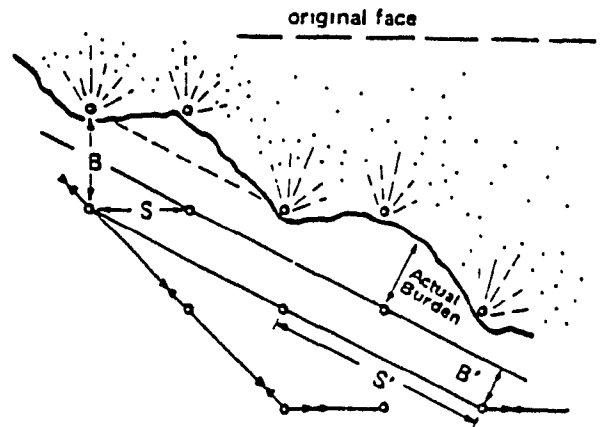


Figure 8 - Development of biplaner free faces.

D.1.5 Delay Timing.

The delay interval between rows in multiple row blasts is of key importance in determining the resulting muckpile looseness and fragmentation. The effect of the delay interval is to provide sufficient time to develop a free face in front of each round in the blast. Optimal fragmentation occurs when the delay interval is sufficient to allow each hole to detach its own quota of rock and displace it horizontally before the next hole detonates. Ultimately the delay should be of sufficient duration to separate the explosive effect of each row of the blast, from that of the preceding one.

Actual values for inter row delays should be in the range of 5 ms per meter of effective burden for short collar, high energy blastholes in strong massive rocks. For blastholes in weaker rocks, low energy factors and longer collared zones, the interval should be about 10 ms per meter of effective burden. With the use of down hole delays, inter row delays of up to 30 ms per meter of effective burden can be achieved (Hagan, 86), without increasing the risk of cut offs.

Delay intervals during the study ranged from a maximum of 14 ms per meter of effective burden in the equilateral blast EM#1, to a minimum of 7 ms in the front half of the square blast EZ#3. The back half of EZ#3 was fired with a delay interval of 10.5 ms.

Too short a delay interval will result in cratering by limiting the horizontal displacement of successive burdens. The resulting muckpile will be high and digging tight (Lang, 72, Langefors, 67 and Hagan, 1974).

Delay accuracy - effects of cap scatter.

It is important to consider that the true delay intervals of pyrotechnic delays do not necessarily conform precisely, with the nominal firing times quoted by the manufacturer. Differences between the measured firing time of a detonator and its suggested firing time, are referred to as delay error (Stagg et al. 1987, Helig et al. 1988 and Winzer et al. 1979). In large multi-round blasts, delay errors result in a phenomena referred to as "cap scatter", which can over-ride the designed sequencing and timing of all delays in a blast. Given the significant influence of blast sequencing on blast results (ie. fragmentation, heave and looseness), it is possible that scatter in the assumed simultaneous detonation of rows in multi-round blasts, could result in localized variations in post-blast conditions.

D.2 UNCCONTROLLABLE VARIABLES - THE ROCKMASS.

This section will address those aspects of a blast over which the blasting engineer has no control; the physical or geotechnical properties of the rockmass.

A rockmass can be described on the basis of two separate and distinct sets of physical properties. The first pertain to the physical characteristics of the rock units that make up a rockmass and are usually determined in the laboratory (ie effects of the rock composition). The second set refer to the spatial

manner in which the various units that form a rockmass are distributed and the structural nature of each unit, as determined in the field (ie. jointing, bedding etc.).

D.2.1 Physical Properties of Individual Rock Units.

Several authors have reported on the influence of numerous physical properties of a rock on the blasting process (Rustan, 1983, Lilly, 1986, Singh, 1986, Langefors 1967 and Hagan, 1980). It has been claimed by Lilly, that virtually every parameter used to describe a rockmass will, to some degree, influence the rock mass' response to an explosive detonated within it. The parameters most commonly cited as influential are: its elastic properties (Young's modulus, Poisson ratio etc), tensile strength, shear strength, density, compressional wave velocity, stress wave attenuation characteristics, Hugoniot elastics limit, compressive strength, ground water and moisture content and porosity.

A more practicable approach to the problem was proposed by Lang and Favreau, 1972, in which the following "physical and mechanical" rock properties were identified as imparting a significant degree of influence over the behaviour of a rockmass during blasting: Compressive strength, tensile strength, elastic properties, longitudinal wave velocity and density.

The compressive and tensile strength values relate to the largest stress levels the rock can withstand before failure occurs. Rock units in which these strength values are high will require a higher stress level for failure to occur. It follows therefore, that at a constant explosive energy level, the units with higher ultimate strength values will not be as well fragmented as those with lower strengths.

The combined influence of key rock physical properties on the explosive process can be summarized, according to Singh (1986) as follows:

"At constant energy levels, higher compressive strengths, Young's modulus, Poissons ratios, tensile and shear strengths indicate less efficient blasting... Higher degree of saturation lead to less energy absorption and better stress transmission, resulting in greater fracturing..."

It is claimed (Lilly, 86) that from a practical stand point, very few of a rocks physical properties impart a significant degree of control over blast performance, namely the specific gravity and hardness of the rock. The influence of the rock mass' structural nature (jointing etc) will bear much more heavily on the resulting fragmentation, than will any of the remaining physical properties discussed (Ash, 1973).

D.2.2 Structural Geology of the Rock Mass.

It has been well documented by numerous researchers that the structural nature of a rock mass imparts a significant degree of control over the blasting process (Belland, 1965, Ash, 1973, Larson, 1973, da Gamma, 1977 and 1983, Singh and Sharma 1983, Yang and Rustan, 1983, Lilly, 1986 and Cunningham, 1987). The effects of geologic discontinuities and their attitude far outweigh the effects of a rockmass' physical-mechanical properties, and in some instances even the effects of blast geometry (burden, spacings etc).

In considering all of the variables that combine to govern the overall performance of a blast, the most difficult to accurately parametrize is the effect of local structural geology. The presence of joints, faults and bedding planes in a rockmass can be responsible for producing blast results quite different than those from a portion of the same rock mass that is more homogeneous in nature.

Work by Lilly, 1986, proposed an empirical method of assessing rock mass blastability based primarily upon the effects of the rockmass' structural properties. Input parameters used to predict blastability by this method are:

1. Structural nature of the rock mass ie. blocky or massive or powdery.
2. Spacing and orientation of weakness planes such as joints, faults, bedding planes or schistocities.
3. Specific gravity of the material.
4. Hardness of the material.

The effect of discontinuity spacing is readily understood in that it essentially defines the dimension of blocks in the rockmass before blasting. In a rockmass with closely spaced joint sets lower explosive energy factors would be required to achieve good fragmentation than in a more massive rock mass where spacing is on the order of meters.

Conversely, the effects on the blasting process from the orientation of weakness planes within the rock mass is more complex and less clearly understood. It has been demonstrated by Singh and Sarma, 1983, that joint orientation relative to a bench free face will influence fragmentation size, diggability and bench floor profile. It is understood that a rock mass with predominantly horizontal weakness planes will be easier to effectively blast than a bench in which joint planes dip in the direction of throw - yielding slabby muck and a saw toothed floor (Lilly, 1986).

Recent work by Singh et al. (1986) has focused on the application of rockmass characterization to blast design in surface mine operations. Separate mine environments were identified/characterized on the basis of both the Geomechanics Classification (RMR System) and Rock Mass Quality (Q System) rock mass characterization systems.

The RMR system was originally developed by Bieniawski in 1973 for use in tunnel design in a jointed rock mass. A Rock Mass Rating, based on the following parameters was determined for separate mine environments:

1. Uniaxial compressive strength of intact rock
2. Rock Quality designation (RQD)
3. Spacing of discontinuities.
4. Conditions of discontinuities.
5. Ground water conditions.
6. Rating adjustment for joint orientations.

Characterization of the same environment by the Q System incorporated the following parameters based on laboratory investigations and field mapping:

1. Rock quality designation (RQD).
2. Joint set number (J_n).
3. Joint roughness number (J_r).
4. Joint alteration number (J_a).
5. Water reduction factor (J_w).
6. Slope adjustment factor (SAF)¹.

The value of the Q system is calculated as follows:

$$Q = (RQD/J_n) \cdot (J_r/J_a) \cdot (J_w/SAF)$$

Singh's use of these classification systems and their dependence primarily upon the spacing, nature and orientation of discontinuities, accents their pronounced influence on blast efficiency. Through studies of separate blasts it was determined that; "the line of blastholes drilled parallel to a predominate joint set generally results in a higher rock recovery and a more uniform stable back face with good fragmentation" (Singh 1986). This conclusion is significant in that it underscores the importance of joint (or other planes of weakness) orientation on blasts performance. As stated, the influence of the orientation of weakness planes is not as apparent, or readily defined, as that of their spacing. The spacing will simply define the size of

¹. The SAF, used for stability and blast evaluation purposes is a modification of Barton's Stress reduction factor (SRF).

in-situ blocks in a rock mass. On the other hand the orientation of these planes in a rock mass will influence the directional flow, upon detonation, of explosive energy in rock mass and hence, the manner in which the rock mass disintegrates eg., the shapes of blocks, the direction in which they are thrown during the blast and consequently their orientation in the muckpile, which may influence diggability. Prior research along similar lines by Belland (1965), and the U.S. Bureau of Mines (Larson et al., 1973) support the findings by Singh.

More recently, research undertaken at Luleå University in Sweden (Yang and Rustan, 1983), has aimed at obtaining quantitative data on how weakness planes influence fragmentation. Their effect has been determined a function of orientation, frequency, joint filling material and strength of the weakness plane. The role of each of these factors on fragmentation was examined by studying the results obtained through single hole shots in each of 40 magnetite-concrete models. Four types of models with the following structural characteristics were investigated :

1. Homogeneous blocks without weakness planes.
2. Blocks constructed with a single weakness plane parallel to the blasted surface.
3. Blocks constructed with weakness planes both parallel and perpendicular to the blasted surface.
4. Blocks constructed with several short weakness planes (to represent 3 or more joint sets in full scale).

It was demonstrated that the shape of the breakage contour is dependant upon the orientation of weakness planes and charge concentration (Figure 9). Regarding the influence of weakness plane orientation on fragmentation, the following main conclusions were drawn:

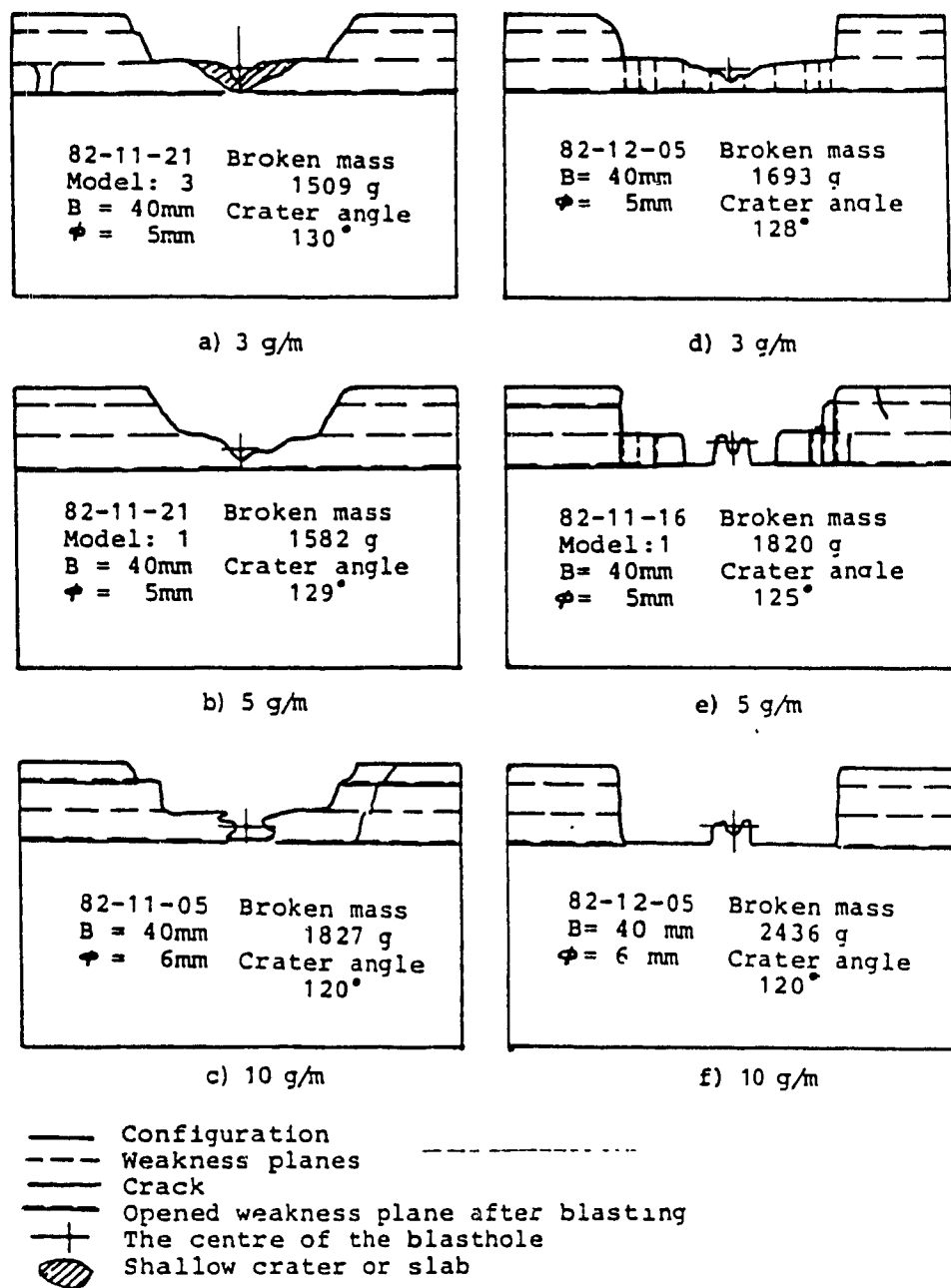


Figure 9 - Influence of weakness plane orientation on breakage contour (Yang et al., 1983).

- As is illustrated by Figure 10, variations in charge concentration, from 3 to 10 g/m, affect the greatest change in fragmentation, k_{50} , in those models with an homogeneous structure (5 times as small), in models with weakness parallel to the blasted surface, the effect of increased charge concentration on k_{50} was of moderate significance (3 times). Finally, in models with weakness planes both parallel and perpendicular to the blasted surface, increases in charge concentration realized the least effect on fragmentation. It is evident, based on these results, that the difficulty in improving fragmentation is proportional to the number of weakness planes in a rock mass.

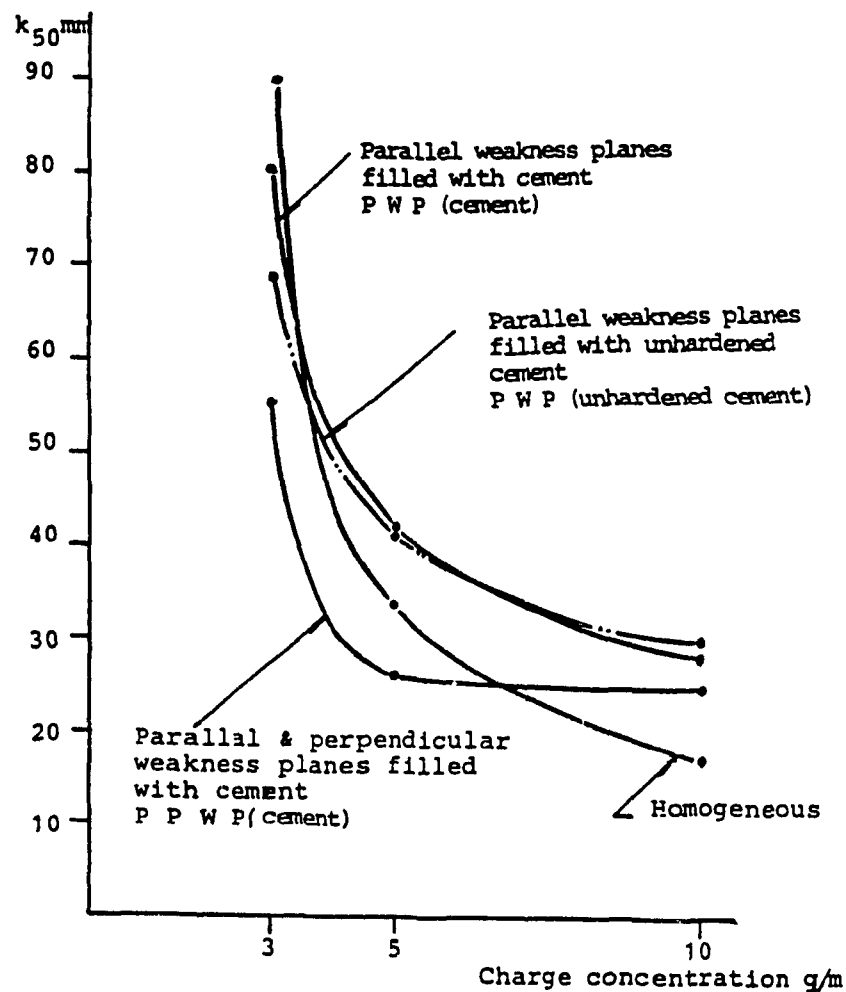


Figure 10 - Effect of charge concentration on fragmentation (Yang et al., 1983).

- It was demonstrated that in models characterized by low and medium charge concentrations (3-5 g/m) and burden of 40 mm, an increase in the number of weakness planes will produce a smaller k_{50} .

- When the same models are subjected to higher charge concentrations (5-10 g/m) there is little change in k_{50} . However, as the number of weakness plane were increased a corresponding decrease in k_{50} and an increase in the volume of breakage was observed.

These conclusions are of interest to the current research in that as stated, a portion of the test bench was intentionally loaded with uncommonly high strength charge concentrations. The heavily charged blastholes were in a bench location characterized by numerous weakness planes (bedding and jointing). The resultant fragmentation in the heavily charged zone was no better than that observed in the surrounding bench area, in which explosive columns of standard strength were employed.

D.3 SUMMARY.

This Appendix has introduced a fundamental overview of the influence of various blast design parameters on the resultant muckpile. The prior works presented, clearly establish rock mass structure (joints, bedding etc.) as a primary control in the success of blasting - fragmentation and diggability. As stated it is not the objective of the current study to advance the nature of research presented in this Appendix. However, several of the aspects of rock fragmentation discussed here within warrant consideration and have been addressed towards the current research where appropriate.

APPENDIX - E

DAILY AVERAGE FRAGMENTATION SIZE DISTRIBUTIONS.

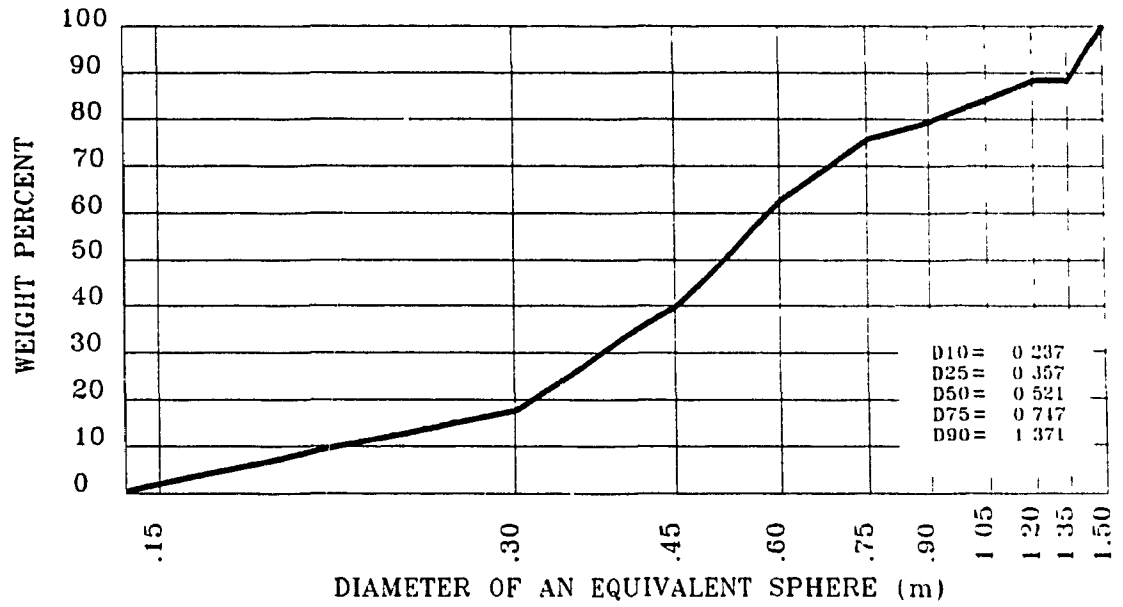


Figure 1 - Cumulative weight percent passing - August 15th.

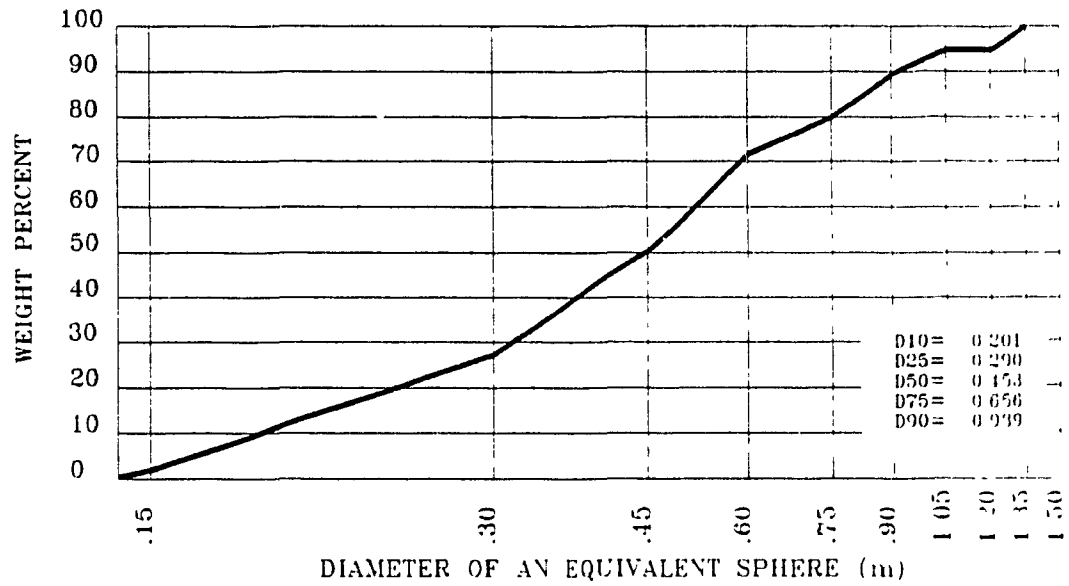


Figure 2 - Cumulative weight percent passing - August 17th.

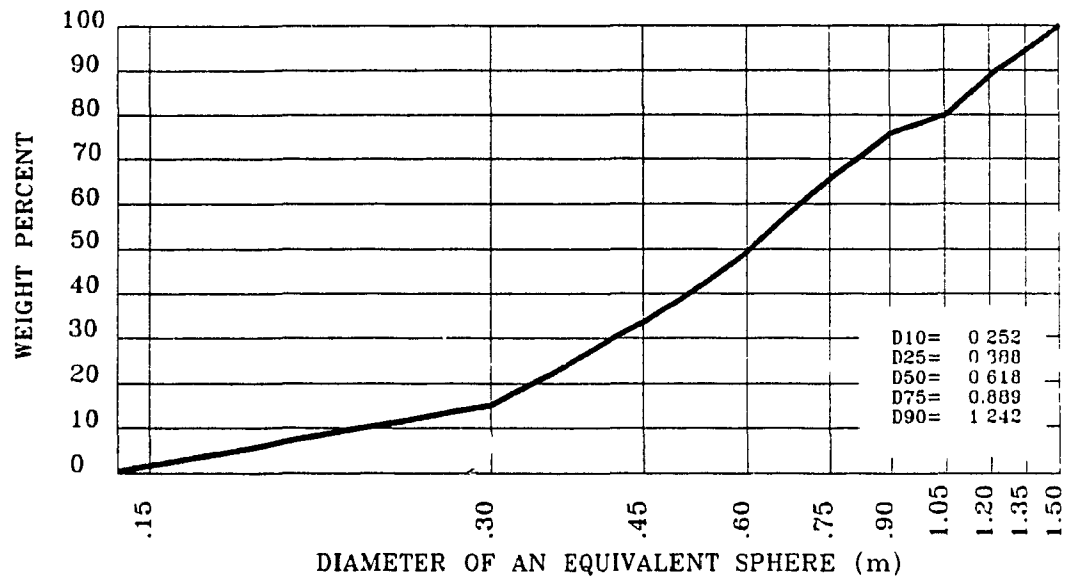


Figure 3 - Cumulative weight percent passing - August 18th.

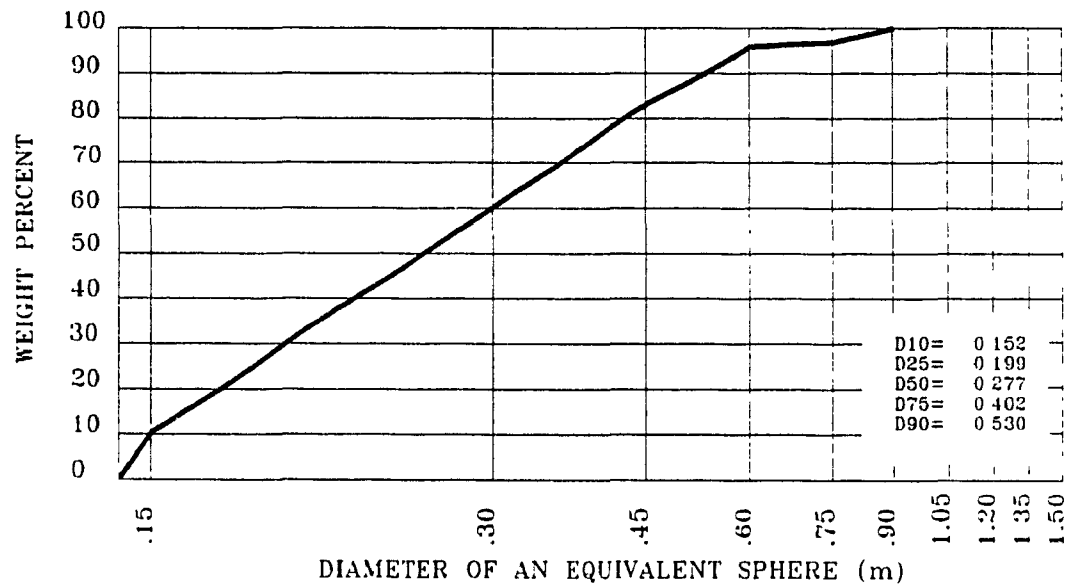


Figure 4 - Cumulative weight percent passing - August 19th.

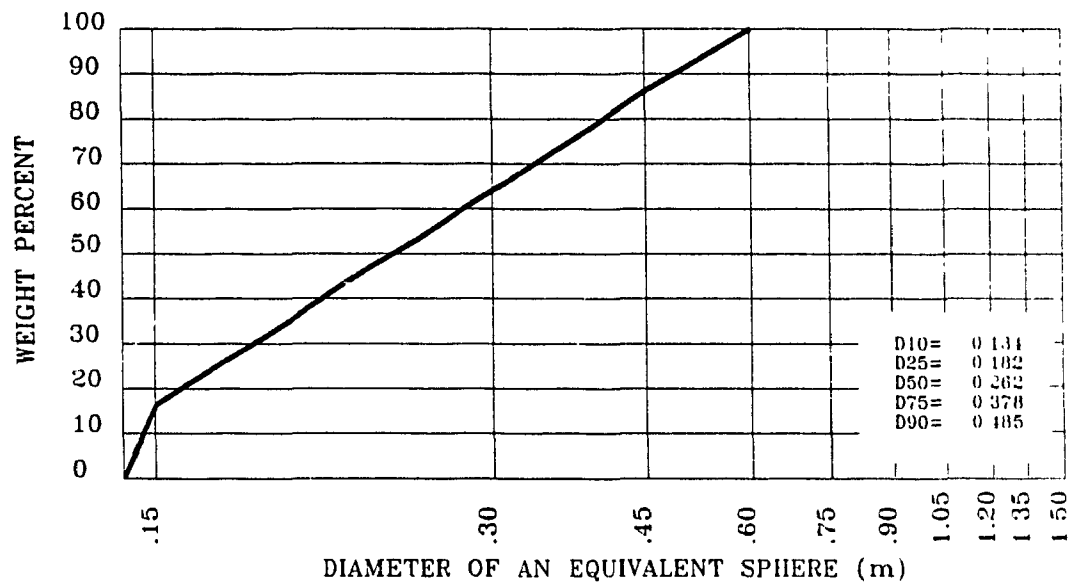


Figure 5 - Cumulative weight percent passing - August 20th.

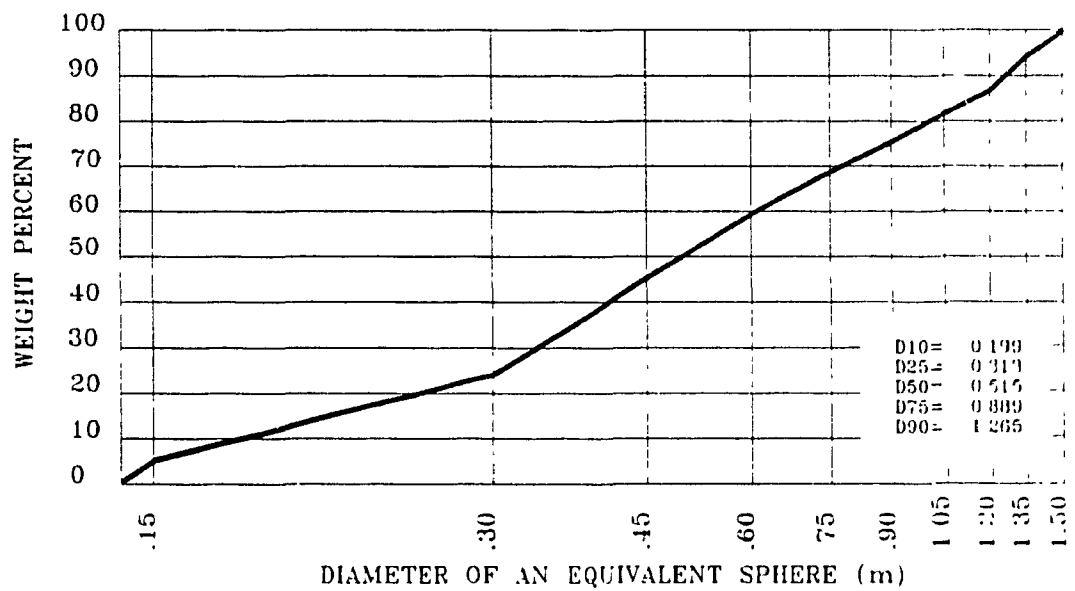


Figure 6 - Cumulative weight percent passing - August 21st.

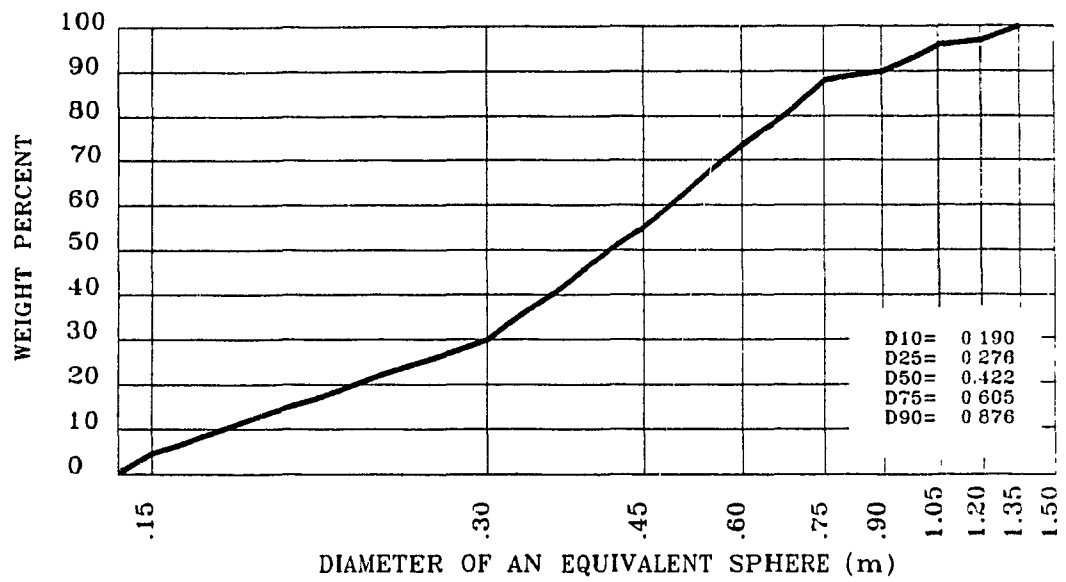


Figure 7 - Cumulative weight percent passing - August 22nd.

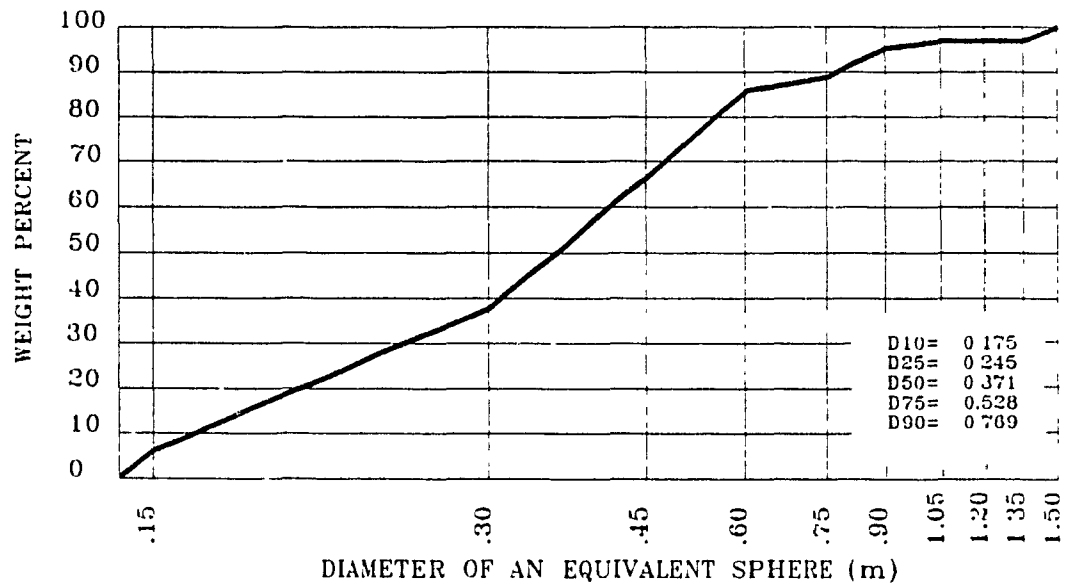


Figure 8 - Cumulative weight percent passing - August 23rd.

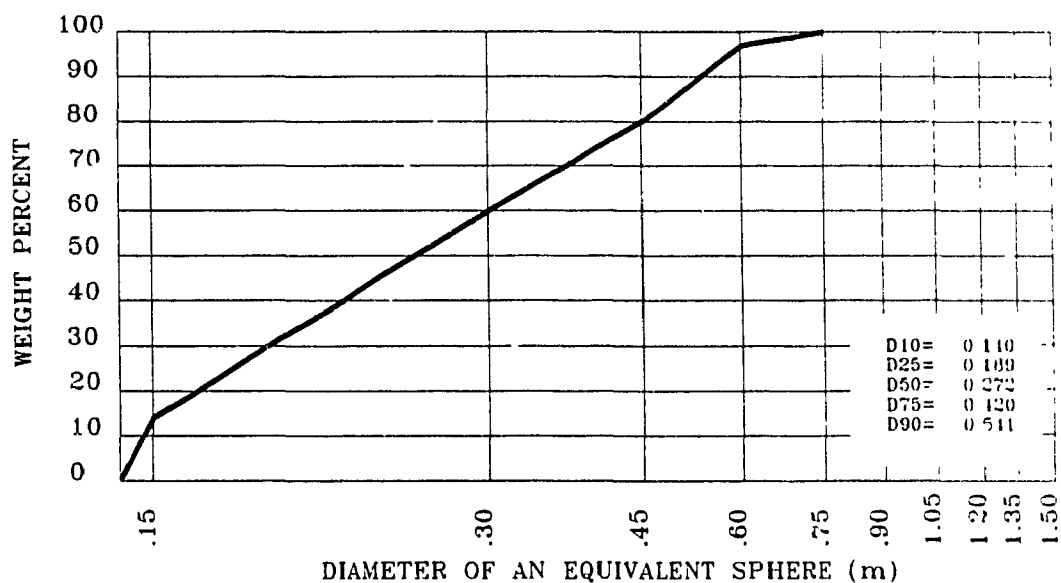


Figure 9 - Cumulative weight percent passing - August 24th.

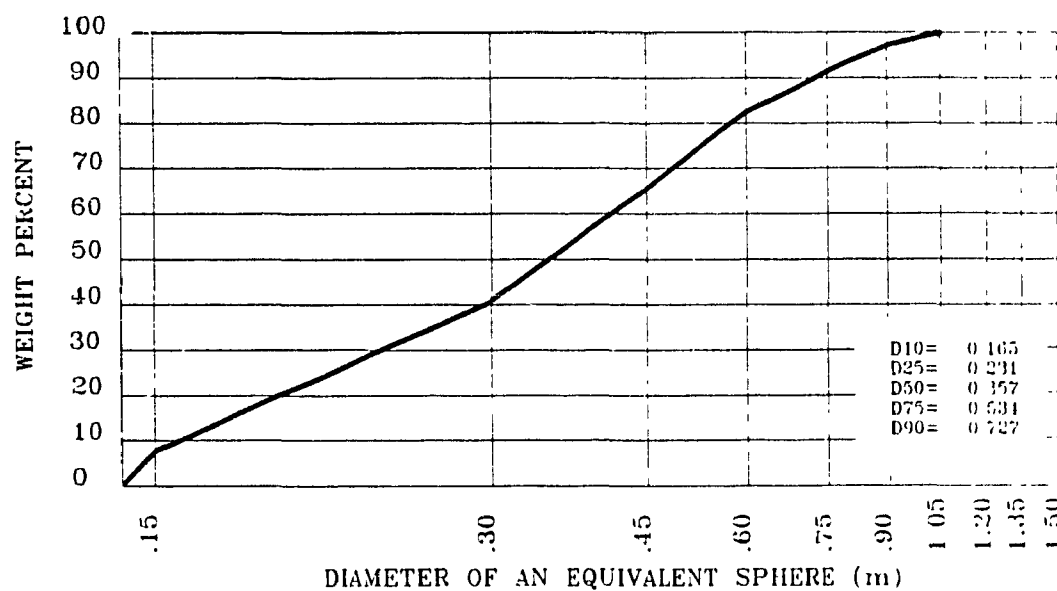


Figure 10 - Cumulative weight percent passing - August 25th.

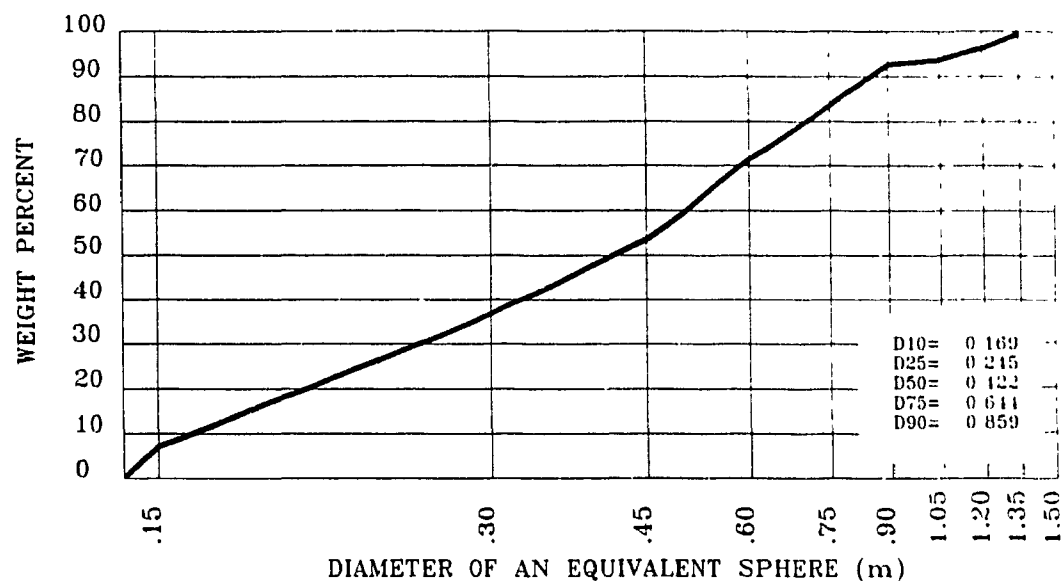


Figure 11 - Cumulative weight percent passing - October 8th.

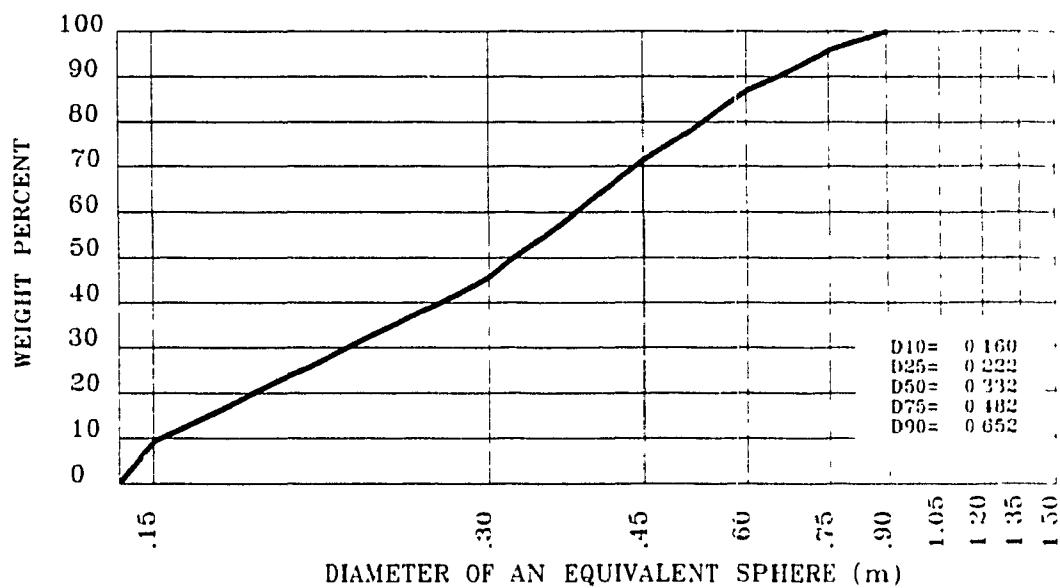


Figure 12 - Cumulative weight percent passing - October 9th.

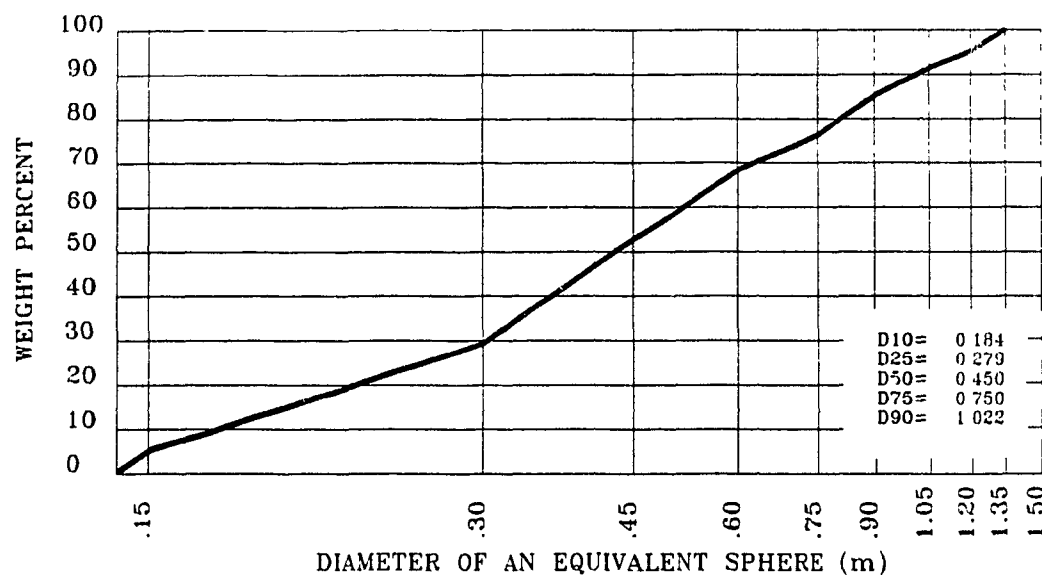


Figure 13 - Cumulative weight percent passing - October 11th.

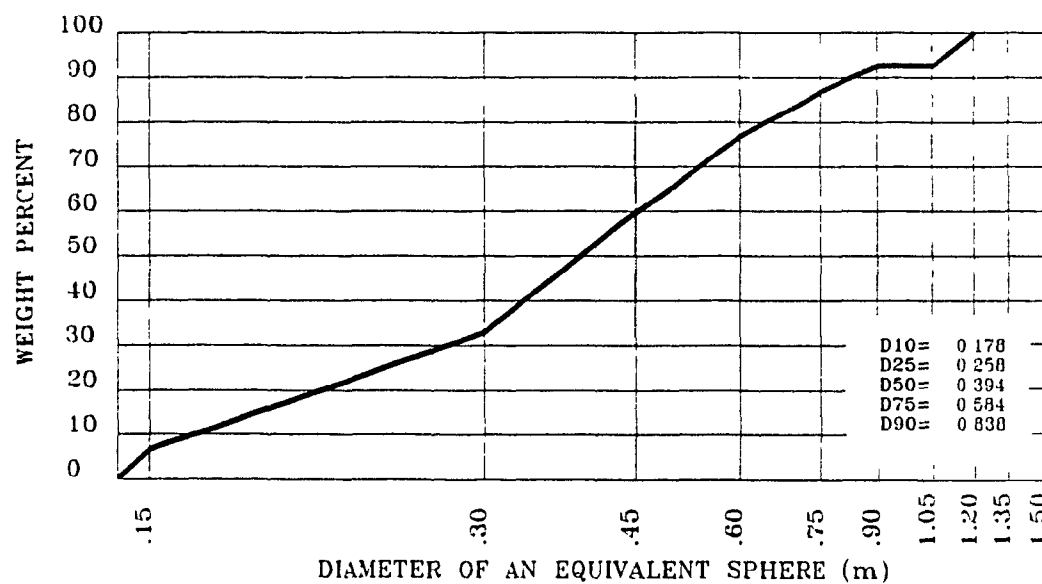


Figure 14 - Cumulative weight percent passing - October 12th.

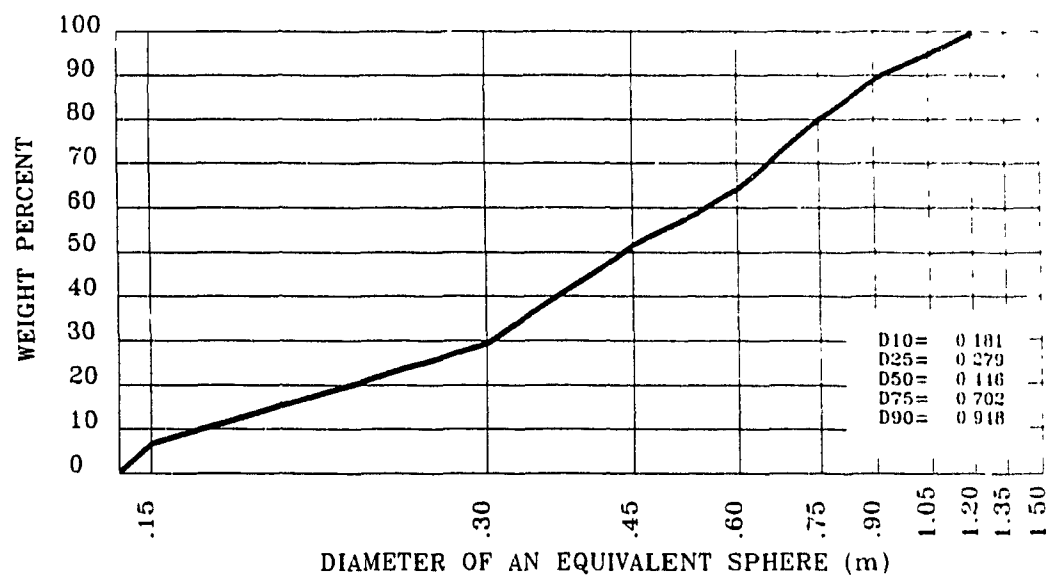


Figure 15 - Cumulative weight percent passing - October 14th.

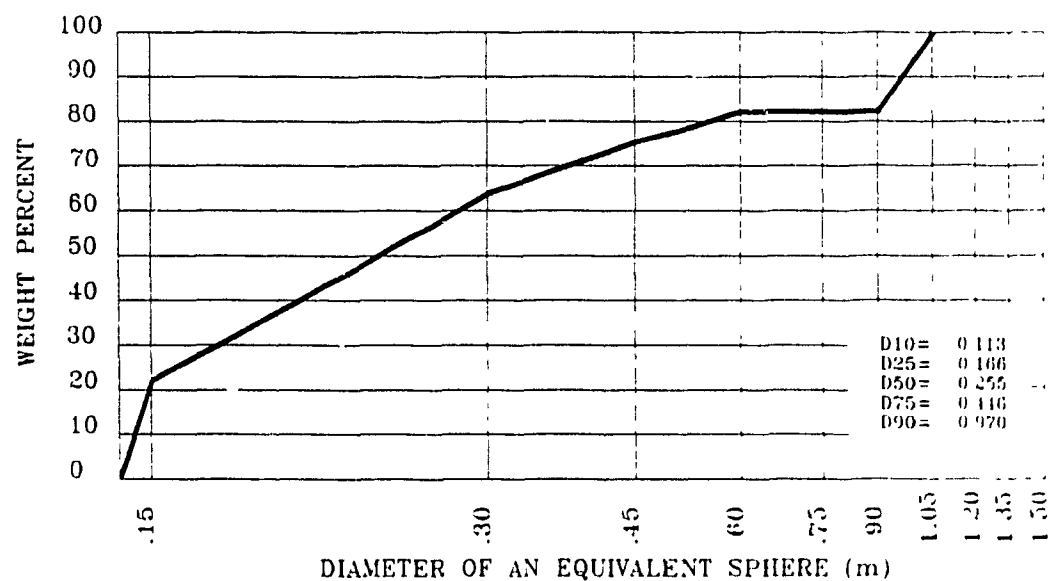


Figure 16 - Cumulative weight percent passing - October 26th.

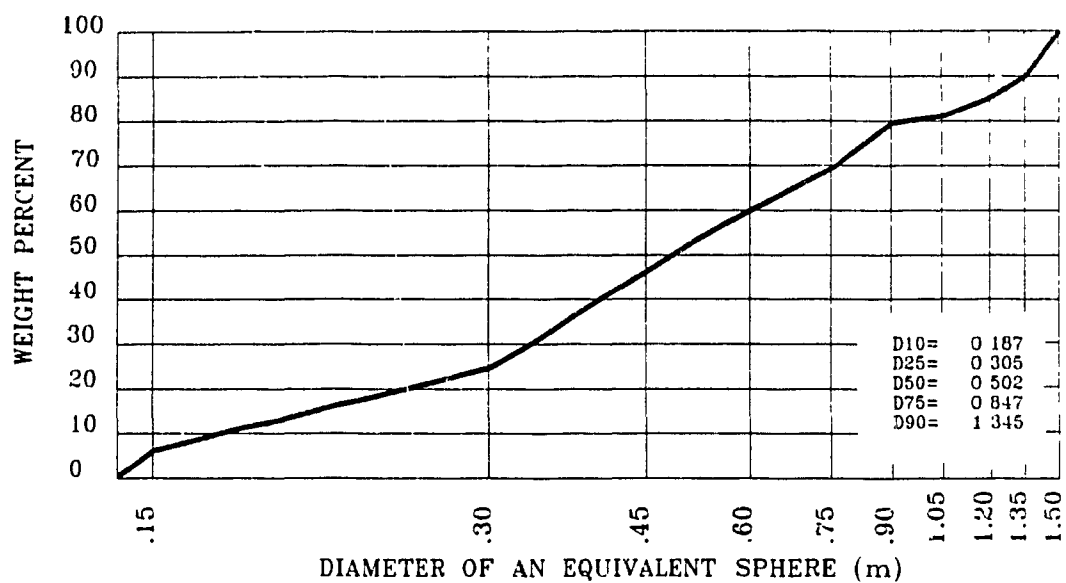


Figure 17 - Cumulative weight percent passing - October 27th.

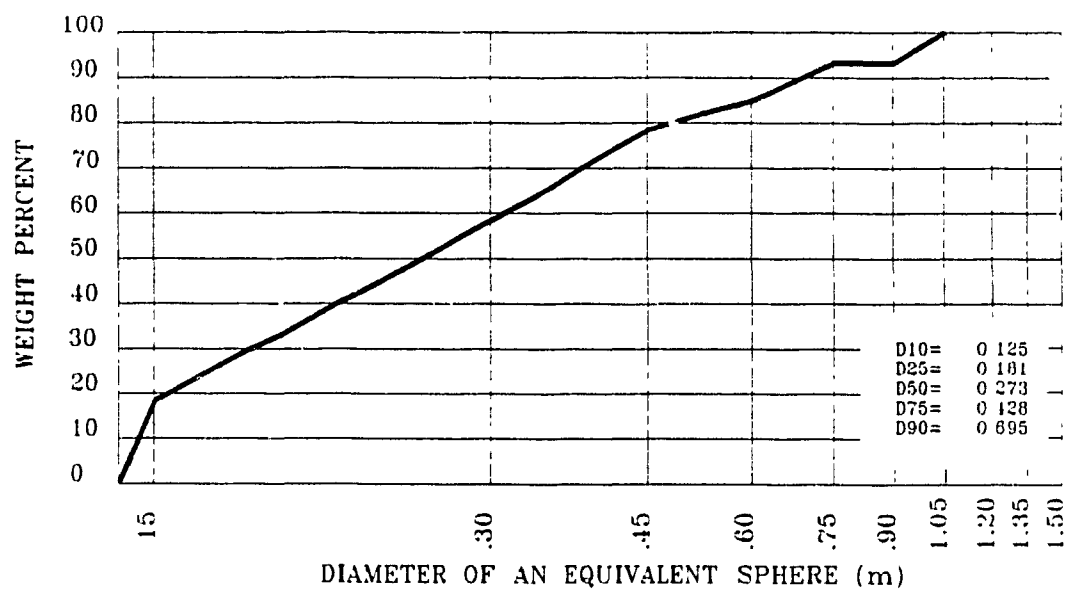


Figure 18 - Cumulative weight percent passing - October 28th.

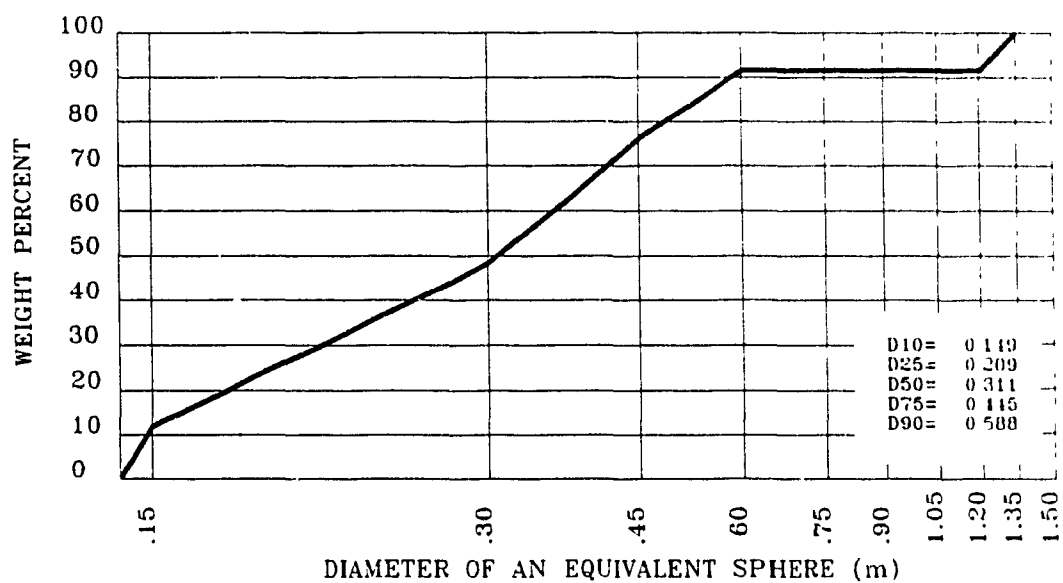


Figure 19 - Cumulative weight percent passing - October 29th.

APPENDIX - F

SUMMARY OF SHOVEL MONITORING STUDIES

AUGUST and OCTOBER 1988.

SUMMARY OF MONITORED STUDIES

<u>DATE</u>	<u>OPERATOR</u>	<u>G.E. MONITOR</u>	<u>POLYCODER</u>	<u>START TIME</u>	<u>PHOTOS</u>	<u>COMMENTS ON PHOTOS</u>	<u>VIDEO COUNTER</u>
AUG. 15	5	A		11:43	X	COARSE RILL	
AUG. 15	5	B		11:55	XX	AVG. FRAGMENTATION	
AUG. 15	5	C		12:11			
AUG. 16		GENERAL ELECTRIC MONITOR BEING RECALIBRATED					
AUG. 17	1	A	A	11:51	X	RILL	970
AUG. 17	1	B		12:09			1800
AUG. 17	1	C		12:58			3380
AUG. 17	1	D		13:07			3620
AUG. 17	1	D	B	13:24	X	AVG. FRAGMENTATION	4060
AUG. 17	1	D	C	13:49	X	RILL	
AUG. 17	1	E		13:57	X	RILL	
AUG. 18	2	A	A	11:56	XX	COARSE RILL	1400
AUG. 18	2	B		12:08	X	AVG. FRAGMENTATION	1880
AUG. 18	2	C	B	12:58	X	COARSE RILL	3560
AUG. 18	2	D		13:21	XXX	FACE + FRAGMENTATION	4100
AUG. 18	2	E		13:54	X	COARSE RILL	0
AUG. 18	2		C	14:02			565
AUG. 19	2		A	12:06	XX	FINE FRAGMENTATION	0
AUG. 19	2	A		12:13			880
AUG. 19	2	Ax	B	12:54	X	AVG. FRAGMENTATION	2500
AUG. 19	2	B	C	13:21	XX	FACE	3290
AUG. 19	2	C		13:41			3870
AUG. 19	2		D	14:02	XX	COARSE RILL	4095
AUG. 19	2	D	E	?	XX	WELL SORTED RILL	4308
AUG. 20	3	A	A	12:08	XXX	AVG. FRAGMENTATION	665
AUG. 20	3		B	12:31	X	AVG. RILL	1775
AUG. 20	3	B	C	12:55	XX	AVG. FRAGMENTATION	2585
AUG. 20	3	C	D	13:24	XX	RILL	3480
AUG. 20	3	D		14:18	XX	AVG. FRAGMENTATION + RILL	3860
AUG. 20	3	E	E	14:30	X	RILL	4020

SUMMARY OF MONITORED STUDIES

<u>DATE</u>	<u>OPERATOR</u>	<u>G.E. MONITOR</u>	<u>POLYCODER</u>	<u>START TIME</u>	<u>PHOTOS</u>	<u>COMMENTS ON PHOTOS</u>	<u>VIDEO COUNTER</u>
AUG. 21	3	A	A	12:11	X	RILL	740
AUG. 21	3	B	B	12:46			2210
AUG. 21	3	C		13:00	X	RILL	2690
AUG. 21	3	D	C	13:14	X	COARSE RILL	3300
AUG. 21	3		D	13:45	XX	COARSE RILL + FACE	3660
AUG. 21	3	E	E	14:14	XX	AVG. FRAGMENTATION	3981
AUG. 22	4		A	11:53	XX	AVG. FRAGMENTATION	0
AUG. 22	4	A	B	12:16			1260
AUG. 22	4	B	C	12:45	XX	AVG. FRAGMENTATION + RILL	2400
AUG. 22	4	C	D	13:12	X	AVG. FRAGMENTATION	2880
AUG. 22	4	D	E	13:36	X	AVG. FRAGMENTATION	3570
AUG. 22	4	E	F	14:06	XXX	RILL + OVERSIZE	4130
AUG. 22	4		G	14:32	X	AVG. FRAGMENTATION	
AUG. 23	4	A	A	12:04	XX	FINE FRAGMENTATION	435
AUG. 23	4		B	12:30	X	RILL	1600
AUG. 23	4	B		12:44	X	AVG	2190
AUG. 23	4	C	C	13:03	XX	RILL	2860
AUG. 23	4	D	D	13:36	XX	RILL + FINE FRAGMT.	3685
AUG. 23	4	E	E	14:13	XX	FINE FRAGMENTATION	3968
AUG. 23	4		F	14:39			4243
AUG. 24	1	A	A	11:29	X	RILL	230
AUG. 24	1		B	11:48	XX	AVG. FRAGMENTATION + RILL	1690
AUG. 24	1	B		12:15			2315
AUG. 24	1	C	C	12:40	XX	AVG. FRAGMENTATION + RILL	2790
AUG. 24	1	D	D	13:10	XXXX	FACE + AVG. FRAGMT.	3200
AUG. 24	1	E	E	13:40			3950
AUG. 24	1	F	F	14:01			4113
AUG. 24	1		G	14:27			
AUG. 25	1	A	A	12:05	X	COARSE RILL	420
AUG. 25	1		B	12:44	XXX	BLOCKY FRAGMENTATION	1615
AUG. 25	1	B		13:02	XX	BLOCKY FRAGMENTATION	2283
AUG. 25	1	C	C	13:17			2800
AUG. 25	1	D	D	13:43			

SUMMARY OF MONITORED STUDIES

<u>DATE</u>	<u>OPERATOR</u>	<u>G.E. MONITOR</u>	<u>POLYCODER</u>	<u>START TIME</u>	<u>PHOTOS</u>	<u>COMMENTS ON PHOTOS</u>	<u>VIDEO COUNTER</u>
AUG. 25	1	E	E	14:08			3275
AUG. 25	1	F	F	14:35	X	AVG. FRAGMENTATION	3735

SECOND BENCH STUDY WITH POLYCODER ALONE - BLAST EM #4 (SQUARE PATTERN)

OCT. 3	3		A	11:45	XX	COARSE RILL + AVG FRAGMT.	25
OCT. 8	3		B	12:13	XXX	COARSE RILL + AVG. FRAGMT	2027
OCT. 8	3		C	12:39	X	FINER FRAGMENTATION	3308
OCT. 8	3		D	13:09	XXX	CLAENING BLAST THROW	4444
OCT. 8	3		E	13:33			5270
OCT. 8	3		F	14:11	XX	AVG. FRAGMENTATION	
OCT. 9	4		A	12:14	X	THROW	21
OCT. 9	4		B	12:38	XXX	COARSE RILL	FAILED
OCT. 9	4		C	13:04	XX	AVG. FRAGMENTATION	2790
OCT. 9	4		D	13:30	XX	COARSE FRAGMENTATION	3729
OCT. 9	4		E	13:58	XXXXXX	COARSE FRAGMT. + FACE	4725
OCT. 9	4		F	14:47	XX	COARSE FRAGMENTATION	5500
OCT. 10	1		A	12:01	XX	AVG. FRAGMENTATION	15
OCT. 10	1		B	12:26		STALLED IN FACE	1668
OCT. 10	1		C	12:49			2865
OCT. 10	1		D	13:26	XX	FACE	3692
OCT. 10	1		E	13:55	X	AVG. FRAGMENTATION	4750
OCT. 10	1		F	14:20	XX	FACE	5439
OCT. 12	1		A	11:46	X	AVG. FRAGMENTATION	25
OCT. 12	1		B	12:10			650
OCT. 12	1		C	12:40	XXXXX	AVG. FRAGMENTATION + RILL	2255
OCT. 12	1		D	13:14			3537
OCT. 12	1		E	13:56	XXX	FACE	4392
OCT. 12	1		F	14:21	X	COARSER RILL	
OCT. 14	6		A	11:54	XX	AVG. FRAGMENTATION	100
OCT. 14	6		B	12:19	XX	RILL + FACE	1950
OCT. 14	6		C	12:43	X	PILL + FACE	3145

SUMMARY OF MONITORED STUDIES

<u>DATE</u>	<u>OPERATOR</u>	<u>G.E. MONITOR</u>	<u>POLYCODER</u>	<u>START TIME</u>	<u>PHOTOS</u>	<u>COMMENTS ON PHOTOS</u>	<u>VIDEO COUNTER</u>
OCT. 14	6		D	13:09			4218
OCT. 14	6		E	13:36	X	COARSE RILL	5173
OCT. 14	6		F	14:23	XX	AVG. FRAGMENTATION	5560
OCT. 15th TO 25th; #15 SHOVEL DOWN FOR REPLACEMENT OF BOTH FRONT AND REAR SWING TRANSMISSIONS							
OCT. 26	4		A	11:41	XXXXX	AVG. FRAGMENTATION	23
OCT. 26	4		B	12:10	XX	WELL SORTED SMALL RILL	2079
OCT. 26	4		C	12:44	X	FINER FRAGMENTATION	2766
OCT. 26	4		D	13:40			3869
OCT. 26	4		E	14:06	XX	FRAGMENTATION + FACE	4850
OCT. 26	4		F	14:34			5768
OCT. 27	1		A	11:39			92
OCT. 27	1		B	12:09	XXX	RILL	2600
OCT. 27	1		C	12:39	XX	RILL + FACE	3575
OCT. 27	1		D	13:08	XX	RILL	4718
OCT. 27	1		E	13:33	X	RILL	5563
OCT. 27	1		F	14:08			
OCT. 28	1		A	12:03	XXX	FINE FRAGMENTATION	250
OCT. 28	1		B	12:30	XX	SMALL RILL	2060
OCT. 28	1		C	12:57	XX	FINE AVG. FRAGMENTATION	3382
OCT. 28	1		D	13:24			4333
OCT. 28	1		E	13:50	XXX	RILL + FACE	5275
OCT. 28	1		F	14:16	XX	AVG. FRAGMENTATION	
OCT. 29	2		A	11:51	XXX	RILL	31
OCT. 29	2		B	12:18	XXX	RILL + FACE	1950
OCT. 29	2		C	12:49	XX	AVG. FRAGMENTATION	3480
OCT. 29	2		D	13:18			4653
OCT. 29	2		E	13:45			5950
OCT. 29	2		F	14:27	XXXXXXXX	AVG. FRAGMT., RILL + FACE	7863

APPENDIX - G

EXAMPLE RECORDS OF MONITORED SHOVEL PERFORMANCE PARAMATERS.

G.E. MONITOR AUG15C
DATA-LOGGER

OPERATOR: S PAGE: 1
TIME: 1241

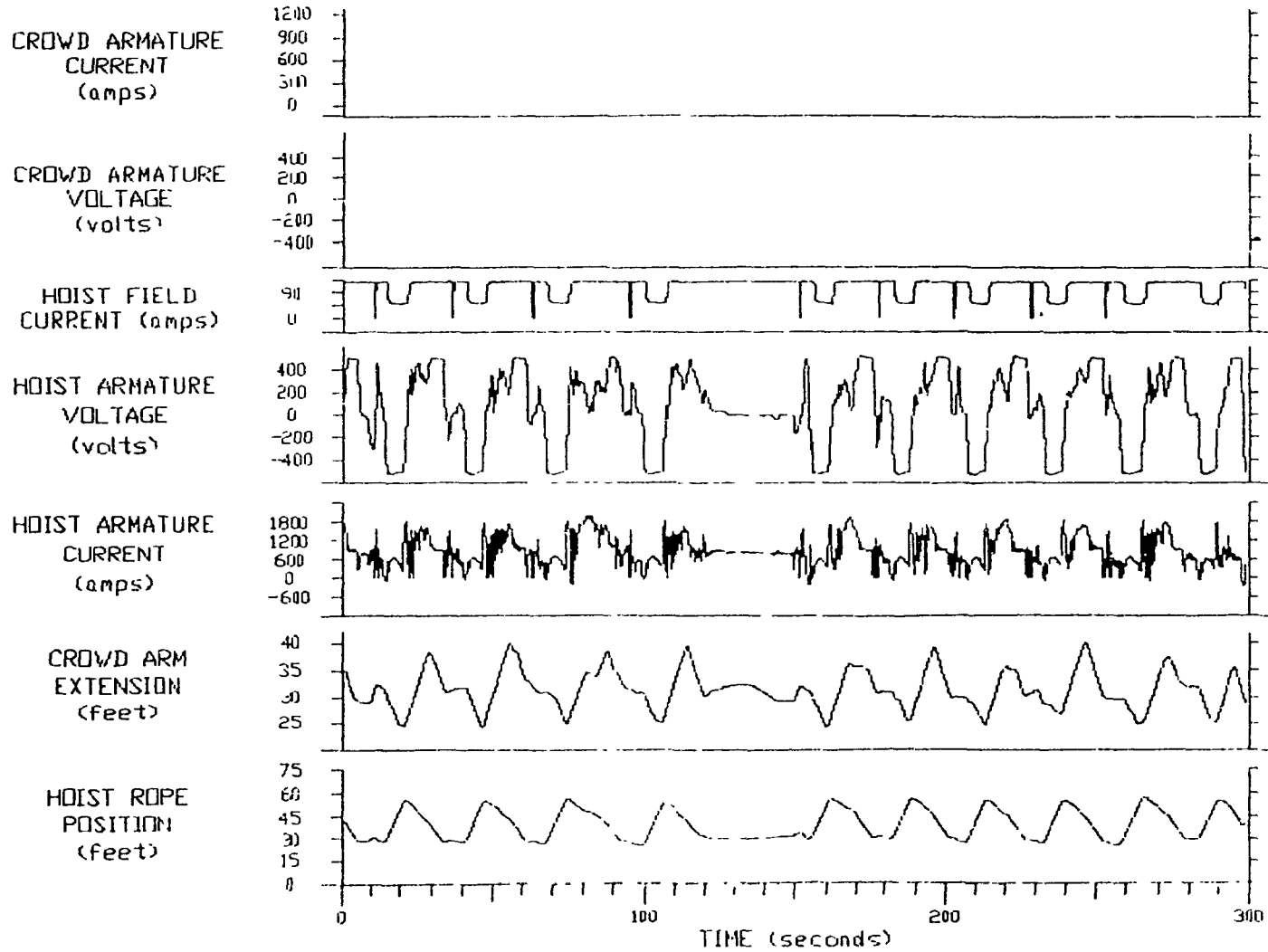


Figure 1 - Example records for August 15th.

G.E. MONITOR: AUG17A OPERATOR: 1 PAGE: 1
DATA-LOGGER: PAUG17A TIME: 11:51

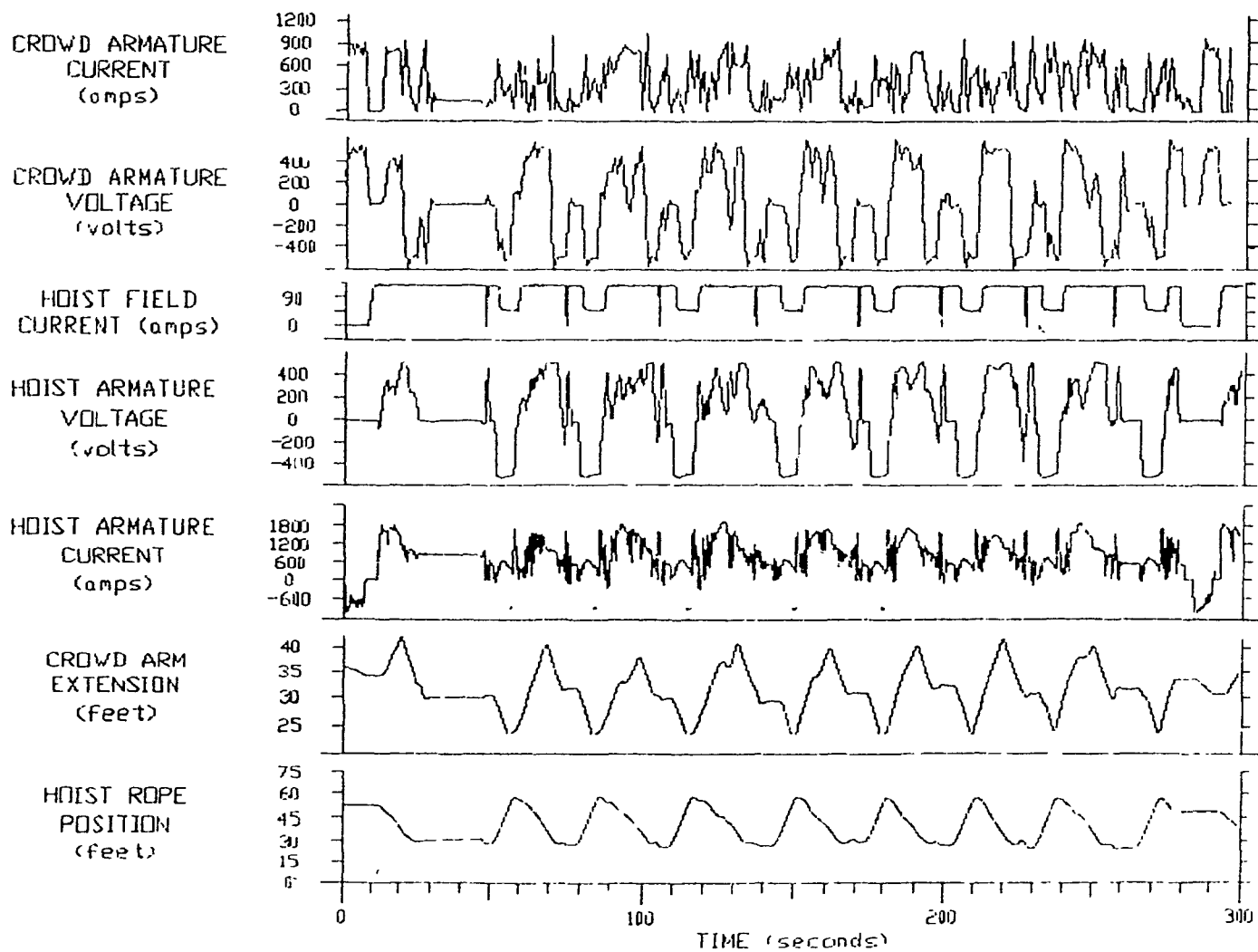


Figure 2 - Example records for August 17th.

G.E. MONITOR: AUG18D
DATA-LOGGER:

OPERATOR: 2 PAGE. 2
TIME: 1321

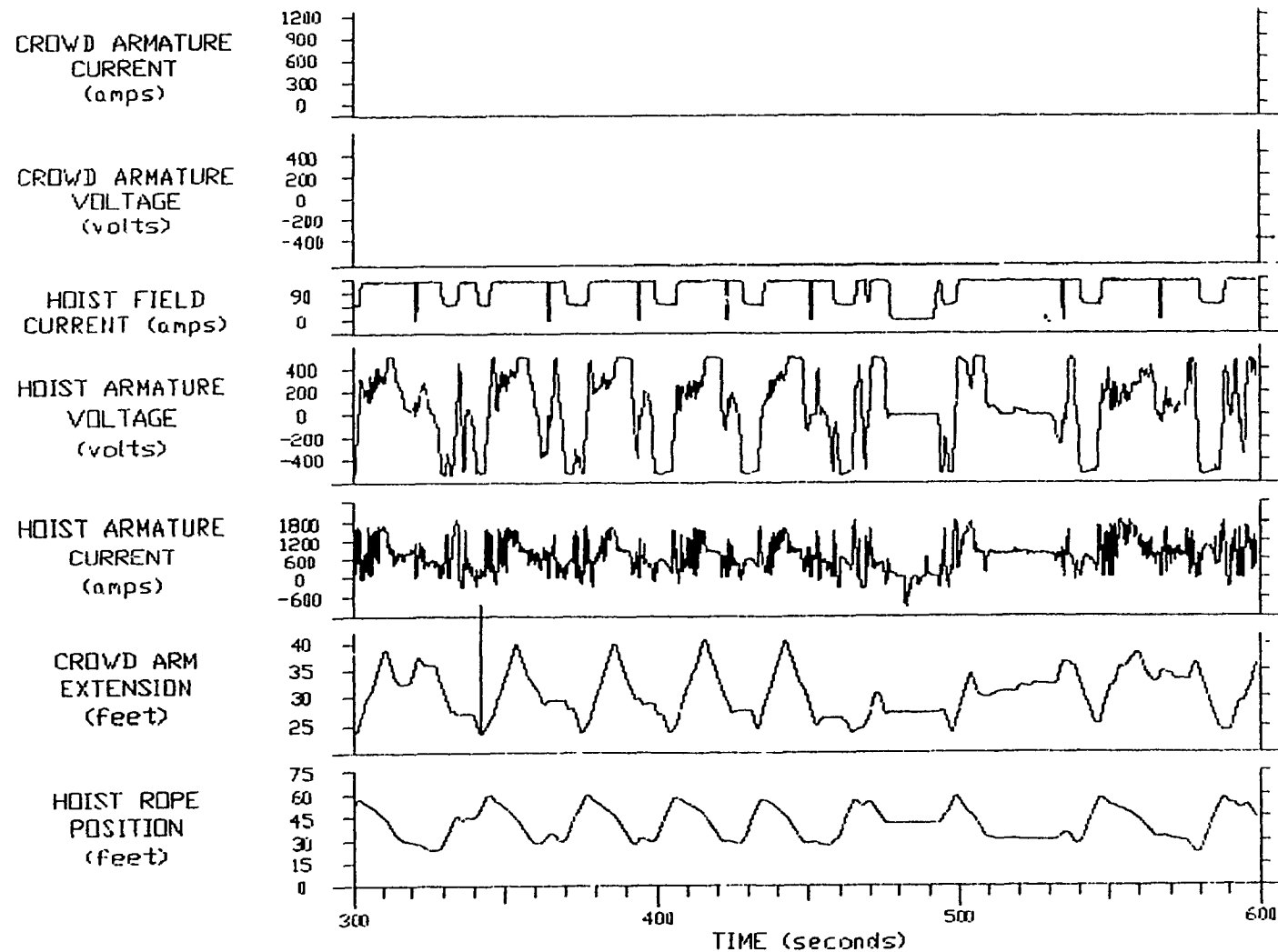


Figure 3 - Example records for August 18th.

G.E. MONITOR: AUG19B OPERATOR: 2 PAGE: 1
 DATA-LOGGER: PAUG19C TIME: 13:21

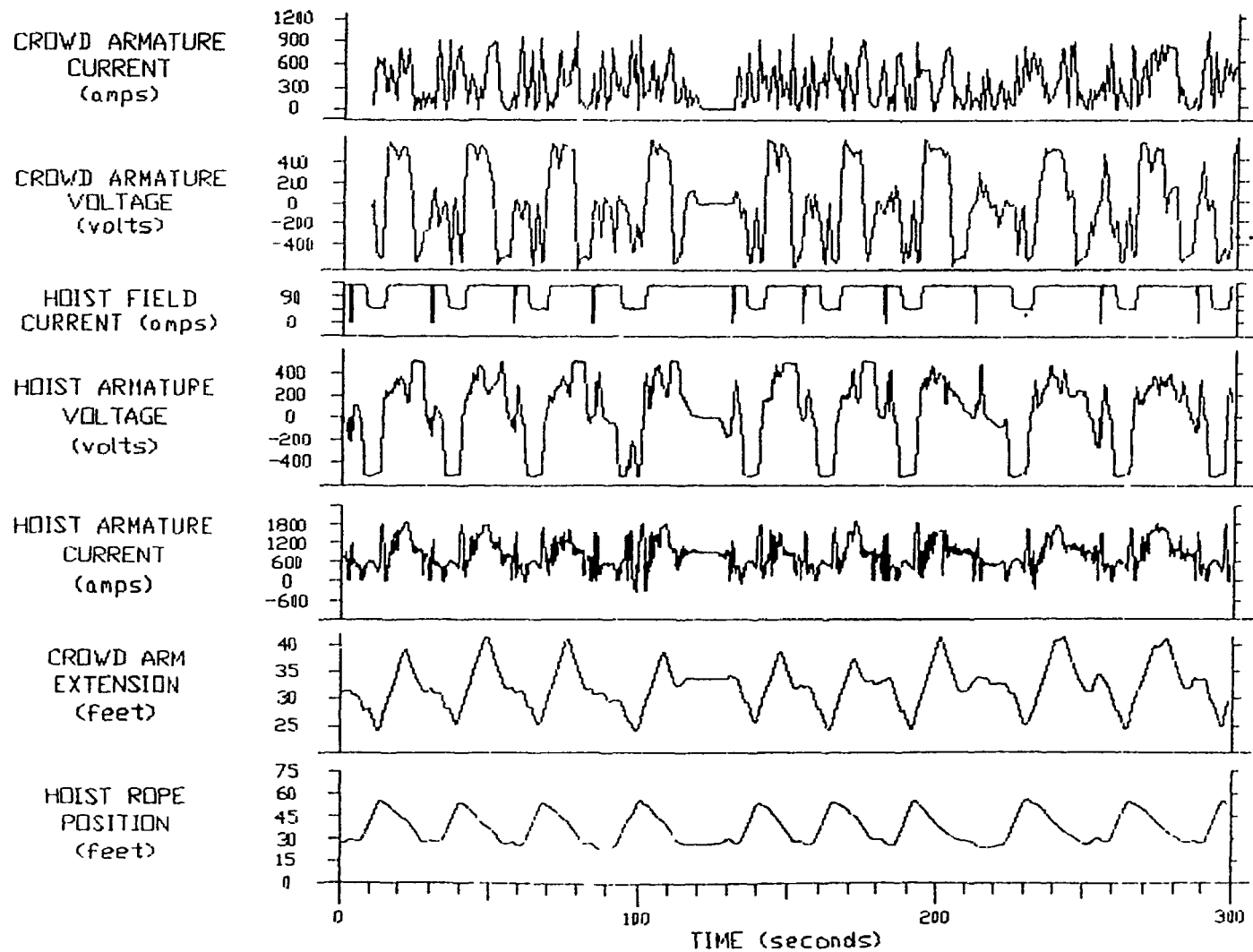


Figure 4 - Example records for August 19th.

G.E. MONITOR: AUG20B OPERATOR: 3 PAGE: 1
DATA-LOGGER: PAUG20C TIME: 12:55

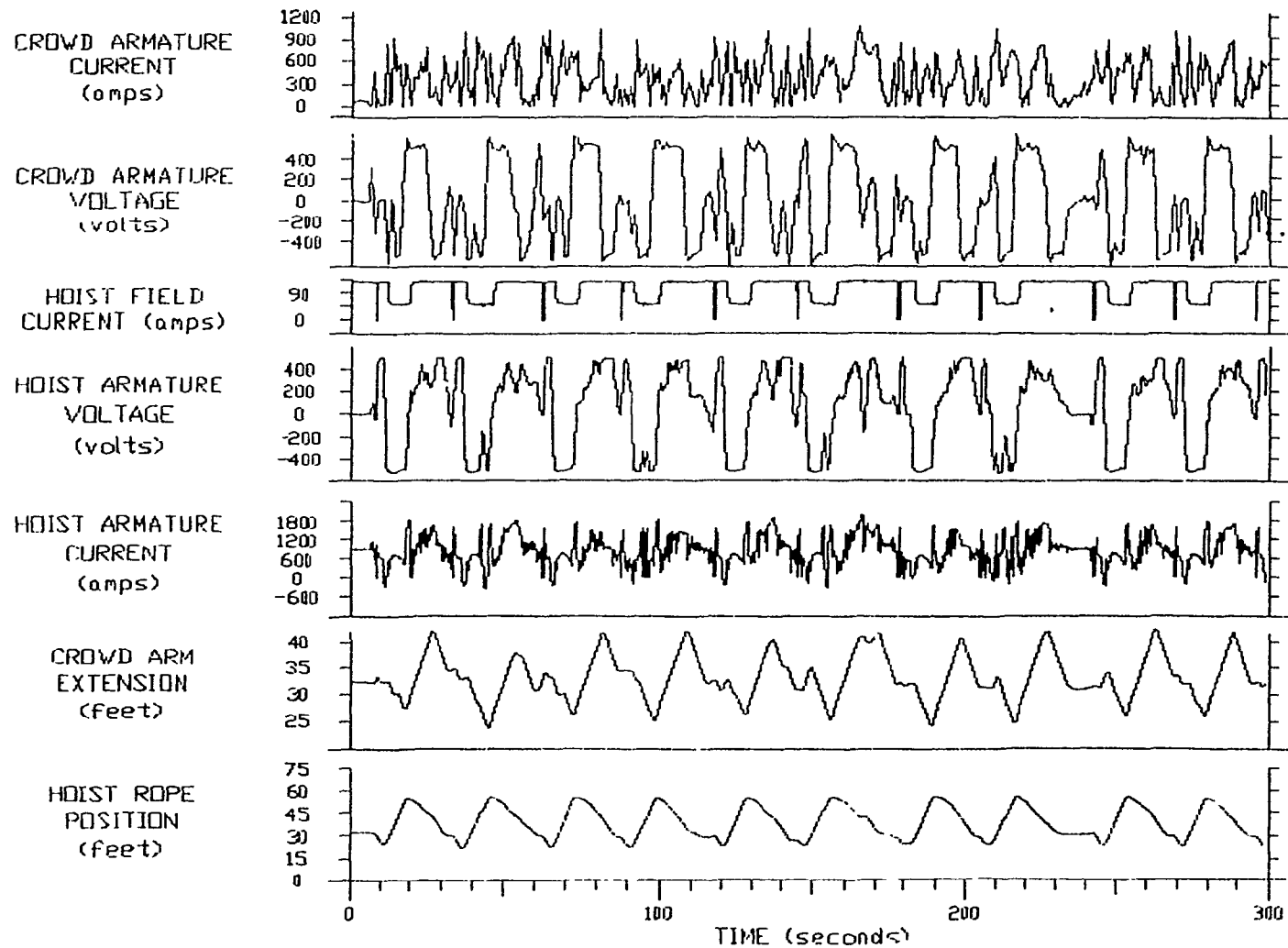


Figure 5 - Example records for August 20th.

G.E. MONITOR: AUG21C
DATA-LOGGER

OPERATOR: 3 PAGE: 1
TIME: 13:00

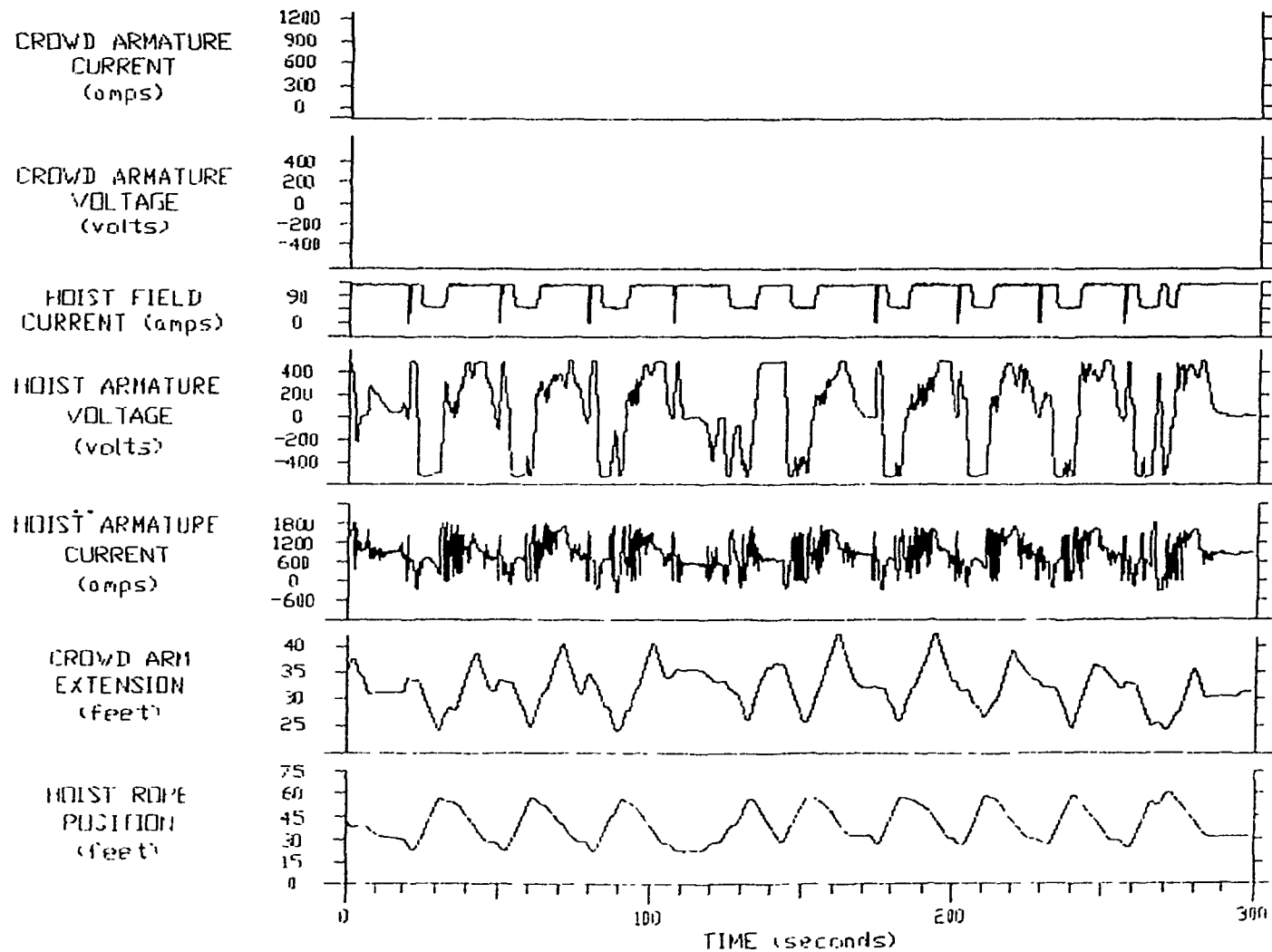


Figure 6 - Example records for August 21st.

G.E. MONITOR: AUG22A OPERATOR: 4 PAGE. 2
DATA-LOGGER PAUG22B TIME 12:16

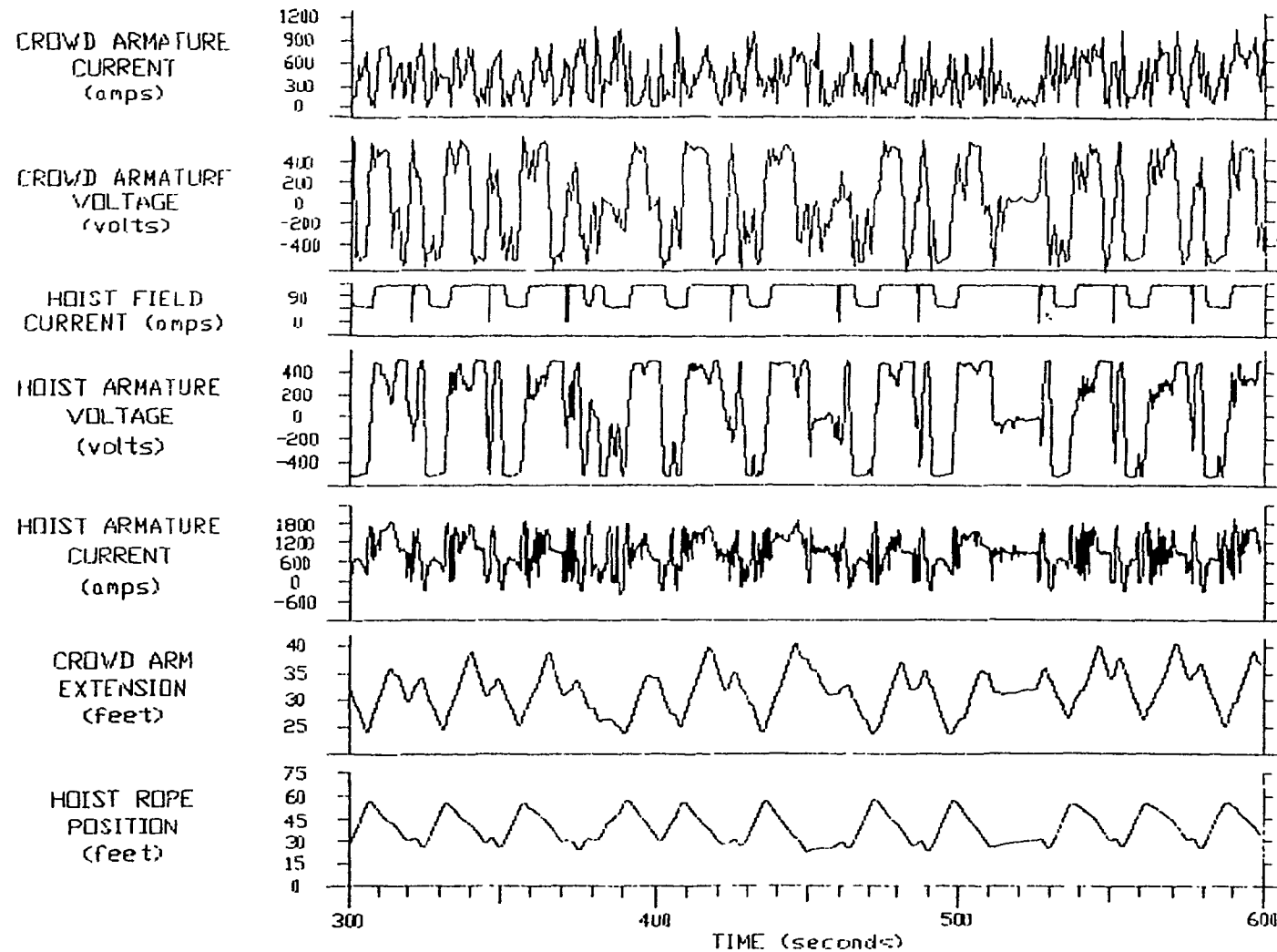


Figure 7 - Example records for August 22nd.

G.E. MONITOR: AUG23D OPERATOR: 4 PAGE: 1
DATA-LOGGER: PAUG23D TIME: 13:36

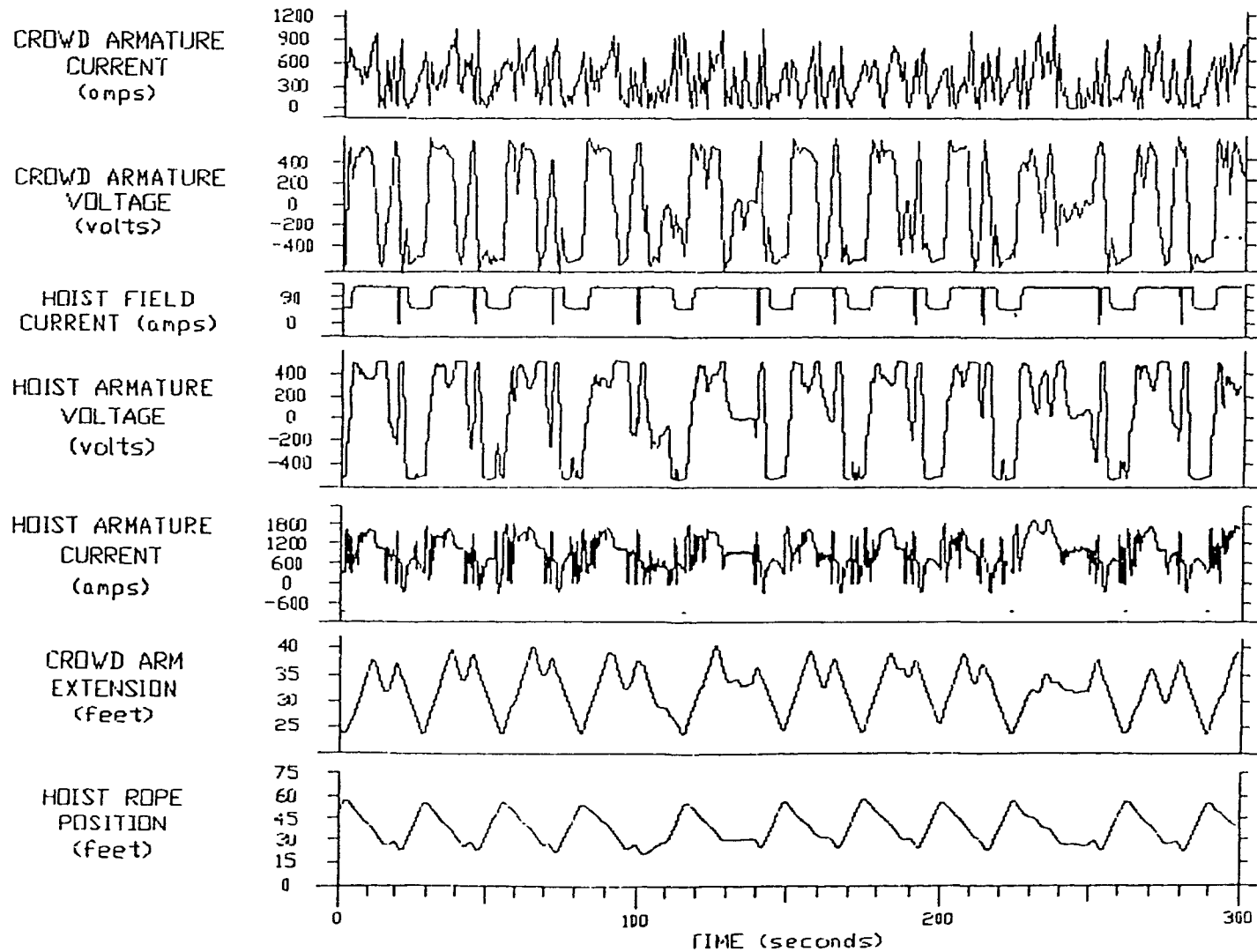


Figure 8 - Example records for August 23rd.

GF MONITOR AUG24C OPERATOR: 1 PAGE: 1
 DATA-LOGGER: PAUG24C TIME: 12:40

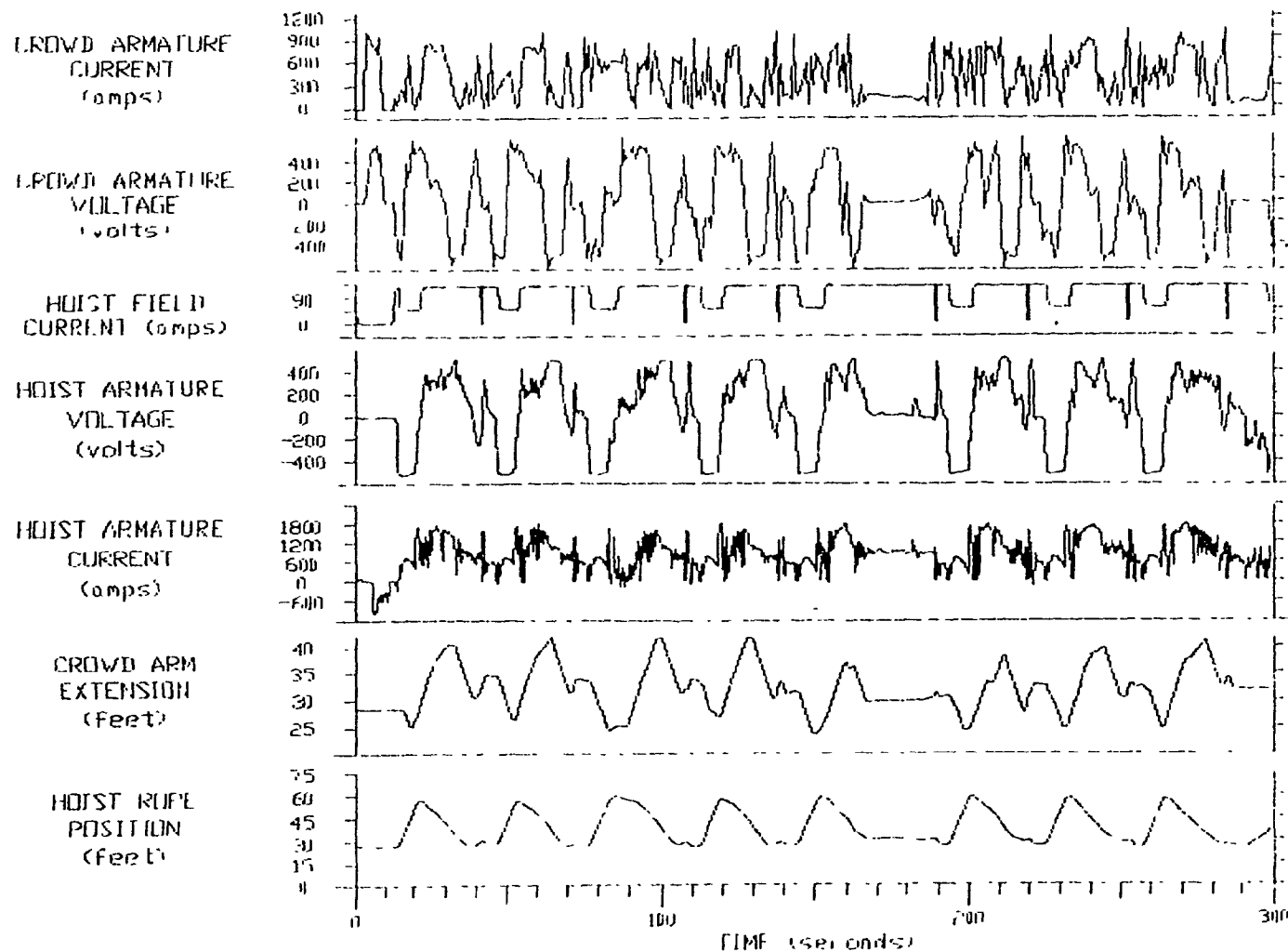


Figure 9 - Example records for August 24th.

G: MONITOR AUG25 OPERATOR: 1 PAGE: 1
 DATA LOGGER: PAUL251 TIME: 1435

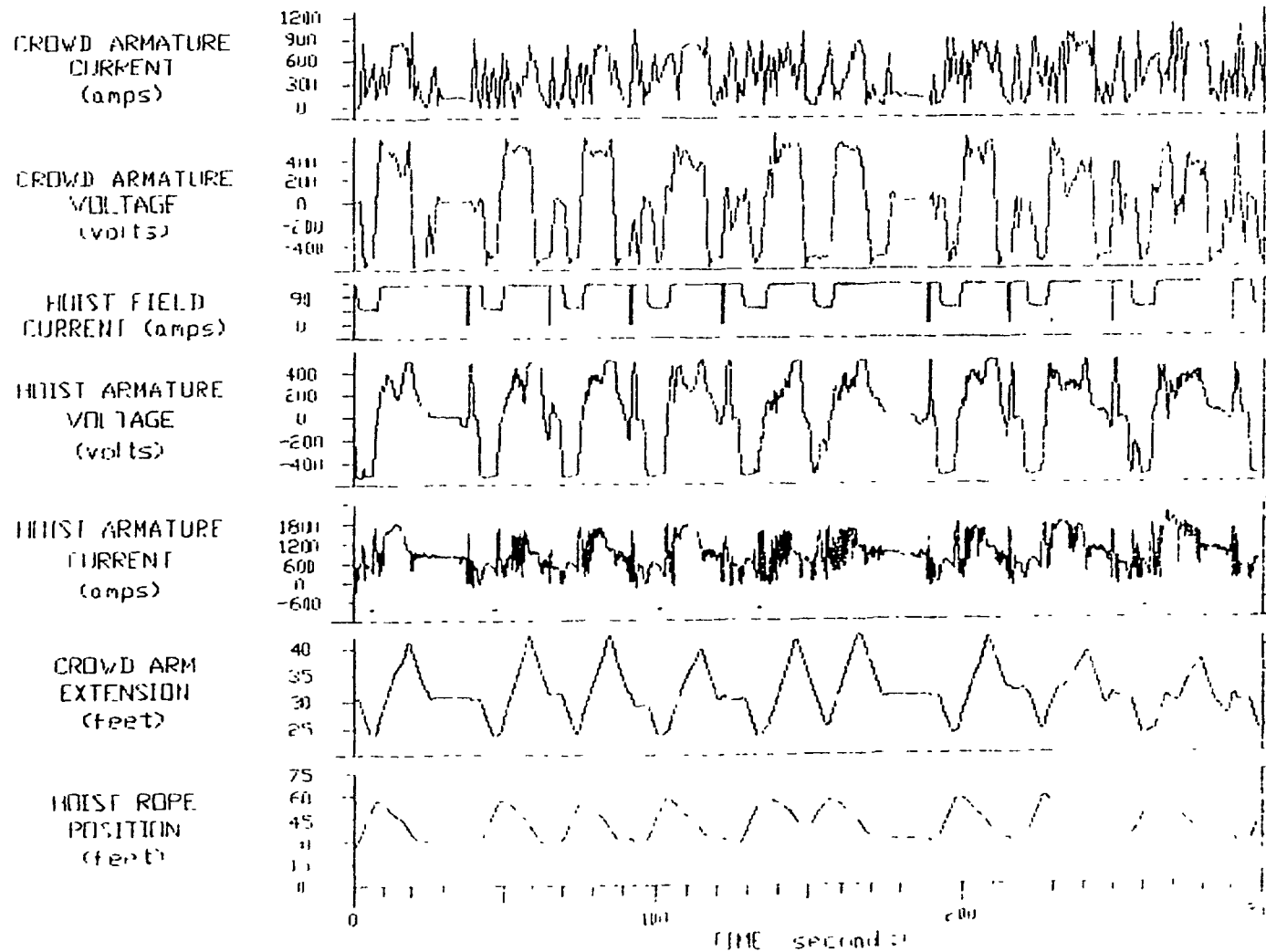


Figure 10 - Example records for August 25th.

APPENDIX - H

SUMMARY CYCLE TIME VALUES OF TIME STUDY

OPERATOR	DATE	STUDY TIME min)	CYCLE TIME (sec)	CYCLE TIME STD.	SWING BANK (sec)	SWING BANK STD.	DIG TIME (sec)	DIG TIME STD.	SWING TRUCK (SEC)
1	AUG17	111.57	40.08	21.21	9.93	2.04	13.86	3.25	7.05
2	AUG18	122.81	37.59	19.16	9.70	1.93	11.73	2.27	6.91
2	AUG19	126.99	35.94	20.62	9.90	1.98	10.66	2.11	7.11
3	AUG20	120.00	31.44	6.78	8.82	1.87	11.33	1.78	6.24
3	AUG21	113.89	35.41	14.76	9.10	1.86	11.92	2.91	6.76
4	AUG22	90.90	34.30	18.40	8.52	1.61	11.03	1.87	7.43
4	AUG23	127.29	32.92	14.75	8.77	1.89	10.89	1.49	6.88
1	AUG24	121.49	41.65	27.98	10.65	2.84	12.49	2.43	7.73
1	AUG25	120.19	43.70	28.94	10.02	2.52	13.35	3.64	7.72
3	OCT8	129.95	36.10	24.22	9.64	3.45	11.68	2.94	6.40
4	OCT9	123.68	37.48	23.43	9.35	1.74	11.80	2.27	6.82
1	OCT11	121.47	35.90	14.69	10.01	2.59	12.58	4.05	6.66
1	OCT12	120.85	37.57	19.45	10.18	2.64	11.95	2.87	6.40
6	OCT14	130.09	40.65	25.42	10.70	2.47	13.69	3.39	7.80
4	OCT26	126.74	34.56	23.36	9.17	1.93	10.96	1.77	6.45
1	OCT27	128.84	35.96	13.21	10.24	2.65	12.26	2.52	6.73
1	OCT28	125.31	39.99	19.06	10.68	2.61	13.40	2.67	6.84
2	OCT29	150.50	37.16	16.28	11.04	3.82	10.85	1.70	7.21

OPERATOR	DATE	SWING TRUCK (SEC)	SWING TRUCK STD.	DUMP TIME (sec)	DUMP TIME STD.	AVERAGE DELAY (sec)	AVERAGE PROPEL (sec)	No. OF DELAYS	No. OF PROPELS	TOTAL DELAYS (min)	TOTAL PROPEL (min)
1	AUG17	7.05	2.11	3.31	2.06	29.47	53.95	30.00	2.00	14.73	1.00
2	AUG18	6.91	1.96	3.33	1.34	26.86	28.74	39.00	4.00	17.46	1.92
2	AUG19	7.11	1.99	3.08	0.70	26.46	45.48	33.00	5.00	14.55	3.79
3	AUG20	6.24	2.06	3.30	0.62	11.34	16.20	34.00	1.00	6.43	0.27
3	AUG21	6.76	2.34	3.36	0.96	20.96	33.76	36.00	2.00	12.58	1.13
4	AUG22	7.43	2.83	3.09	0.52	18.30	42.40	32.00	2.00	9.80	1.41
4	AUG23	6.88	2.43	3.05	0.64	16.29	34.51	39.00	4.00	10.59	2.30
1	AUG24	7.73	2.43	3.14	1.31	39.36	39.28	31.00	3.00	20.34	1.96
1	AUG25	7.72	2.23	3.08	0.64	39.66	16.59	38.00	4.00	25.12	1.11
3	OCT8	6.40	2.11	3.09	0.63	18.04	76.53	42.00	5.00	12.63	6.38
4	OCT9	6.82	4.16	3.03	1.04	30.13	33.24	36.00	6.00	18.08	3.32
1	OCT11	6.66	2.02	3.11	0.89	21.13	27.91	33.00	2.00	11.07	0.93
1	OCT12	6.40	1.30	2.98	0.47	31.31	24.76	35.00	3.00	18.26	1.24
6	OCT14	7.80	2.34	3.36	1.14	31.31	33.19	27.00	4.00	14.09	2.21
4	OCT26	6.45	2.23	2.87	0.49	30.57	42.10	34.00	2.00	17.33	1.40
1	OCT27	6.73	1.76	2.97	0.54	16.81	17.07	41.00	7.00	11.49	1.99
1	OCT28	6.84	1.62	3.07	0.61	24.83	20.02	39.00	8.00	16.14	2.67
2	OCT29	7.21	2.20	3.26	0.86	23.76	19.52	45.00	5.00	17.82	1.63

PERCENT OF TOTAL CYCLE TIME ENGAGED IN:

OPERATOR	DATE	% OF CYCLE SWING BANK	% OF CYCLE DIGGING	% OF CYCLE SWING TRUCK	% OF CYCLE DUMPING	% OF CYCLE DELAYS	% OF CYCLE PROPELLING
1	AUG17	24.78	34.57	17.58	8.25	13.21	1.61
2	AUG18	25.80	31.20	18.37	8.06	14.22	1.56
2	AUG19	27.55	29.66	19.77	8.57	11.46	2.98
3	AUG20	28.05	36.04	19.84	10.49	5.35	0.23
3	AUG21	25.71	33.67	19.08	9.50	11.04	0.99
4	AUG22	24.83	32.16	21.65	9.02	10.78	1.55
4	AUG23	26.63	33.07	20.91	9.27	8.32	1.81
1	AUG24	25.57	29.99	18.56	7.53	16.74	1.62
1	AUG25	22.93	30.54	17.67	7.05	20.90	0.92
3	OCT8	26.70	32.37	17.74	8.57	9.72	4.91
4	OCT9	24.94	31.48	18.19	8.09	14.62	2.69
1	OCT11	27.87	35.03	18.56	8.67	9.11	0.77
1	OCT12	27.09	31.81	17.03	7.93	15.11	1.02
6	OCT14	26.32	33.68	19.19	8.28	10.83	1.70
4	OCT26	26.54	31.70	18.67	8.31	13.67	1.11
1	OCT27	28.48	34.08	18.71	8.26	8.92	1.55
1	OCT28	26.71	33.50	17.11	7.67	12.88	2.13
2	OCT29	29.70	29.19	19.41	8.77	11.84	1.08

LESS DELAYS AND PROPEL EPISODES

OPERATOR	DATE	% OF CYCLE SWING BANK	% OF CYCLE DIGGING	% OF CYCLE SWING TRUCK	% OF CYCLE DUMPING
1	AUG17	29.09	40.58	20.64	9.68
2	AUG18	30.64	37.04	21.81	10.52
2	AUG19	32.20	34.67	23.11	10.02
3	AUG20	29.71	38.17	21.01	11.11
3	AUG21	29.23	38.28	21.69	10.80
4	AUG22	28.33	36.69	24.70	10.29
4	AUG23	29.63	36.08	23.26	10.31
1	AUG24	31.31	36.73	22.73	9.22
1	AUG25	29.33	29.06	22.60	9.01
3	OCT8	31.27	37.91	20.78	10.04
4	OCT9	30.16	38.07	21.99	9.78
1	OCT11	30.93	38.87	20.59	9.62
1	OCT12	32.30	37.93	20.31	9.46
6	OCT14	30.09	38.51	21.94	9.46
4	OCT26	31.14	37.20	21.91	9.75
1	OCT27	31.81	38.07	20.90	9.23
1	OCT28	31.42	39.41	20.14	9.03
2	OCT29	34.11	33.53	22.29	10.08