A dragline simulation model for strip mine design and development

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A DRAGLINE SIMULATION MODEL FOR STRIP MINE DESIGN AND DEVELOPMENT

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IN THE NAME OF GOD

This thesis is dedicated to my dear *family*

and my dear *parents*

for their love and patience

AFFIRMATION

The work as presented in this thesis is an authentic record to the best of my own knowledge and belief and it is based on the work carried out in the Department of Civil and Mining Engineering at University of Wollongong. I hereby certify that this thesis contains no material which I have submitted, in whole or in part, for a degree at this or any other institution. The following publications have been based on this thesis:

Baafi, E.Y., **Mirabediny, H.** and Whitchurch, K., (1995), "*A Simulation Model for Selecting Suitable Digging Method for a Dragline Operation*", 25th International Symposium on the Application of Computers and Operations Research in the Mineral Industry, The Australasian Institute of Mining and Metallurgy, QLD, pp: 345-348.

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Mirabediny, H., (1995), "*Dragline Simulation: Case Studies*", ECS's Users Annual Conference, Published Internally, Bowral, NSW. pp: 8-15.

Mirabediny, H. and Baafi, E.Y., (1996). "*Effect of Operating Technique on the Dragline Performance: A Monitoring Data Analysis*", Mining Science and Technology, Geo, Y. and Golosinski, T. S. (eds.), A. A. Balkema, Roterdam, pp: 479-485.

Mirabediny, H., (1996), "*The Use of Monitoring Data in Conjunction with Dragline Simulation*", ECS's Users Annual Conference, Published Internally, Bowral, NSW. pp: 17-24.

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Mirabediny, H. and Baafi, E.Y (1998), "*Dragline Digging Methods in Australian Strip Mines: A Survey*", First Australasian Coal Operators Conference (COAL98) , Wollongong, NSW).

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ABSTRACT

During recent years, the Australian coal industry has increasingly used large walking draglines as the dominant waste removal equipment in open cut coal mines. Because of the nature of the coal formations, dragline operations in Australian coal mining situations are quite complex and draglines are frequently used in applications beyond their normal capabilities. With the current trend to increasing dragline sizes in most of the Australian coal mines, the draglines become the highest capital investment item in these mines. It is therefore necessary to give detailed attention to the optimising operating procedures of the dragline.

Dragline productivity and its stripping capabilities are directly affected by the selection of digging method, strip layout and pit geometry. Every mine has a unique combination of geological conditions. The operating methods that work well at one mine may not necessarily work at another site. Selection of an optimal stripping method, strip layout and pit geometry for a given dragline must be considered with respect to the geological conditions of the mines. With increasing geological complexity of Australian strip mines, it is becoming more important to use sophisticated techniques such as computerised mine planning methods to assist in optimising the dragline operations.

A computerised dragline simulation model (CADSIM) has been developed for use in selection of optimum strategies for a dragline operation. The procedure developed links with a geological ore body model to develop a geological database for simulation. CADSIM model can be used in selection the most cost effective dragline digging method. A specific simulation language, "DSLX", was used to program seven common and innovative dragline methods currently used in Australian open cut mines. The DSLX language uses predefined functions to build strip geometry, working benches, blast profiles and spoil piles. The outputs from CADSIM model in form of volumetric, swing angles and hoist distances data were then aggregated with dragline specifications and site time study data to compare productivity and costs of the selected digging methods. The results of two case studies showed that this procedure lends itself to the "optimum" solution for dragline mine planning and design problems for a given coal deposit.

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LIST OF SYMBOLS AND ABBREVIATIONS

2D	:	Two Dimensional
3D	:	Three Dimensional
ACARP	:	Australian Coal Association Research Program
ACRIL	:	Australian Coal Industry Laboratories
ADPC	:	Australian Dragline Performance Centre
ASCII	:	American Standard for Communication Interchange
bcm	:	bank cubic metre
BE	:	Bucyrus Erie (commercial name)
CAD	:	Computer Aided Drafting
CADSIM	:	Computer Aided Dragline Simulator (a dragline simulation model developed in this thesis)
Co.	:	Cooperation
DAC	:	Discounted Average Cost
DCF	:	Discounted Cash Flow
deg	:	degree
DMS	:	Dragline Monitoring System
DSLX	:	Dragline Simulation Language for X windows (commercial name)
ft	:	feet
GMS	:	Gridded Seam Modelling
hr	:	hour
IDW	:	Inverse Distance Weighted
IRR	:	Internal Rate of Return
km	:	kilometre
l	:	litre
lbs	:	pound
m	:	metre
m^3	:	cubic metre
Max	:	Maximum
Mbcm	:	$\times 10^6$ bank cubic metre
Min	:	Minimum

MSL	:	Maximum Suspended Load
Mt	:	$\times 10^6$ tonne
MUF	:	Maximum Usefulness Factor
N.A.	:	Not Available
No.	:	Number
NPV	:	Net Present Value
NSW	:	New South Wales
Op. hr	:	Operating hours
PC	:	Personal Computer
QLD	:	Queensland
R	:	Correlation Coefficient
R^2	:	Coefficient of Determination
R_e	:	Dragline effective reach
Reh	:	Rehandle
SDE	:	Specific Dig Energy
sec	:	second
SF	:	Swell Factor
St. Dev.	:	Standard Deviation
t	:	tonne
y	:	year

CHAPTER ONE

INTRODUCTION

1.1 GENERAL

Australia is currently the largest coal exporting country in the world and one of the most efficient producers. To maintain this status, the Australian coal mining industry must remain economically competitive with other coal exporting countries. The viability of Australian coal mines primarily depends upon maintaining export contracts. The industry is sensitive to export market price fluctuations. Reducing overall mining costs by increasing the efficiency of equipment through systematic mine planning is the usual strategy adopted by the management of these mines.

Open cut mining in Australia is facing the greatest challenge in its history in attempting to compete not only with other operations internationally, but also with underground operations domestically. Most flat dip and shallow depth surface-mineable coal reserves have been depleted during the last two decades and new open cut operations must extract deeper coal deposits. As open cut coal mines move into deeper areas and the stripping ratios increase, the relative cost of overburden removal also increases. It therefore becomes even more important to design the mine around the optimum overburden removal scheme. The deeper mines are usually multi-seam operations with a more difficult geology and with more geotechnical and hydrological problems. The production efficiency of mines with irregular geology is influenced by many factors. Deeper mines are therefore subjected to more and greater problems requiring more involved mine planning and design, such as the selection of the optimum mining method and pit layout. In planning and design of such operations, the number of alternative methods which need to be considered is consequently greater.

1.2 STATEMENT OF THE PROBLEM

Unlike underground mining, the productivity of Australian open cut coal mining has disappointingly been static during the last two decades with the annual raw coal output per man employed remaining the same in 1986/87 as it was in 1970/71 (Wentworth, 1988). Although there are several reasons for this steady status, the major factor is insufficient technical improvement in mining methods. In NSW, there has been a significant growth in using dragline operations compared with other mining methods since 1980 (Figure 1.1). In the past twenty years the walking dragline has emerged as the dominant overburden removing machine in surface coal mining operations in Australia. There are now over 60 large walking draglines operating in Australian open cut coal mines (Aspinal, 1992). Four new units were expected to be ordered in 1996 and possibly another four units in the next five years. The book value of these new draglines is about A$800 million (Hamilton, 1996).

Please see print copy for image

Figure 1.1- Comparison of coal production by principal mining method in NSW (NSW Coal Industry Profile, 1996).

To remain competitive in today world's market, open cut mines must reduce the overall cost of mining operations. In a typical open cut coal mine, overburden removal accounts for more than 60% of the total mine-site costs. It is therefore important for open cut operators to concentrate on overburden removal for possible reductions in mining costs. A dragline with a 50 m^3 bucket in a typical mine may make 350,000 cycles per year with the average cycle time over one minute. With a stripping ratio of 10:1, a 1% decrease in the average cycle time (0.6 second) would uncover an additional 18,000 tonnes of coal per year. At $30 a tonne of coal this amounts to about $540,000 a year extra income for a typical operation. This 1% increase in productivity of all of Australia's dragline operations could increase the industry's sales of coal by more than $30 million a year.

In most of Australian strip mines, draglines are operating in geological conditions different from those in other parts of the world. The overburdens are deeper and complex with many seams. Overburden depths at many mines have already reached depths which draglines alone cannot handle them without additional pre-stripping equipment such as truck and shovel. Many Australian mining companies are currently faced with the decision either to continue stripping to increasing depths or to commence underground mining operations (Wentworth, 1988). These specific conditions require an extensive analysis of each dragline's working method to establish:

1. the operating limits for the machine,
2. the productivity during chop cut and rehandling operations, and
3. the efficient sequences of different mining activities.

A review of several case studies of stripping operations by Atkinson et al (1985) clearly indicated that the stripping capabilities of the draglines used in Australian open cut coal mines were not fully utilised, resulting in their low operating efficiency. There are several ways to increase the efficiency of overburden removal operations, such as improved design of dragline components. However, dragline productivity improvement through the modification of the digging method is the most cost effective and usually the most efficient means (Pippenger, 1995). The feasibility of significant improvement in dragline performance (up to 20%) through modifications to the digging method has been reported by several mines. The idea of modifying the digging method becomes increasingly more attractive as stripping ratios increase with mine life, particularly in multi-seam operations. Various operating scenarios that can improve the efficiency of a dragline operation can be evaluated by the use of scientific management techniques such as system simulation.

1.2.1 Development of a Database for Digging Methods

In order to develop an effective simulation model for a dragline operation it is necessary to have a thorough understanding of the characteristics of the digging method and sequencing of the excavation operations. A review of related literature showed that most of the available literature describing basic dragline digging methods applied to the US coal fields. Australian dragline mines generally have greater overburden and to some degree have more difficult geological conditions than US and European strip mines. Small draglines are rarely used and no tandem dragline operation currently exists in Australia. Many Australian dragline operations are using innovative digging methods to cope with these more difficult geological conditions and to increase dragline capabilities such as maximum reach and dump height (Brett, 1985). Because of the deeper overburden, most Australian strip mines have wider pits, typically 60-80m versus 40-50m pit width overseas, in order to reduce rehandle and avoid both spoil and highwall failures.

There is no comprehensive study evaluating various digging methods currently in use by Australian open cut coal mines. Very limited information can be found describing innovative digging methods and most of them are internal and confidential mine reports. However, to provide basic information for this study and to highlight the current status of Australian dragline coal mining, a questionnaire was prepared and sent to twenty eight open cut coal mines with a total of about sixty large walking draglines as major overburden removal units among the mines. The questionnaire sought information about general geology of the coal deposit, the mine's dragline(s) and other major equipment specifications, details of the pit geometry with a particular reference to the dragline digging methods. A number of site visits was also undertaken to directly observe and evaluate current dragline operations. The results of the questionnaire have been summarised in Table 1.1.

Table 1.1- Summarised results of the digging method survey.

No	Geology Condition				Dragline Specification					Digging Method	Prod-uctivity (Mbcm/y)	Digging Method Description
	Number of Seams	*Coal Thick-ness (m)*	*Waste Thick-ness (m)*	*Strip Width (m)*	*Model*	*Bucket Size (m^3)*	*Operat-ing Radius (m)*	*Dump Height (m)*	*Dig Depth (m)*			
1	1	8	55 (then to 35)	80	Marion 8200	55	87.2	45	47	Modified Low wall Technique	12 (Total) 9 (Prime)	After a cast blast, a small fleet of shovel and truck reduces the overburden depth from 35m to 55m and then dragline with a modified lowwall pass removes the rest of the overburden. Using this method, rehandle is reduced from 30% to 7%.
2	1	0.5 - 6	25	60	BE 1370-W	47	87.2	57	47	Cast Blasting & Extended Bench	9.8 (Total) 6.2 (Prime)	In some areas the dragline removes two seams in one pass, chopping the first interburden. The main overburden is initially cast blasted with 30% thrown and then the dragline removes remaining of material using an Extended Bench method.
3	3 Seam 1 Seam 2 Seam 3	0.5 - 1 0.5 - 1 5 - 7	26 - 30 1.5 -5 2 - 6	60 - 70	BE 1570-W	52	92	47.8	45.7	Three Pass Method	6.5 (Prime)	The first pass method is a standard underhand technique with a highwall keycut. The spoil is directly cast into the previous strips void with no bridging involved. In the second and third pass dragline technique is a lowwall pass involving the digging of the mid-burden from lowwall pad.
4	1 (up to 4 splits)	18 - 22	65	70	BE 1350-W	41	85	30	45	Two Highwall Passes	10 (Total) 3.9 (Prime)	Two highwall pass standard method. The first pass is a Simple Side Casting with some rehandle for the key cut. The second pass is a standard key with the Extended Bench.
5	1 (up to 6 splits)	20	70	80	Marion 8750	103	87	57	63	Standard Extended Bench & Advance Bench	22 (Total) 16 (Prime)	A single pass Extended Bench method where overburden thickness is less than 45m. A single pass Extended Bench with overhand chopping and cast blasting for more than 45m overburden. A two pass Extended Bench where overburden thickness exceeds 60m. The first pass is more productive.

Table 1.1- Summarised results of the digging method survey (Continued).

No	Geology Condition				Dragline Specification					Digging Method	Prod-uctivity (Mbcm/y)	Digging Method Description
	Number of Seams	*Coal Thick-ness (m)*	*Waste Thick-ness (m)*	*Strip Width (m)*	*Model*	*Bucket Size (m^3)*	*Operat-ing Radius (m)*	*Dump Height (m)*	*Dig Depth (m)*			
6	1	13-17	35 to 45	90	BE 1370-W	47	87.5	52.2	48.8	Extended Bench & Extended Key Cut	N.A.*	Two of the three pits are mined using standard Extended Bench method with a 10 metres advanced bench, and the third pit uses Extended Key Cut associated with cast blasting.
7	1	6 - 8	33 - 40	80	2 BE 1370-W	46	88	41	50	Extended Key Cut	12.5 (Total) 10.5 (Prime)	The overburden is removed in a two-stage operation. The first stage involves the use of shovel/loader and truck pre stripping operation to provide a dragline working level of 33 m. An 11% improvement in performance was obtained in 1993 changing digging method from Extended Key Cut to Extended Bench.
8	1	10	40	60	Marion 8050	54	73.6	38.1	51.8	Lowwall In-Pit Bench	10.2 (Total) 8.8 (Prime)	The overburden is removed in a two-stage operation. In the first stage dragline sits on a pad prepared on spoil pile and chops a narrow main cut. The material is spoiled in previous pit to make a new pad (bench) for next dragline position. In the second stage dragline removes the old pad and spoils material in the final spoil room.
9	3 Seam 1 Seam 2 Seam 3	 1 - 5 2 - 6 2 - 4	 15 - 20 25 15	50	BE 1570-W	60	87.5	45	55	Single Highwall & Double Lowwall	7.5 (Prime)	The first pass method is a standard underhand technique with a highwall keycut. The spoil is directly cast into the previous strip void with no bridging involved. In the second and third pass the digging method is a lowwall pass involving the digging of the mid-burden from lowwall pad.

* *N.A = Not Available*

Table 1.1- Summarised results of the digging method survey (Continued).

No	Geology Condition				Dragline Specification					Digging Method	Prod-uctivity (Mbcm/y)	Digging Method Description
	Number of Seams	*Coal Thick-ness (m)*	*Waste Thick-ness (m)*	*Strip Width (m)*	*Model*	*Bucket Size (m^3)*	*Operat-ing Radius (m)*	*Dump Height (m)*	*Dig Depth (m)*			
10	2 Seam 1 Seam 2	 2 - 10 7.5	 30 -50 3 - 30	70	Marion 8050 BE 1370-W BE 1260-W	47 47 25	87.2 84.5 74.7	45.7 39.6 39.6	53.3 51.8 51.8	Standard Extended Bench & Extended Key Cut	N.A.*	The Extended Bench method is used in the single seam. The maximum overburden depth is generally 45m. The second method (Extended Key Cut) is employed in multi-seam. The critical factor in this method is spoil room. When designing a multi-seam area, a maximum spoil height of 80m is used.
11	1	4 - 6	15 - 200	70	3 BE 1370-W 2 Marion 8050	48 48	86 83.8	43.5 44.7	45.7 51.8	Extended Bench, Extended Key Cut & In-Pit Bench	70 (Total for all draglines)	A trial was implemented at the mine to establish an Extended Key Cut method. Although higher swing angles are required in this technique, the overall coal exposure rate is increased due to lower rehandling of the material. Rehandle decreases from 63.4% to 47.1% and a 17% time saving was provided through the implementation of new method.
12	2 Seam 1 Seam 2	 2-3 4-6	 0-25 11	40	Marion 7700	19	68	32	45	In-Pit Bench Chopping	2.9 (Total) 2.3 (Prime)	The first interburden is cast blasted and the material is pushed with dozer or a small fleet of truck and shovel to make an in-pit bench for dragline. The dragline pulls back material of the second interburden.
13	2 Seam 1 Seam 2	 6 - 10 8 - 10.5	 15 - 45 10 - 30	55-75	2 BE1370-W 2 BE1350-W 4 Marion 8050	48 36 48	84.5 87.0 87.2	39.6 30.5 45.7	52.8 45.0 53.3	Extended Bench, Extended Key Cut & In-Pit Bench	N.A.	Coal is removed by open-cut mining methods, using five draglines for overburden removal. A BWE and associated conveyors used to assist in pre-stripping operations. Two truck and shovel stripping fleets also operate at the mine.

* *N.A = Not Available*

Table 1.1- Summarised results of the digging method survey (Continued).

No	Geology Condition				Dragline Specification					Digging Method	Prod-uctivity (Mbcm/y)	Digging Method Description
	Number of Seams	*Coal Thick-ness (m)*	*Waste Thick-ness (m)*	*Strip Width (m)*	*Model*	*Bucket Size (m^3)*	*Operat-ing Radius (m)*	*Dump Height (m)*	*Dig Depth (m)*			
14	2	4 - 5	30-55	60	3 BE 1370-w 3 Marion 8050 Marion 195M-2	48 48 12	86 83.8 N.A.	43.5 44.7 N.A.	45.7 51.8 N.A.	Chop Cut In_Pit Bench & Stacked Multi-Seam Method	N.A.*	The first method involves a three-pass operation in conjunction with cast blasting. Using the chopped material an in-pit bench is constructed progressively along the pit to complete the first pass. The final pass involves removing the remaining materials in spoil pile. This method showed a 20% increase in the uncovering rate compared with standard Extended Bench method as a result of a decrease in rehandle about 25%.
15	1 (up to 3 Splits)	4 - 6	20 - 60	45 - 70	4 Marion 8050 BE 1370-W	46 46	87 87.5	45 43	50 48	Extended Bench, Extended Key Cut & In-Pit Bench	25.2 (Total) 15.0 (Prime)	A two-pass sequence with the alluvial unit being stripped down to expose the top of the sandstone forming an in-pit bench on which the dragline would operate to strip out the sandstone unit. Drilling and blasting operations would follow the stripping of the first pass.
16	5	3.5 - 4	15 - 50	60 - 70	Marion 8200 Marion 8200 Marion 7900 Marion 7901 BE 1370-W	72 57 27 30 47	87.5 92 72 72 87.2	50.9 48 38 38 43	60 60 40 40 45	Extended Bench, Extended Key Cut & In-Pit Bench	39.4 (Total) 29.0 (Prime)	A combination of three common methods is used. The current methods are standard Extended Bench with chop, in pit bench method and Extended Key Cut. A 72 m^3 replacement dragline was erected on site in December 1993.
17	3 Seam 1 Seam 2 Seam 3	 0.5 - 1 0.5 - 1 0.4 - 4	 26 - 30 1.5 -5 2 - 6	40-50	BE 1370-W	46	92	47.8	45.7	Single Highwall & Double Lowwall passes	9.0 (Prime)	The first pass method is a standard underhand technique, with a highwall keycut. The spoil is directly cast into the previous strips void with no bridging. In the second and third passes digging technique is a lowwall pass involving the digging of the midburden from lowwall pad.

* *N.A = Not Available*

Table 1.1- Summarised results of the digging method survey (Continued).

No	Geology Condition				Dragline Specification					Digging Method	Prod-uctivity (Mbcm/y)	Digging Method Description
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18	3 Seam 1 Seam 2 Seam 3	2 - 4 0.5 - 3 0 - 2.5	12 - 50 6 - 30 2 - 22	50-70	4 Marion 8200	57	87	48	60	Multi-Pass Operation	N.A.	The operation utilises four draglines for overburden and interburden removal, generally in two passes. The first pass is a highwall keycut and the second pass is a chop pass from a lowwall bench. Where parting exceeds 7 m in thickness, a lowwall pass is utilised.
19	2 Seam 1 Seam 2	2 - 19 2 - 10	13 - 32 6 - 22	55-60	Marion 7820	28	80	45	40	Extended Key Cut & In-Pit Bench	N.A.	The operation starts with digging an extended key cut along the highwall, side casting overburden into the adjacent diversion channel beyond the designed lowwall batter. The key cut spoil is then used as an elevated working level for the dragline whilst pulling back the remaining prime overburden.
20	1	5-7	25-58	55	P&H 9020	93	87	57	65	Extended Key Cut	22 (Total) 17 (Prime)	The thick interburden is cast blasted after pre-stripping the top seam overburden. The overburden is then blasted using a cast blasting technique with 25% thrown. The dragline first makes an extended key cut along the 500m strip in the first pass and then walks to the a leveled spoil side bench to pull back the remainder of material.

* *N.A = Not Available*

Of the twenty-eight mines, twenty-one mines (75%), covering fifty-one dragline operations responded to the questionnaire. One mine was not using its dragline any more. The remaining 25% did not respond because of either lack of operational data or the company did not have personnel available to gather the requested data. The information provided by the mines was classified according to the mine geology. The details included number of dragline passes, number of lifts per pass, dragline positions, whether or not a thrown blasting technique is used, and cut and spoil procedures.

Seven digging methods were identified to be representative of most of the Australian dragline operations. The common stripping methods are:

1. Simple Side Casting,
2. Standard Extended Bench with an advance bench,
3. Split Bench (deep stripping),
4. Chop Cut In-Pit Bench,
5. Extended Key Cut,
6. Single Highwall and Double Lowwall Multi-Pass, and
7. Double Highwall and Single Lowwall Multi-Pass.

As shown in Figure 1.2, there is a significant tendency towards digging methods with higher productivity such as Extended Key Cut and In-Pit Bench method. During recent years there have been modifications to the conventional techniques for a variety of reasons, including:

- Requirement for closer control on production costs,
- Introduction of more efficient cast blasting techniques,
- Development of multi-seam operations, and
- Significant increases in overburden depths.

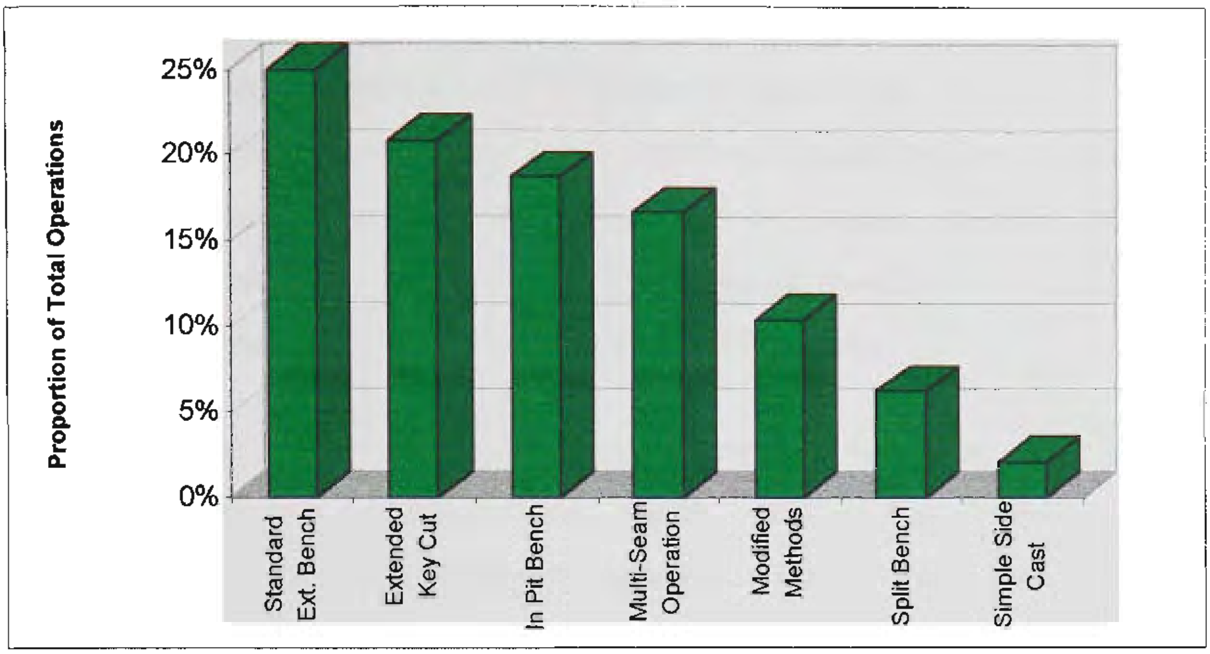


Figure 1.2- Dragline digging methods used by most Australian open cut coal mines.

The selection of the best digging method depends on a combination of geological conditions, dragline size and characteristics, and management planning targets. The nature of the coal deposit and geological conditions such as the number of seams, overburden/interburden thickness and coal thickness are among the most important factors governing the choice of a digging method. Other factors such as geotechnical conditions, spoil stability, blasting techniques, material strengths and engineering and operator's experience are also important in the selection of a digging method. The combination of various factors results in using a wide variation of methods at strip mines. Shared experience among different sites of a company owning various draglines is an important factor in selection of digging method. For example, BHP-Utah Coal Limited (BUCL) operates 35 draglines of varying sizes across the Bowen Basin of Central Queensland (Hill, 1989). The four common methods used by the BUCL group are:

1. standard extended bridge,
2. deep prestrip (split bench),
3. extended key cut, and
4. in-pit bench.

Ideally the digging method which results in the highest coal exposure rate must be adopted for a particular operation. The choice of strip geometry is mainly governed by the selected stripping method and the size of dragline. The stripping operations commenced with box-cuts on the shallow area at depths up to 15 to 25m. The depths have increased over the years and average overburden depths now are around 50 to 55m in single seam operations. In many cases additional stripping capacities such as truck and shovel fleets are being used ahead of dragline operation. In some instances, draglines are being used to dig depths as much as 70 metres.

Unlike overburden depth which is mainly governed by the geology, strip width is a determinant factor which can be varied within a practical range. Variations in strip width affects productivity of dragline operation. Pit geometry, especially the strip width, must be evaluated in conjunction with the digging method adopted by the mine. Wide strips (greater than 60m) are more preferable for methods such as the standard extended bench method due to the reductions in the rehandle, while narrower pits are more productive for methods using a cast blasting technique, such as extended key cut or in-pit bench method. The strip widths currently employed by the mines ranged from 40 to 90 metres with an average of 60 to 70 metres.

Various sizes of draglines are in use in Australian mines. The bucket size of the current draglines varies over the wide range of 12 to 103 m^3. Normally smaller draglines are used to remove the shallow depth interburdens. Most of the recently ordered draglines or those under contract have larger stripping capacities when compared with the old generation of draglines (Seib and Carr, 1990). The dominant form of dragline ten years ago was a medium size dragline such as BE 1370W or Marion 8050 with bucket capacity around 47 m^3 (Atkinson et al, 1985). The new generation of draglines in Australian mines has an average bucket capacity around 75 m^3. Contributing factors toward the very large draglines are the increasing overburden depths, the need to increase stripping capacity of the mine to improve total economics, and advances in the technology of the dragline manufacturing. Figure 1.3 shows the changes in dragline size and its stripping capability during the last two decades.

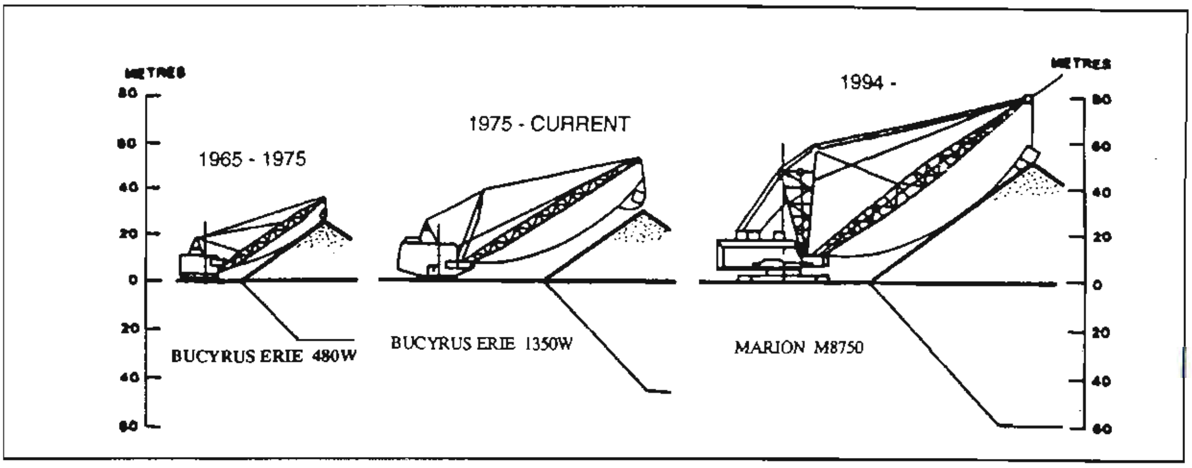


Figure 1.3- Increases in the dragline size over the last two decades (After Seib and Carr, 1990).

Draglines move waste at the lowest cost per unit volume only when they work within their normal range. Both efficiency and productivity of a given dragline drop off dramatically with changes in its effective operational factors. In order to improve the performance of a dragline, its mode of operation and influencing parameters must first be fully understood and analysed. Finding the normal working ranges for a given dragline and optimising its operation requires repetitive arithmetic and analytic solutions. This problem is ideally suited to the application of computer aided simulation methods. Better mine planning and mining method selection through computer simulation has been successful in many cases and this has been strongly recommended for the Australian operations (Atkinson et al, 1985; Hill, 1989; Wentworth, 1988; Aspinal, 1992; Sengstock, 1992). Most of the strip mines surveyed reported that a simulation model which can simulate different mining methods (particularly the innovative ones) would be a useful means for selection of the optimum dragline digging method for a given geology.

Computer simulation of dragline operation has the potential for rapid, low cost analysis of different mining scenarios. Simulation of the dragline operation enables an operator to test the logic of how the machine should be used, and the design of optimum operating methods for the varying mining conditions. Such an application may also be used as training simulators or to evaluate dragline performance in a given geological and operational condition. Computer simulation can also be used for evaluating proposals for modifications to existing operations and is also useful in comparing the performance of different types of new draglines which are being considered for purchase (Hill, 1989).

Although several computer packages have been developed to simulate dragline coal mining operations, most tend to be limited to regular geological structures where the total mining area can be represented as a simple generalised cross section. Most of the currently available packages are limited to the standard digging methods or specific mining conditions. Often an inflexible "black box" approach is used to determine the "best" mining parameters. This implies that the user cannot follow the logic of the computer package and also has no means to change or extend the software limitations (Michaud and Calder, 1988). Conventional computer based dragline operation simulators use a trigonometric approach to carry out the required calculations such as volumetric calculations. With the conventional approach, the simulation process becomes inefficient and tedious for an irregular geology. This is especially true in optimisation processes where iterative runs are necessary. To evaluate various mining scenarios for an irregular geology, the dragline simulation must be performed on a full set of closely spaced sections that do not necessarily have similar geological characteristics. Recent developments in 3D CAD computer packages have provided the opportunity to automate the process and this has overcome some of the limitations of the conventional approach.

In the mining industry, there is an increasing tendency to use computer systems which provide an integrated approach with related modules for all phases of a mine's development. Most of the recently developed commercial computer packages consist of modules for all mining activities from the initial exploration through to the mine closure. These computer packages have graphical facilities shared among all modules to provide some degree of manipulation and presentation of different aspects of the mining activities. Many of mine planning computer packages are linked with 3D CAD systems to achieve extensive flexibility in producing interactive graphics appropriate to the varying needs of the user.

1.3 OBJECTIVES OF THE THESIS

The main objective of this thesis was to develop a computerised simulation model that can be used as a tool to evaluate mine planning alternatives for dragline operations. The simulation model evaluates the effect of changes in digging method characteristics, strip layout, pit configuration and machine size based on either productivity or costs of the operation for a given set of geological conditions. An automated approach is taken to speed up the repetitive analytical procedures and input/output processes required for a dragline mine planning analysis. This approach uses 3D CAD procedures to carry out the required cut and spoil calculations for a given geology and dragline specifications. The automation of the whole process allows the model to simulate numerous sections on a full deposit quickly and hence a number of mining options can be evaluated and compared to arrive at an optimal solution. To accomplish the thesis objectives, a computerised dragline simulation model (**CADSIM**: **C**omputer **A**ided **D**ragline **SIM**ulator) was developed which consists of three main inter-connected sub-models and one auxiliary sub-model. These are:

I. Geological interface model,
II. Dragline stripping model,
III. Analytical model, and
IV. 3D graphical image sub-model

Geological Interface Model: The geological model intersects cross sections and gridded structural surfaces to create strings representing the geology of the sections to be simulated. These strings are than stored into ASCII files to be accessed during the simulation of the dragline stripping operation. The original pit layout and critical points such as intersection of strip lines and sections are also defined in this stage.

Dragline Stripping Model: The stripping model simulates the digging, spoiling and walking actions of the dragline. The dragline simulator provides relevant data required for productivity calculations and 3D graphic outputs. In this thesis, the dragline simulator repeats a set of calculations for a large number of mining blocks within a coal deposit. The result of the calculation is highly dependent on the geological conditions which may significantly change along the strips and also from one strip to another. To include important tasks such as effect of coal access ramps on the dragline spoil room, effect of adjacent blocks on each other and walking grade control, the simulation must be carried out on a whole deposit rather than a few representative blocks. Unlike CADSIM model developed in this thesis, most of the commercial available dragline computer packages do not have such a capability.

The cyclic nature of a dragline operation requires an automated simulation process. Here an automated process means the elimination of unnecessary interruptions by the user. For example, the geological information for all the simulating blocks must be defined once and the appropriate information is automatically accessed during the simulation of each block. Once the geology is set up, the process of the calculation of cut and spoil profiles and volumetric calculations is continuously repeated for all the sections and strips for a given mine design. The automated process which was developed in this thesis allows the design to be carried out for the whole deposit quickly and evaluate various mining scenarios within a reasonable time. Such an approach also provides detailed block by block information, including coal and waste tonnage and rehandling percentages. This type of information is very useful in short term scheduling as well as cost ranking calculations.

Due to the variety of digging methods currently used by open cut mines, a more general approach was necessary for simulation rather than using standard digging methods such as extended bridge. In this thesis a highly flexible simulation language "DSLX" has been used to program different dragline digging scenarios. Such an approach provided a library of various dragline digging techniques. The results from the simulation stage are then aggregated with time study data into a spreadsheet software to estimate productivity and costs of the operation.

Analytical Model: This phase calculates productivity of the simulated options so that a comparison can be made for selection the optimum scenarios among various options simulated for a given geology. The analytical model also estimates the cost of the operation and investigates the sensitivity of the results due to variation in input parameters.

3D Graphical Image Model: The image model draws a 3D view of the simulated pit at any stage of the mining operation using the output from the dragline stripping phase. This phase was designed as a graphical aid for visualisation of the operations such as location of the ramps or profile of the final spoil.

A schematic of the modelling approach is summarised in Figure 1.4.

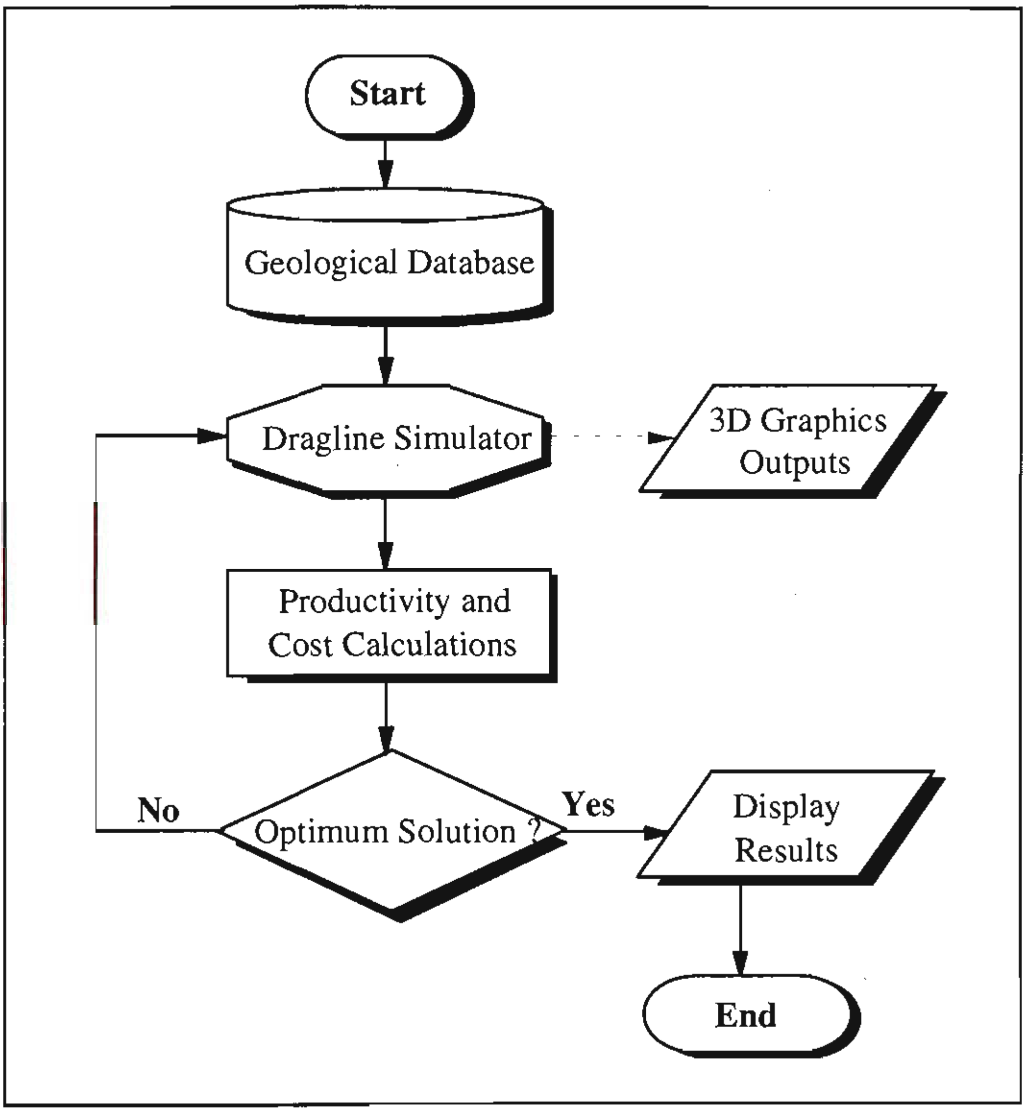


Figure 1.4- A schematic of the modelling approach.

CHAPTER TWO

OVERBURDEN REMOVAL WITH A DRAGLINE

2.1 INTRODUCTION

For shallow and single seam operations, there is little variation in dragline operational techniques. Often standard dragline digging methods such as Simple Side Casting and Extended Bench are the most satisfactory operating techniques adopted for the removal of overburden in shallow to medium depths. However, as operations extend into deeper areas with more difficult geological conditions, it is essential that different dragline operating techniques are employed. These techniques differ in operating characteristics in terms of the number of dragline passes, dragline positions and walking patterns, digging modes (e.g. underhand or overhand), rehandle percentage, swing angle, hoist distance and cycle time components. For example, in a multi seam dragline mining system, the dragline works from the highwall in the first pass and during the second pass phase, walks to the lowwall side to pull back the interburden from the spoil side in-pit bench. In addition to the changes in the mode of operation, the mine geology, pit configuration, blasting technique, operator's experience and the dragline specifications affect the performance of the machine, and hence the rate of coal exposure.

2.2 DRAGLINE DIGGING OPTIONS

A dragline is mode of operation depends both on the digging positions and on how it drags the bucket. The bucket can be dragged in three distinct operational modes which are:

1. Normal or underhand digging,
2. Overhand digging or chop down operation, and
3. Pull back operation from spoil side.

A normal or underhand digging mode consists of removing the key cut, main cut and Extended Bench. In underhand digging the dragline usually works from the highwall and digs material from below its pad elevation (Figure 2.1). The swing angles are relatively short, usually within 30 to 120 degrees.

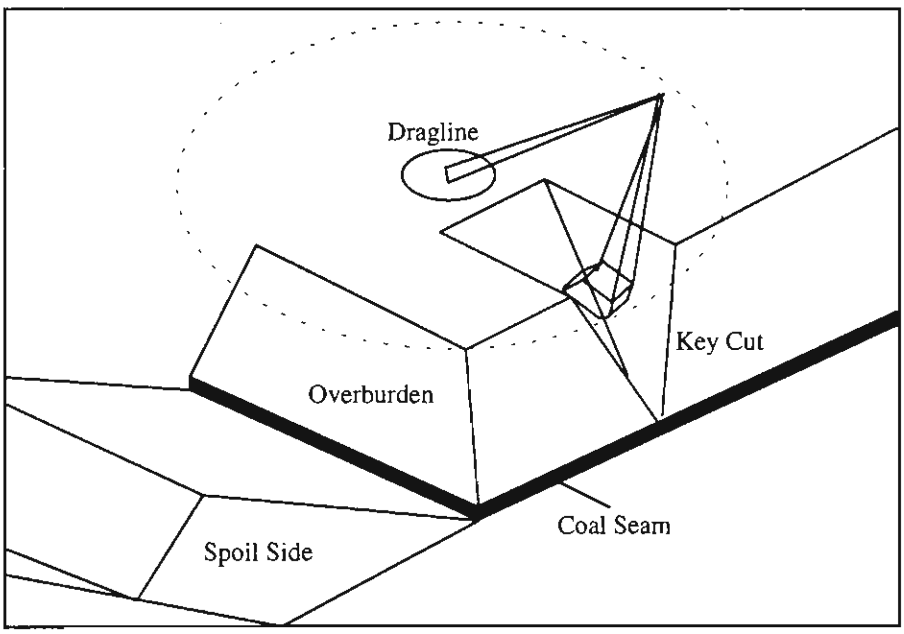


Figure 2.1- Dragline excavating a key cut in a normal underhand mode.

The overhand chop cutting mode is another dragline digging mode that refers to the dragline excavating material from a bench above its working bench (Figure 2.2). This mode of operation is frequently met in soft or undulating ground surfaces. In this situation, the bucket is usually held in a dump position and the teeth are dropped into the material to give the bucket penetrating force. To complete the digging procedure and filling of the bucket, the bucket is dragged downwards and towards the dragline. The

The overhand chop cutting provides a stable and even working bench for the dragline in unconsolidated or rolling areas. It also increases the maximum digging depth of the dragline (Bucyrus-Erie Co, 1977).

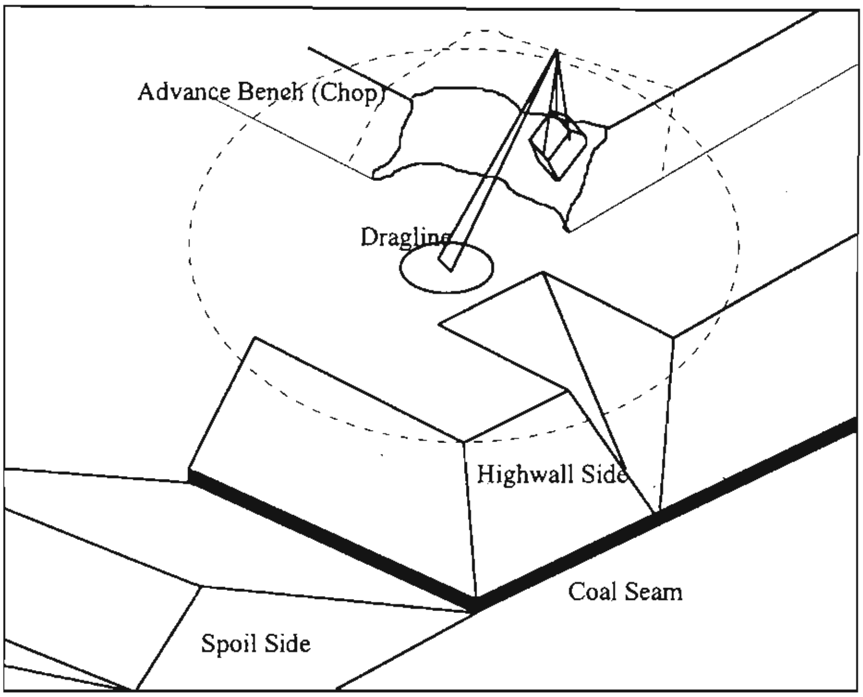


Figure 2.2- Dragline removing an advance bench in an overhand chopping mode.

Overhand digging generally decreases the total productivity of the dragline because the machine is working in a less efficient mode. The productivity losses result from longer bucket filling times, lower fill factor, longer swing angle (usually between 130 to 150 degrees) as well as increasing the dragline movements compared with normal conditions. In addition, the situation increases wear on the rigging, ropes and bucket resulting in an increased down time and repair costs.

Pull back from the spoil side is a common method in multi-pass operations or where the dragline does not have enough room to spoil all the material from a normal bridge. The dragline pad in the spoil side is built at a higher level than a normal bridge and closer to the spoil area (Figure 2.3). The technique is often used as an alternative to extended benching, particularly when the size of the dragline is insufficient to permit spoiling the material in the existing spoil room from a highwall bench. Pull back digging has been used frequently in multiple seam mining situations. The technique is also suitable for tandem dragline operations, where two or more draglines are working with each other.

Like the chop down operations, the dragline works in a less efficient mode than the normal mode of operation (underhand digging).

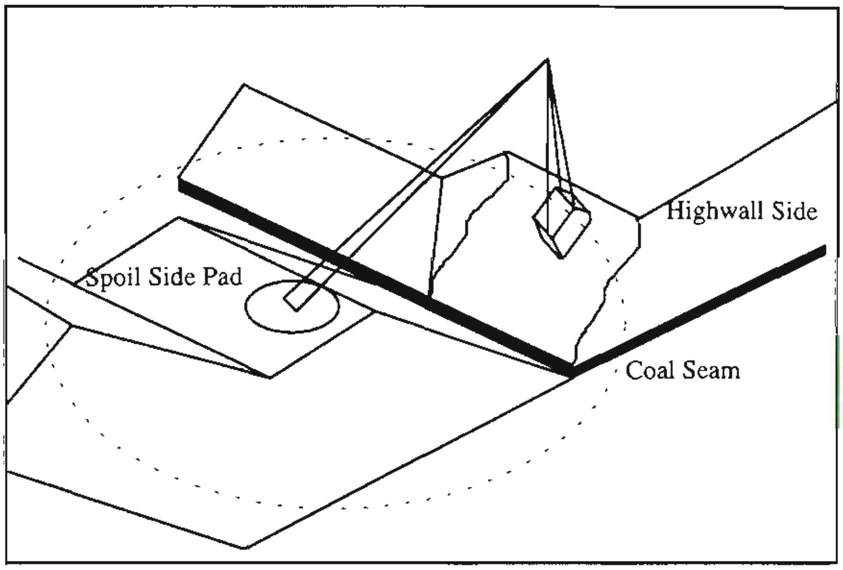


Figure 2.3- Dragline pulling back material by chopping against the highwall from a spoil side pad.

The main disadvantages of the pull back technique are:

1. increased dragline walking times,
2. inherent hazards of unstable spoil piles,
3. complexities of the planning and sequencing of the mining operations due to the number of different situations specially in multi-seam operations, and
4. additional dozing operations to prepare the dragline pad on the spoil side and the building of an additional ramp and bridge.

Spoil side stripping can result in lower productivity due to the longer swing angles and bucket fill times because of the "chopping" action of the bucket against the highwall (Elliott, 1989). Although there are several disadvantages associated with the pull back operation, the technique is used by many mines with multi-seams and deep overburden conditions.

2.3 DRAGLINE DIGGING METHODS

The main use of a dragline in a strip mine is to remove overburden material in order to expose the underlying coal. To accomplish this task a limited number of basic modes of operation is used in practice. Considering the geological conditions and equipment size, different logical combinations of these modes can be sequenced to complete the stripping of a block of material by the dragline. The sequencing of the operating modes performed by the dragline is defined as the "digging method" (also called stripping method). There are more than twenty traditional dragline stripping methods worldwide (Michaud, 1991). The common digging methods currently used by Australian mines are:

1. Simple Side Casting,
2. Standard Extended Bench with an advance bench,
3. Split Bench (Deep Stripping),
4. Lowwall In-Pit Bench,
5. Extended Key Cut,
6. Single Highwall and Double Lowwall Multi-Pass, and
7. Double Highwall and Single Lowwall Multi-Pass.

These digging methods are characterised by the number of coal seams, overburden depth, the dragline movements and the number of dragline lifts.

2.3.1 Simple Side Casting Method

For a single seam operation with shallow overburden depths the basic method is *Simple Side Casting*. Theoretically no rehandle is involved in this method, although some rehandling is required around the coal access ramps. For a medium size dragline with a 90m operating radius, Simple Side Casting is only possible in shallow depths (usually less than 30m). Shallow depth is not only the depth of the overburden, it is a relative variable which is also related to the size of the dragline used. For example, if a dragline can excavate all of the depth of overburden by Simple Side Casting, then the overburden depth can be called "shallow".

The basic operations in a Simple Side Casting method are shown in Figure 2.4a.

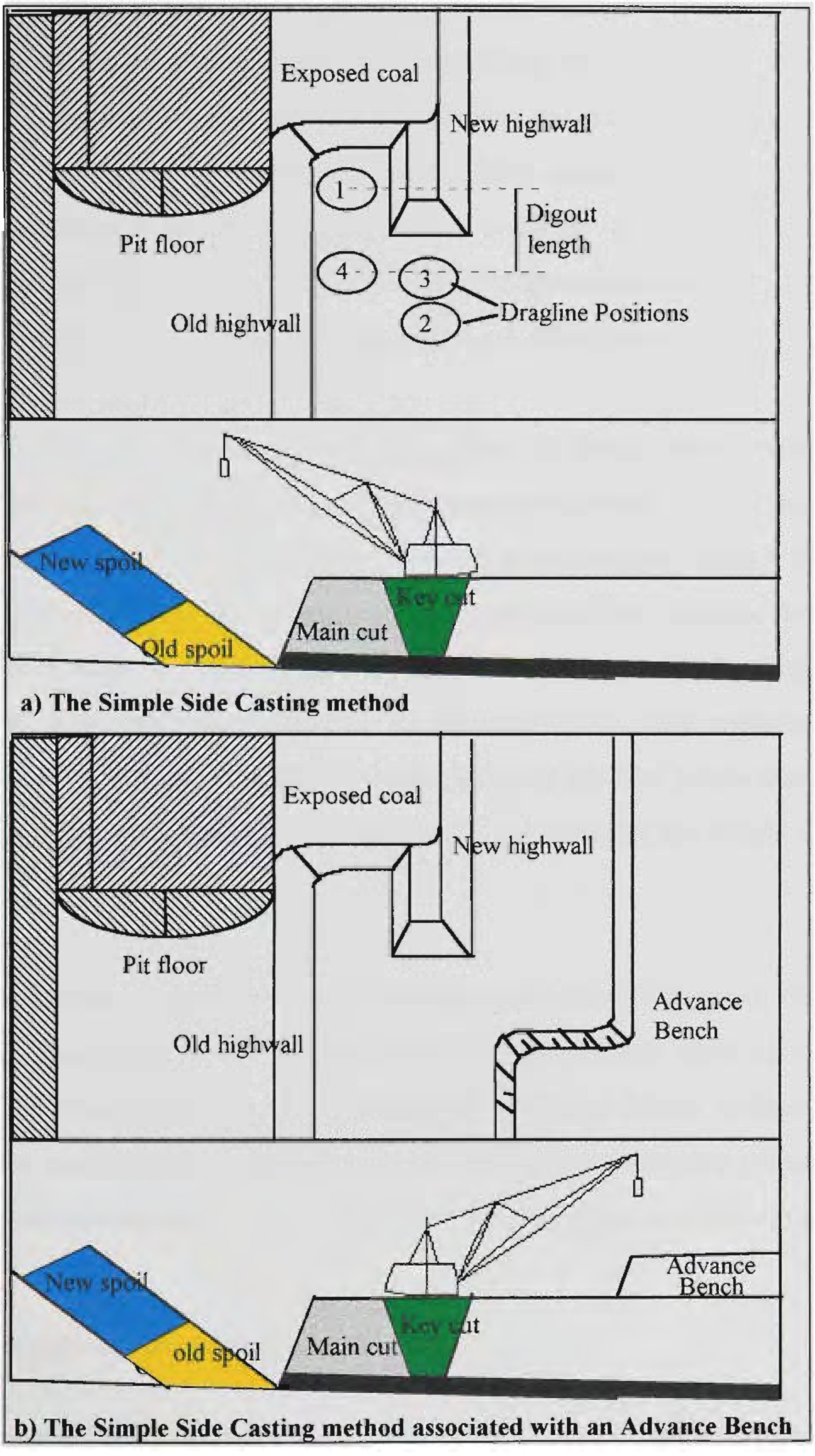


Figure 2.4- Plan and section view of a Simple Side Casting method.

To remove the overburden, the dragline walks from the last position (Position 1) sitting on the natural surface to remove a key cut from the new planned highwall line. The key cut is a narrow trench about one and a half bucket width at the bottom. While excavating the key cut, the dragline sits on a line over the middle of the key cut in order to establish a clean and safe new highwall. Key cutting is usually the most difficult part to excavate due to the poorer fragmentation and lack of a third free face. The dragline may move to position 3 from position 2 to prevent the drag ropes from dragging on the overburden with increasing depth of key cut. After excavating the key cut the dragline moves to position 4 to complete removal of a block of overburden. The distance between two positions 1 and 4 is called a digout length which may vary from 20 to 35m, depending on the size of the dragline and the overburden depth.

When the overburden depth increases, very often, the Simple Side Casting method can be modified to an Advance Bench method to avoid rehandling. In this method, the level of the dragline working bench is kept lower than the original surface by forming the advance bench (Figure 2.4b). The main disadvantage of the Advance Bench is that the dragline must work in an overhand chop mode of operation. Swing angles are greater (130 to 180 degrees) when removing an advance bench, thus reducing the dragline productivity. There is a limitation for the height of an advance bench that a dragline can handle. In many operations a maximum of 15m is used for the height of the advance bench.

As the overburden depth increases and because of the disadvantages of chop operations, many strip mines prefer to use alternative digging methods such as two underhand passes (Split Bench method) and the standard Extended Bench method. The choice between the use of either an advance bench or a method involving rehandling depends on the overall productivity, coal exposure rates and the associated mining costs.

2.3.2 Standard Extended Bench Method

With increasing depth of overburden and for wide pits, the dragline cannot spoil all material in a Simple Side Casting operation without rehandling a high percentage of the material moved. By increasing the dragline reach factor and sitting the dragline closer to the spoil pile it is possible to provide enough spoil room so that all the overburden can be dumped in the void of an old pit. To increase the capabilities of the dragline, a bridge is usually made by extending the dragline working level toward the spoil pile. This extension allows the dragline to reach the required spoil peak. The material used to make the extended bridge is provided from both the key cut and the advance bench. The extended bridge is a rehandling material and must be excavated from the next block to clear the coal edge. The minimum required bridge extension can be determined by balancing the volume of the block being removed and the spoil room available. The dragline spoiling capabilities, material swell factor, highwall angle and spoil repose angle are critical factors that effect the amount of rehandle.

It is also possible to combine the standard Extended Bench method with a throw blasting operation. A general view of such an operation used by some Australian dragline operations is illustrated in Figure 2.5. To start the excavation of a new block, the dragline walks off the last position (normally extended bench) and sits on a new position in line with middle of the key cut. The spoil from the key cut is used to form the bulk of the new extended bench. If the excavated material from the key cut is not sufficient to make the bridge, the dragline continues to extend the bridge using material resulting from extending the main cut to a new position. The location of this new position is dictated by the dumping position and operating radius of the dragline, in other words, the dragline should reach from this position to the final extension of the bridge.

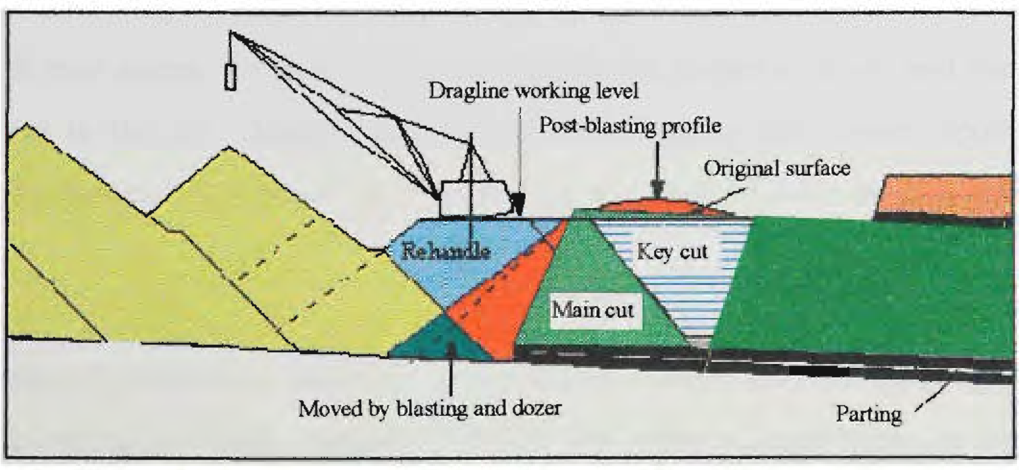


Figure 2.5- General view of a standard Extended Bench method combined with a thrown blasting technique.

After forming the extended bench, the dragline progressively moves onto the newly constructed extended bench. In wider pits (exceeding 60m) the dragline usually uses a third position in order to keep swing angles to a minimum. This ensures that the inside corner of the remaining material is removed from the final position. The dragline must be able to place material directly on the final spoil room from the third position. The removal of the previous extended bench is carried out from final position of the dragline near the edge of the new extended bench. A trench with a width equal to one dragline bucket (5 to 7m) may also be used to ensure that the lowwall coal edge is exposed.

2.3.3 Split Bench (Deep Prestrip Method)

For a deep operation, the dragline working level may be limited by the maximum overburden depth that the dragline can dig. The percentage of rehandle material also increases significantly with increasing overburden depth. One possible way to reduce rehandle is the use of an advance bench ahead of the dragline main pass. This method is called Split Bench method and is commonly used to remove thick overburden covering single thick coal seam, as found in the Central Queensland. In order to increase the maximum advance bench depth and to avoid a chop down operation, two underhand dragline passes are used in each strip. A general view of the method is shown in Figure 2.6. The method involves two highwall passes using simple side casting in the first pass and a conventional extended bench in the main pass. The main dig level is controlled by the maximum dig depth of the dragline and the spoil balance. When the method is used in thick coal seams, the main dig level controls the undercut trench and the amount of coal lost in the rib. Ideally, the method must balance coal losses, spoil room and rehandle material by controlling the dragline dig level in both the first and the main passes.

Theoretically there is no rehandle associated with the first pass stripping. However, due to limited dragline reach, specially in wider pits, some material from the first pass key cut may be rehandled. A dragline tail clearance of 25 to 30m is allowed in the main pass. The overburden depth which can be removed by a medium sized dragline such as BE 1370-W (45 m^3 bucket size) can be increased up to 70 to 80m using this method.

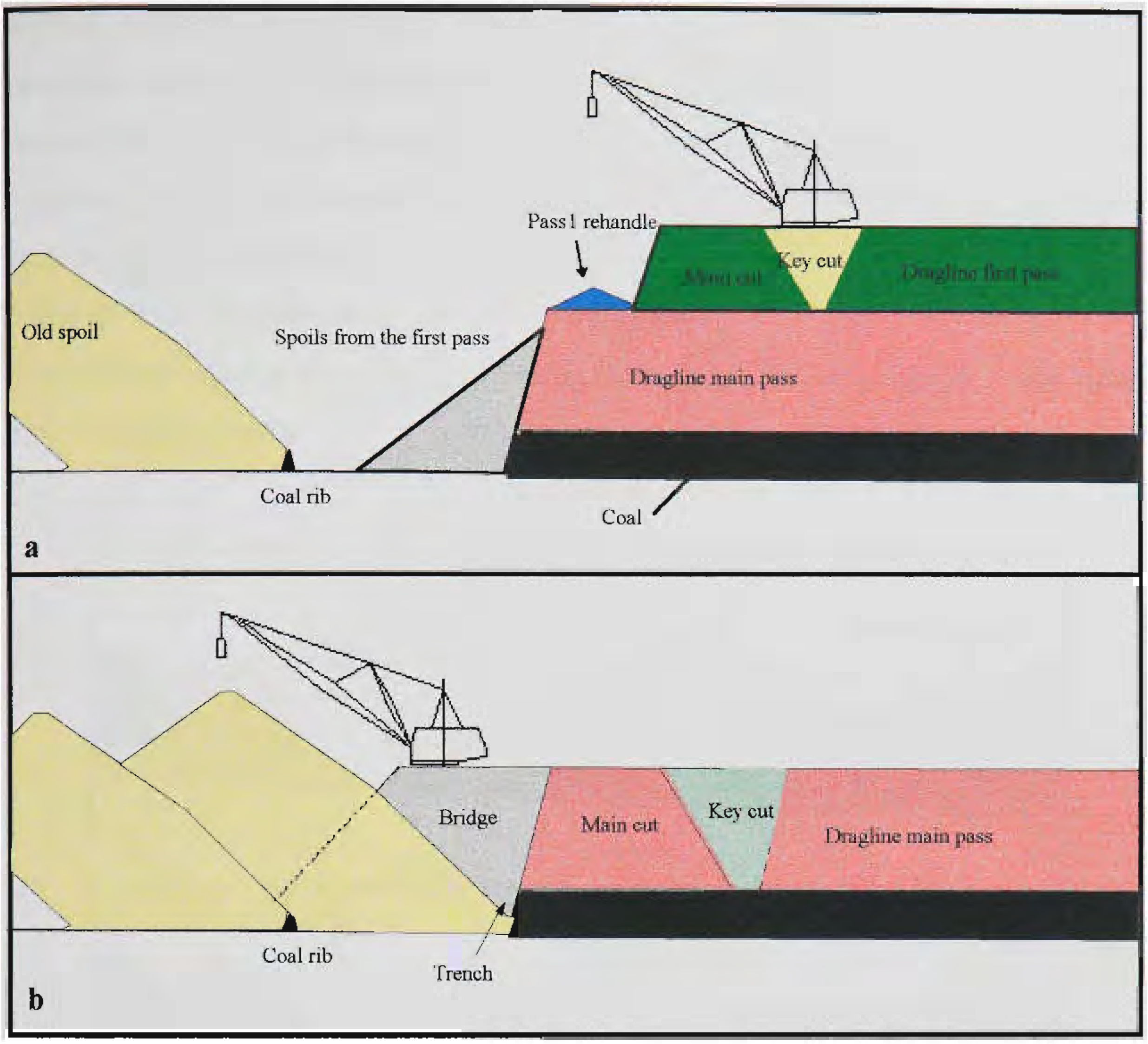


Figure 2.6- A typical cross section of the Split Bench dragline digging method.

2.3.4 Lowwall In-Pit Bench Method

In many cases and for a medium depth overburden, the dragline dump height is not a limiting factor. The depth of the extended bench affects the amount of reh4andle, so a significant reduction in rehandle can be achieved by having the bridge level as low as possible while still providing sufficient spoil room. This can be achieved by using an in-pit bench in a lower elevation instead of a normal bridge. There is also less restriction on the level of the in-pit bench than in digging methods such as the standard Extended Bench. It is possible to minimise the amount of rehandle material by optimising the level of the in-pit bench.

Two common methods which use lower benches to reduce rehandle are the lowwall In-Pit Bench and the Extended Key Cut. A general view of a Lowwall In-Pit Bench method is shown in Figure 2.7. This method uses a single pass two lifts lowwall operation employing a highwall chop in the first lift and a pull back operation in the second lift. The method is associated with a throw blasting technique. Typically the dragline bench in the lowwall side is about 10 to 15 metres below the pre-blasted surface. As the operation progresses along the strip, a dragline return road is built at a width of 40 to 45 metres to allow the dragline to walk back to the opposite digging area after completing a strip. The dragline digging sequences for a typical In-Pit Bench method are as follows:

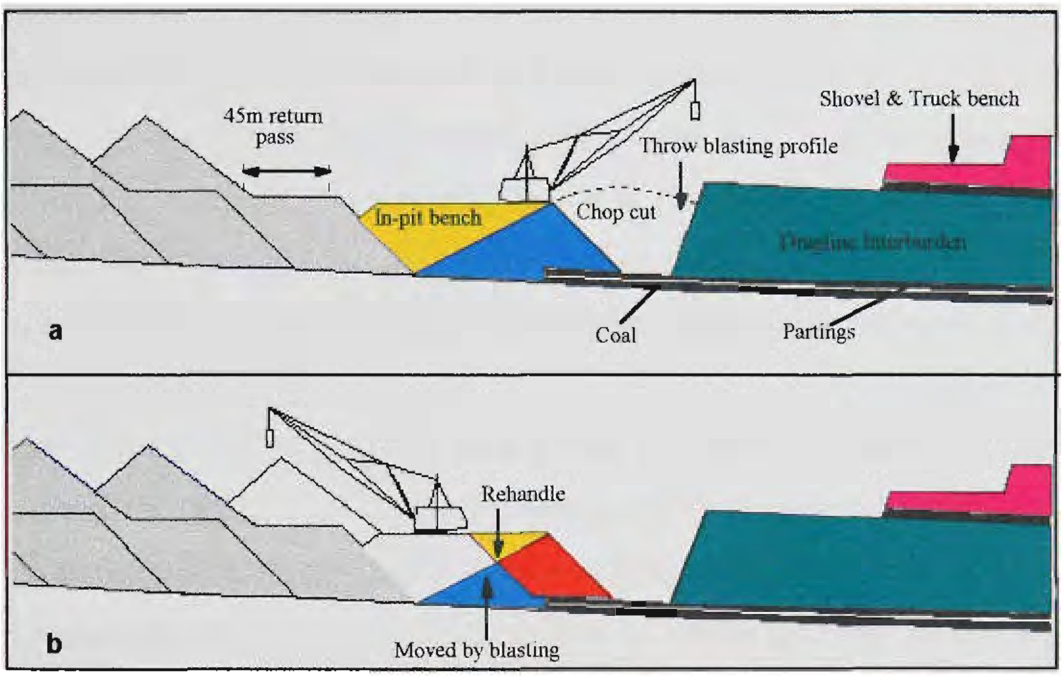


Figure 2.7- A general view of the Lowwall In-Pit Bench chop cut method.

First the overburden is blasted in such a way as to achieve maximum throw of the material into the old pit. The dragline then walks towards the lowwall and makes a pad (in-pit bench) on the blasted material at the lowwall side of the pit. The dimension of the in-pit bench is governed by the geometry of the pit, blasting profile and the dragline specifications (particularly its operating radius and its maximum dump height). Usually the in-pit bench is made 10 to 15m below the original highwall level. Once set up, the dragline works solely from the in-pit bench in the lowwall side. The dragline chops the highwall to remove the bulk of the key cut material from its first position in a chop down mode of operation. The material from the key cut is used to establish an in-pit bench at a designated height and grade for the next block along the strip. The dragline progressively moves backward and extends the key cut in a pullback mode of operation.

After creating the in-pit bench, the excessive material is dumped into the final spoil room. The dragline moves backwards into a final position to remove the remaining material of the old bench. The dragline reach from this position must be sufficient to remove the inside corner of the key cut in its final position. The dragline dumps the bulk of the old bench into the area behind the bench so that a 40 to 45m wide return road is left to allow the dragline to walk back after completing the strip.

2.3.5 Extended Key Cut Method

The Extended Key Cut method is a two-pass operation employing a highwall extended key cut in the first pass and a lowwall pull back operation in the second pass as shown in Figure 2.8. As with the In-Pit Bench method, this method is also associated with a throw blasting technique. The dragline bench in the first pass is on the highwall side and in the second pass the dragline works from a bench in the lowwall side about 10-15 metres below the pre-blasted surface. The dragline digging sequences are as follows:

First the overburden is blasted in such a way to achieve a maximum throw of the material into the old pit. The highwall bench is levelled using auxiliary equipment such as a dozer to make a 30m wide pad for the dragline on the post blasting profile above the highwall key cut. The pad must be at least 25 meters far from the crest of the new highwall to provide rear clearance of the machine while it turns. The dragline spends roughly a third of the total digging time to extend a key cut along the strip and forms a pad (in-pit bench) on the blasted material at the lowwall side of the pit at a pre-designed height and grade. The first pass does not uncover coal and therefore the method requires shorter cuts than the Extended Bench and Lowwall In-Pit Bench digging methods (Hill, 1989). The length of the pit with this method must satisfy the coal excavation inventory. Usually a minimum of 500 metres is required for the strip length.

After completing the first pass, the dragline constructs a ramp across the pit towards the lowwall to walk to the pre-prepared in-pit bench. The dragline then chops the highwall to remove the remaining material in a pull back mode of operation. In the second pass, the dragline dumps the spoil in the mined out area behind itself. A 45m wide return road must be left to allow the dragline to walk back after completing the strip. The

Extended Key Cut digging method is characterised by higher swing angles, especially in wide strips.

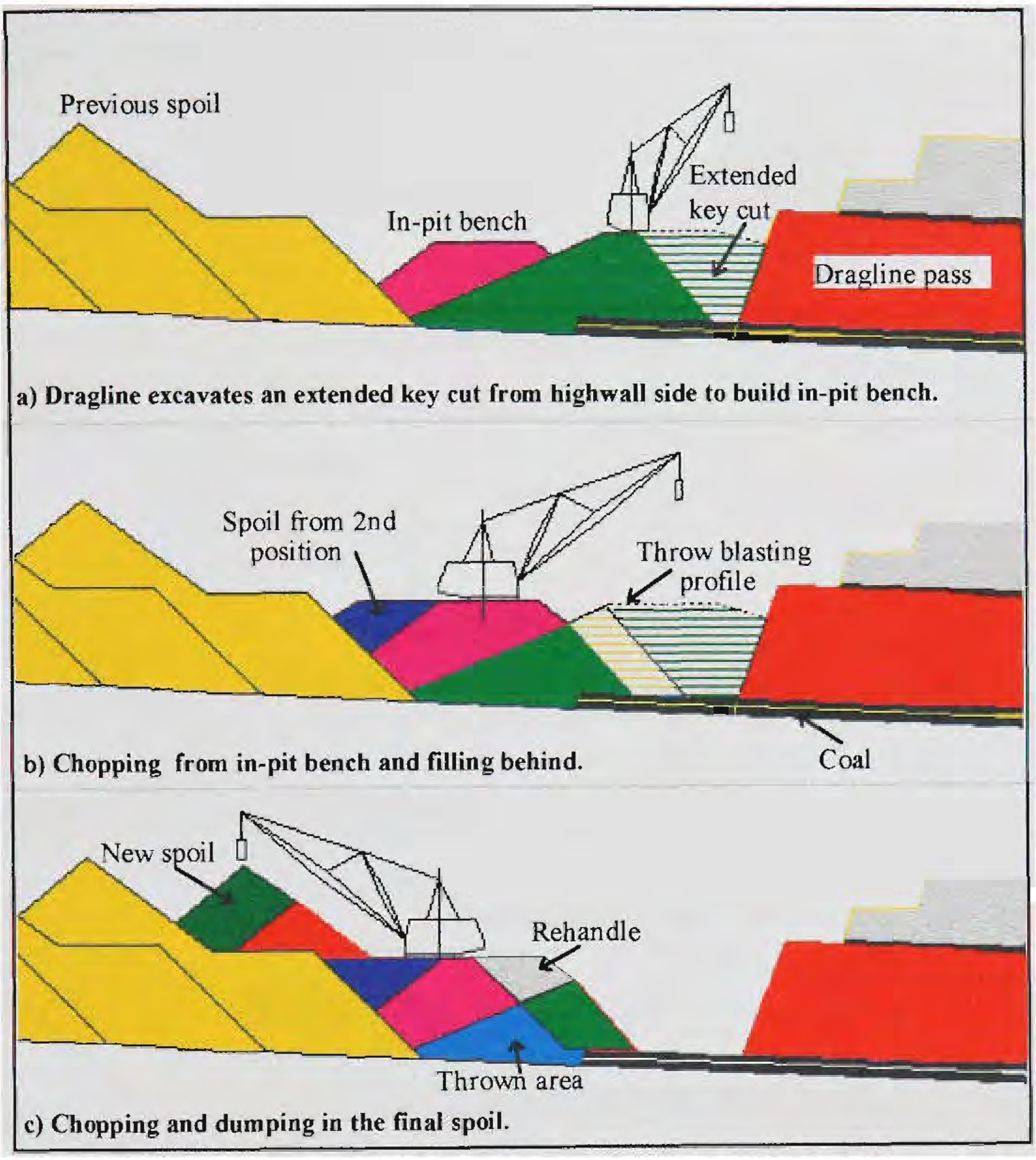


Figure 2.8- A general view of the Extended Key Cut method.

2.3.6 Multi-Pass Extended Key Cut

A variation of the Extended Key Cut method which is designed for deep overburdens without using throw blasting is a typical Multi-Pass Extended Key Cut method. This method is best suited for thick coal seams since the coal loss is minimised as no heavy blasting is required. The sequence of the operation is shown in Figure 2.9.

In this method, the first pass is a Simple Side Casting method and its bench level is determined by a spoil balance between maximum spoil room and total prime volume.

After completing the first pass, the dragline walks down to the main pass dig level to start the second pass (Figure 2.9b).

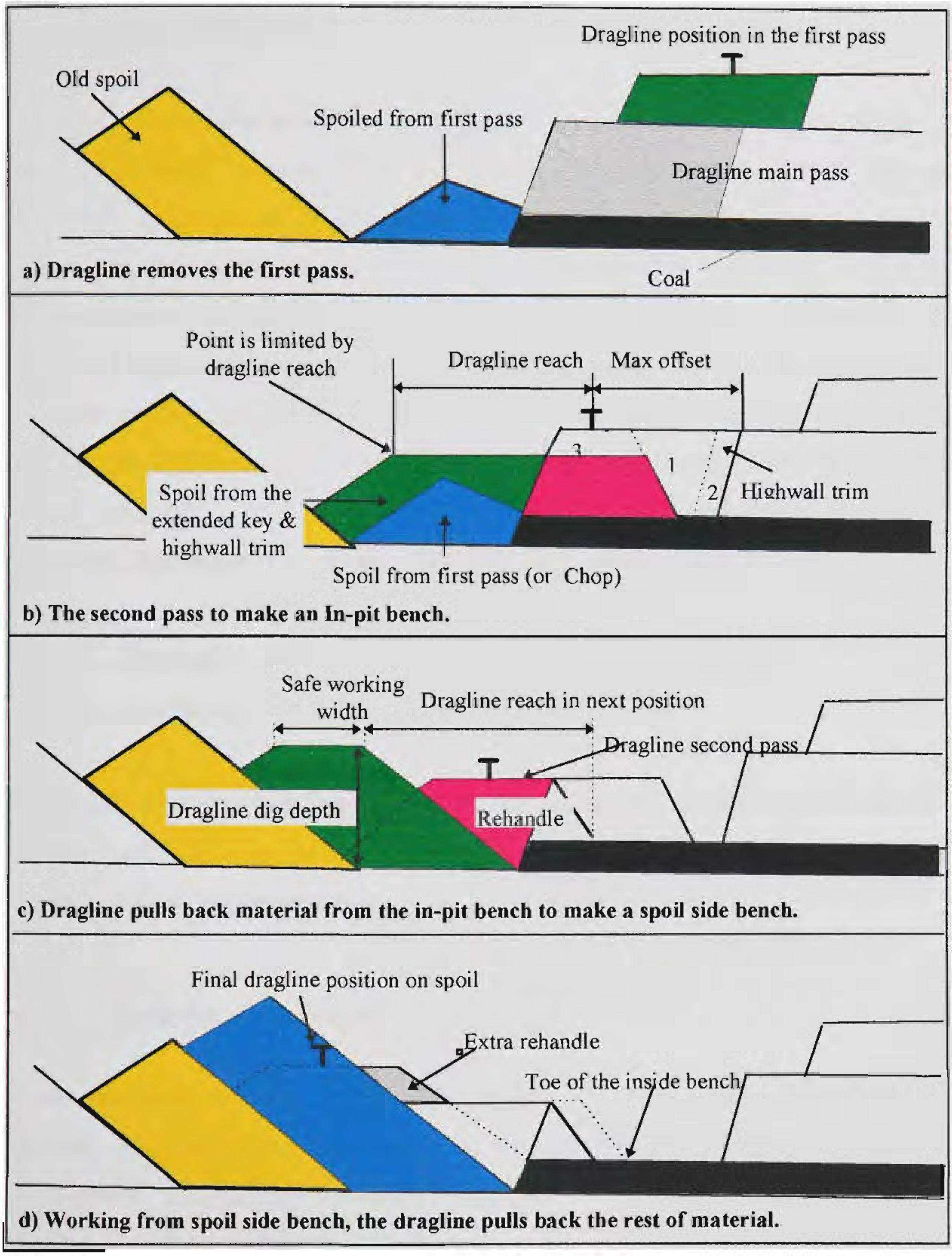


Figure 2.9- Sequence used to complete a Multi-Pass Extended Key Cut.

The second pass involves removal of the following three parts:

1. an inside key with a bucket width at the bottom of the key,
2. a highwall trim cut, and
3. a horizontal layer from the top of the remainder of the main cut.

The depth of the main pass is controlled by the maximum dragline digging depth. Both the material from the second pass and the spoil of the first pass are used to form an in-pit bench. Less material is rehandled when the intermediate bench level is lower than the digging level of the main pass. However, the level of the intermediate bench cannot be reduced too much as the dump height must be enough to create a final bench. There must be a balance between the material from the extended key and the volume required to make the intermediate bench (Figure 2.9b). If the level of the intermediate bench is not high enough to dump all the material in the spoil area, a higher bench must be built in the spoil side. In addition to the level of the spoil side bench, the other two parameters that affect the operating efficiency with this method are:

1. the dragline reach to the toe of the intermediate bench, and
2. a minimum safe working width on the final bench.

When the dragline cannot reach the toe of the in-pit bench, the final bench width should be increased. However, increasing the width of the spoil side bench will result in extra rehandling from the final bench as shown in Figure 2.9d.

2.3.7 Multi Seam Operations

Australian open cut mines rarely expose more than three main coal seams with dragline. In multi seam operations, the selection of a suitable digging method is influenced by the thicknesses of both the overburden and interburdens. Sequencing of the various dragline positions as well as coal mining is more complex in a multi seam operation; these are often the key to success of the entire strip mine operation. A combination of different digging methods is used to complete a multi seam operation. Usually a combination of highwall and lowwall dragline passes is used for waste removal.

Two common methods for a typical three seam operation are:

1. Single Highwall and Double Lowwall, and
2. Double Highwall and Single Lowwall.

Figure 2.10 shows a general view of a typical Single Highwall and Double Lowwall dragline operation. In this method the first pass is a standard underhand technique with the highwall key and main cut components. The spoil is directly dumped into the previous strip void so there is no rehandle for bridging. However, some rehandle may occur mostly due to the coal haulage ramps.

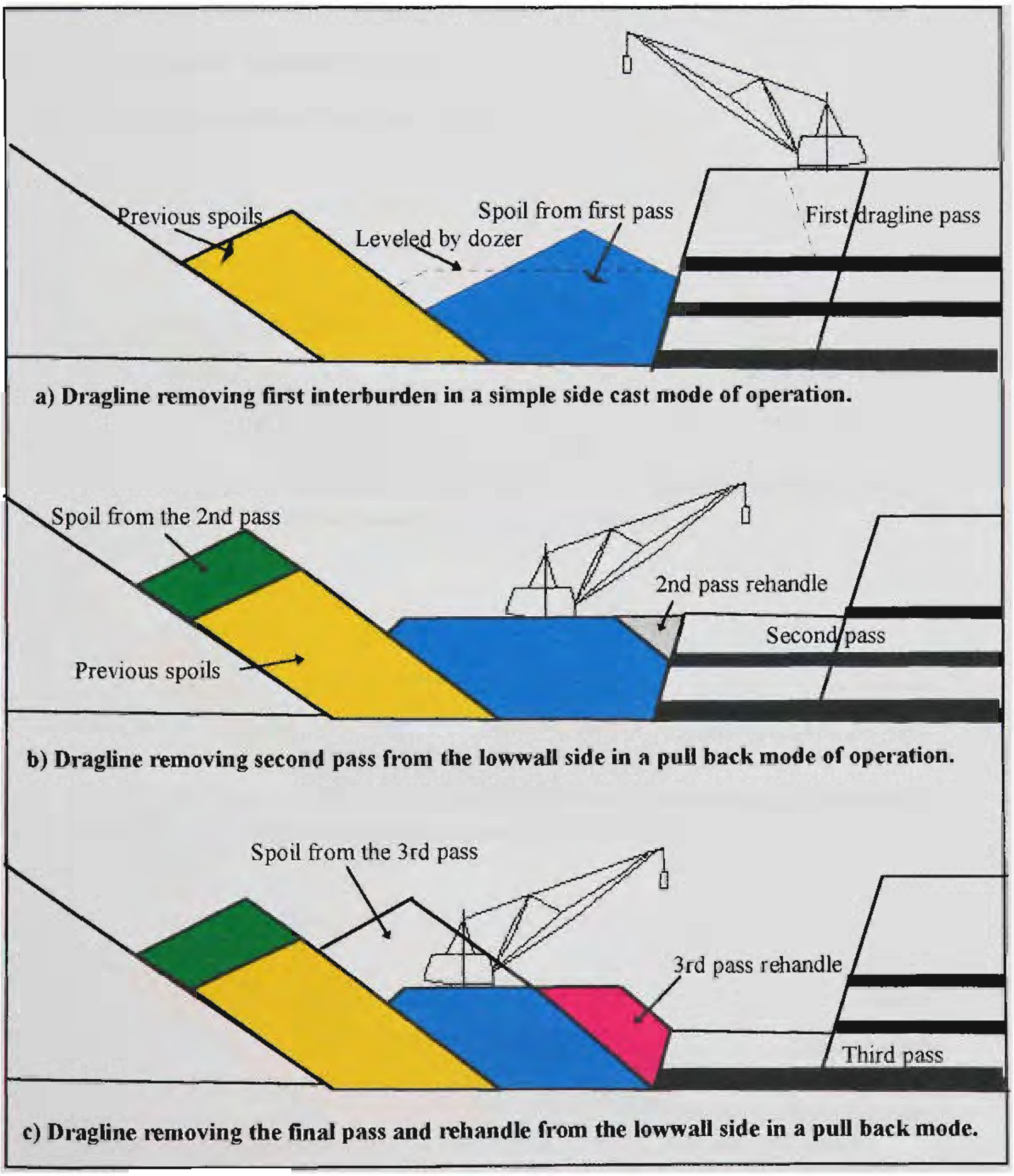


Figure 2.10- Three seam operation, Single Highwall and Double Lowwall method.

The dragline digging technique in the second pass is a lowwall pass method involving chopping operations from an in-pit bench. In the second pass the dragline is subject to tight spoiling and dumping to its maximum height. The need to dump behind the machine greatly increases the cycle time due to a longer swing angle. The third pass is essentially the same as the second pass, but with shorter swing angles. Rehandle percentage in both the second and the third passes depends on the strip width, thickness of interburdens and the dragline size, particularly its operating radius. Normally, the volume of material rehandled decreases with narrower strips.

With a decrease in the thickness of the top overburden, there is a point at which there is not enough material from the first pass to form the lowwall to the required elevation. In this case a Double Highwall and Single Lowwall method may be adopted. Figure 2.11 shows a general view of such an operation.

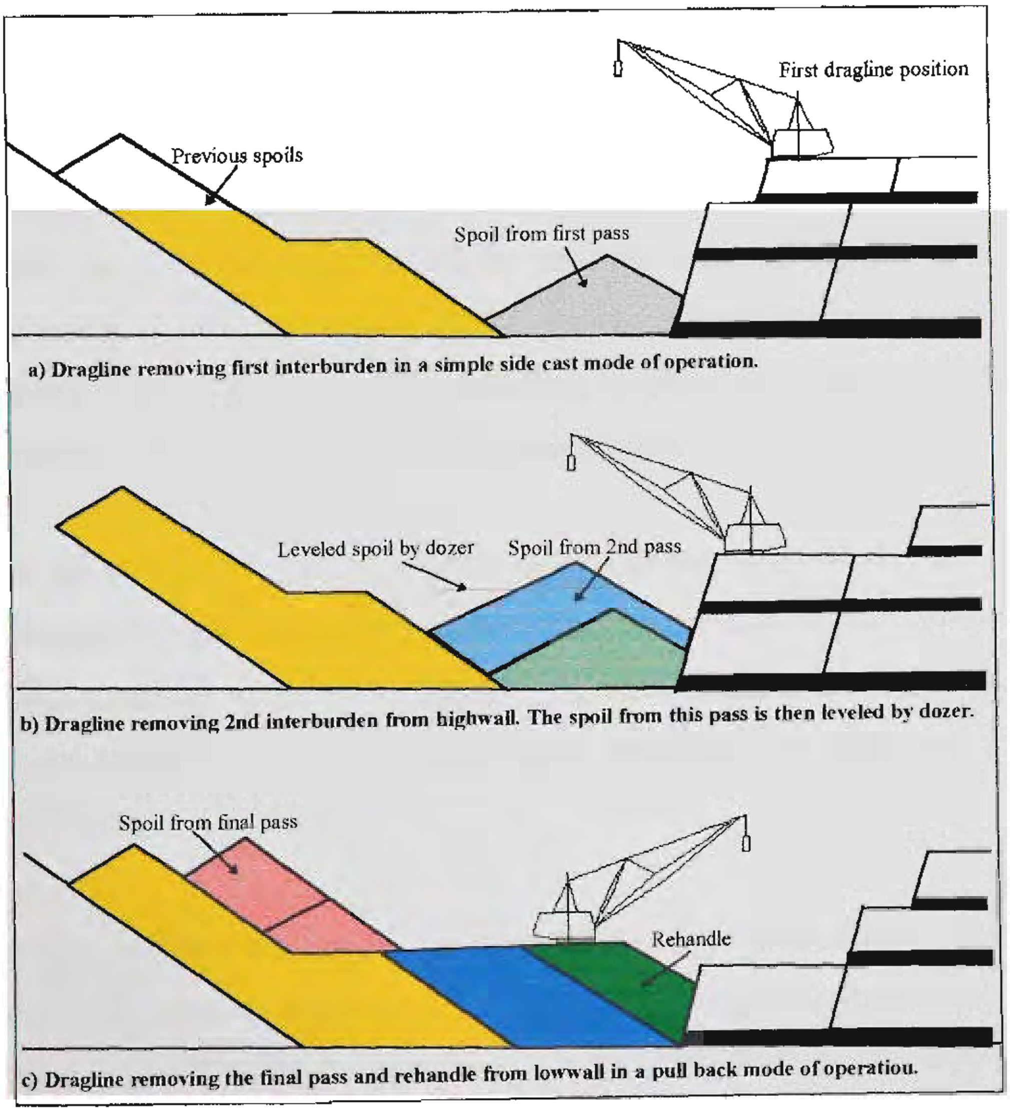


Figure 2.11- Three seam operation, Double Highwall and Single Lowwall method.

In both the three passes digging methods the dragline essentially carries out the same routines. However, in Double Highwall and Single Lowwall method there is no rehandle during removal of the second pass. Pad preparation and dozer works are also required after completing the second pass. In practice, the Double Highwall and Single Lowwall method is more productive than the Single Highwall and Double Lowwall method, provided that the pit geometry, especially the thickness of the interburdens suits the method. The dragline swing angles are shorter compared with the single highwall and double lowwall method. In summary, the Double Highwall and Single Lowwall method is preferred at where the depths of the first and second pass are less than the depth of the third pass.

2.4 BLASTING FOR THE DRAGLINE

Blasting for the dragline is not different from other open cut blasting operations but more care must be taken as this operation is more critical for a dragline operation than for a shovel operation. A dragline excavates overburden by dragging the bucket over the material instead of pushing the bucket into the bank as a shovel does. In poorly fragmented material, dragline productivity can drop more rapidly than that of shovels working in similar material (Morey, 1990). Various diggability indices give the means of assessing blasting efficiency and quantifying its effect on productivity. Some of the more important diggability indices are discussed below.

Specific dig energy (SDE) is the energy consumed per cubic metre for bucket filling and is perhaps the best form of diggability index. A low SDE will be indicative of good diggability regardless of the bucket fill rate. Therefore even if the operator fails to make full use of available drag power under good conditions the SDE will still be low (Phillips, 1989).

Dig force is the force required for bank penetration and bucket filling. This force will be highly dependent on diggability. It varies with operator's proficiency, especially penetration depth of the bucket teeth. Failure to use full machine power by the operator can result in slow drag velocity and low fill rate even under good digging conditions.

Diggability refers to the ease and speed with which the dragline bucket can be filled. It is a function of blasting practice and geology (rock type). Diggability will influence dragline productivity through bucket fill times and volumes. Diggability can be measured by assessing filling time and bucket fill factor from tonnes moved per cycle. If the dragline monitor records the absolute dragline positions, each measure of the dig index will be associated with a three dimensional point. It is then possible to produce plans of diggability contours. These plans can be plotted to the appropriate scale and superimposed on blast plans producing a means of visually assessing blast efficiency. Using the operator codes, areas of uniform dig technique (key cut, rehandle, etc.) can be identified on the plan to account for the influence of dig mode on diggability index.

2.4.1 Throw Blasting

There are two common blasting methods used for dragline stripping. The first method, termed stand up (or standard) blasting, is to use a blasting pattern to loosen bank materials rather than the highwall. This provides the dragline with a stable seat on the highwall when removing the overburden. The second method, termed throw blasting, is to use the energy of the explosives to push the overburden into the spoil area as much as possible thereby reducing the volume of the material that must be removed by dragline. The greatest advantage of the stand up technique is the relatively lower cost of drilling and blasting. It also provides a safer and more efficient working area and requires less levelling works by dozer. The greatest advantage of the throw blasting method is the increased dragline productivity. In case of thick overburden, throw blasting causes a lower dragline working level, resulting in a lower dragline rehandle. However, there is doubt about the cost efficiency of throw blasting because it involves a large increase in drilling and blasting costs and to some extent the dragline pad preparation costs (Morey, 1990). In most dragline operations using the cast-blasting technique, the dragline must work from the spoil pile due to the unsuitable working conditions after a heavy blasting. This increases the swing angle and bucket filling time thus reduces the machine productivity.

Figure 2.12 is an example of the throw blasting for a dragline pit and Figure 2.13 schematically shows details of the throw blasting technique. In this case the dragline working level is lowered by blasting hence reducing the rehandle due to the extended bench. Throw blasting moves a substantial portion of overburden which should be moved by dragline into the final spoil. Both these factors increase the dragline productivity.

Figure 2.12- An example of the throw blasting technique used for a dragline pit.

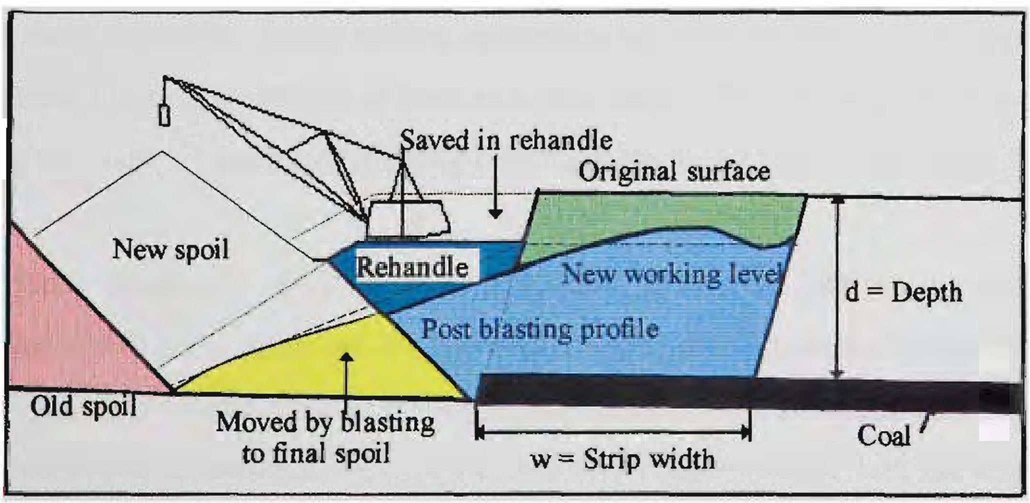


Figure 2.13- Details of a throw blasting technique.

Blast profiles can be measured by survey techniques (conventional or laser methods) and provide a measure of the blast performance relative to the dragline operation. Data calculated from pre-blast and post-blast profiling which affect the cost of overburden removal include (ICI Technical Services, 1996):

1. swell factor,
2. thrown percentage(moved to final spoil),
3. cost of coal damage or loss,
4. rehandle volumes,
5. improved digging rates in looser muckpiles,
6. volumes of overhand digging, and
7. dozer preparation requirements.

Some advantages of cast blasting (Elliott, 1989) are:

1. eliminating benching dug in the chop down mode,
2. reducing rehandle,
3. increasing mining width, and
4. removing partings that are presently being excavated with a shovel and truck system.

All the above items would yield a decrease in operating costs. With the introduction of deeper overburdens in most Australian coal operations, the advantages of throw blasting become more desirable. Many mining operations are now considering the applicability of the throw blasting technique at least in a trial stage[1]. The characteristics and factors affecting the result of the throw blasting technique are reviewed in more detail below.

Depth/Width Ratio: A measure of the efficiency of cast blasting is the thrown percentage which is expressed as the percentage of the prime volume moved by blasting into the final spoil area. Thrown percentage increases with a reduction in strip width. Alternatively, for a constant strip width, the thrown percentage will be expected to increase with increasing depth. A direct relationship exists between thrown percentage and depth/width ratio.

---

[1] Personal communication with mines.

Atkinson (1992) suggested a simple relationship for depth to width ratio (d/w) and thrown percentage shown by the following formula:

$$\text{Thrown percentage} = \left(\frac{57.5 \times d}{w} + 18\right) \times 100$$

where: d = depth of the bench to base of the coal seam (m), and

w = width of the strip (m).

The equation is valid only when d/w varies between 0.4 and 0.9. Field results show that the above equation is optimistic and a high percentage of the thrown material is difficult to achieve (Paine, Conley and Payne, 1992).

It is also important to design the dragline pit so that maximum benefits can be achieved from the throw blasting technique. This can also have adverse effects on the productivity and total economics of the mine. For example, narrow pits result in improved thrown percentage, but total excavation economics are adversely affected by increased dragline rehandle volumes and the influence of pre-splitting costs. The improved dragline productivity through the use of throw blasting can increase the rate of coal exposure. However, throw blasting may not necessarily result in reduced operating costs, due to the large increase in drilling and explosive costs involved (Sengstock, 1992). In specific cases, when capital costs are included, costs are comparable. Increased drilling and blasting costs are offset by reductions in the dragline operating and capital costs. The capital costs can be reduced by selecting a smaller dragline for a new operation or by increasing stripping capacity without a need to buy new equipment.

The economic evaluation of throw blasting is particularly sensitive to pit width. Wide pits (greater than 60m) have traditionally been used in most Australian mines to minimise rehandle and for creation of improved in-pit working area during mining operations. Narrower pits result in improved thrown percentage, but total excavation economics are adversely affected by increased dragline rehandle volumes and the influence of pre-splitting costs. For mines that can maintain good highwalls without pre-splitting, it is expected that the optimum pit width would be less than 55m.

Critical factors that must be considered in an economic analysis of a throw blasting operation are:

- dragline productivity (bcm/hr),
- dragline rehandle percentage,
- coal edge and surface damage,
- powder factor,
- percentage in final spoil position, and
- drilled metres.

The thrown percentage varies considerably due to the site conditions and therefore, any decisions on the selection of this method must be based on a comprehensive study including site trials.

2.5 SUMMARY

Dragline digging methods are determined by a combination of several factors, foremost of which are the number, thickness and spacing of coal seams and secondly the dip of the coal seams. In the Australian context, the dip is the most significant parameter in determining the overburden height which must be prestripped ahead of a dragline.

In most cases the economics of the mining program depend upon the choice of production rate, excavation system, size of equipment and appropriate digging method (Fourie and Gerald, 1992). Selection of a suitable overburden removing method can significantly improve the economics of the mining project. For example, if an alternative method can increase the maximum of dragline dig depth, the need for a pre-stripping operation will be reduced for a certain situation. The geometry of the dragline pits must be determined and optimised in conjunction with the selected digging method.

Selection of a mining method must be based on careful consideration of several factors, including geological and geotechnical parameters, coal characteristics and distribution, equipment size and expected production rate. Heuristic methods are normally used for selection of an optimum digging method. Various possible digging methods are tried considering the effect of geological and operational parameters. The optimum method is then the one which meets production rate while providing lower costs. Productivity and/or rehandle values are usually used as the preliminary criteria in selection a dragline digging method. Computer simulation models are best suited for the digging method selection process as a large number of options can be evaluated quickly.

CHAPTER THREE

STRIP MINE PLANNING AND DESIGN

3.1 INTRODUCTION

A strip mine is a series of parallel and relatively narrow cuts made in the ground surface for the purpose of extracting bedded deposits such as coal. To access this coal, it is usually necessary to excavate and move large quantities of waste. Figure 3.1 shows a typical multi-seam strip mine in the Hunter Valley area, NSW. The common overburden removal techniques in a strip mine are dragline and shovel and truck operations. The most suitable stripping method for a coal deposit is selected primarily on the basis of the geology of the deposit, overburden and interburden depths, topography condition, production requirements as well as reclamation considerations. In general, the dragline stripping system is preferable due to its higher production rate and versatility. The operating costs are less with the dragline while a lower initial investment is required for a shovel and truck system (Learmont, 1983). The factors that must be considered during the development of a strip mine are almost identical in both dragline and shovel and truck operations. In each case the selection of optimum waste excavation sequences and pit configurations rely on complex engineering decisions.

Operational parameters such as pit geometry and sequencing of the dragline operations are usually considered in conjunction with the selected digging method. Once the suitable digging method has been selected for a dragline operation, the next step is then the definition of those factors which will influence or control the strip mine. In this stage the thickness of overburden assignable to a dragline and a strip width which suits the selected method are both established. The end result of this strategic planning phase is to establish the strip layout, the mining rate and accordingly the mining sequence.

Figure 3.1- A general view of a multi-seam strip mine in the Hunter Valley area of NSW.

3.2 STRIP MINE PLANNING

The basic aim of mine planning is to establish mine layouts and schedules that allow a given operation to be optimised. In selecting optimum strategies for a dragline operation, there are a number of options available and normally each option results in a different set of design requirements. Planning for dragline applications is a multi-task activity involving several procedures and decision variables. Some of the more important procedures and key components involved in planning of a strip mine are depicted in Figure 3.2. These procedures and the design parameters for a dragline operation have been discussed by Aspinal (1979), Stefanko (1983), Aiken and Gunnett (1990), Fourie and Gerald (1992), Hrebar (1992), Humphrey (1990), Morey (1990), Westcott, Ryder and Thrift (1991), Runge (1992), Sengupa (1993) and White and Jeffreys (1993).

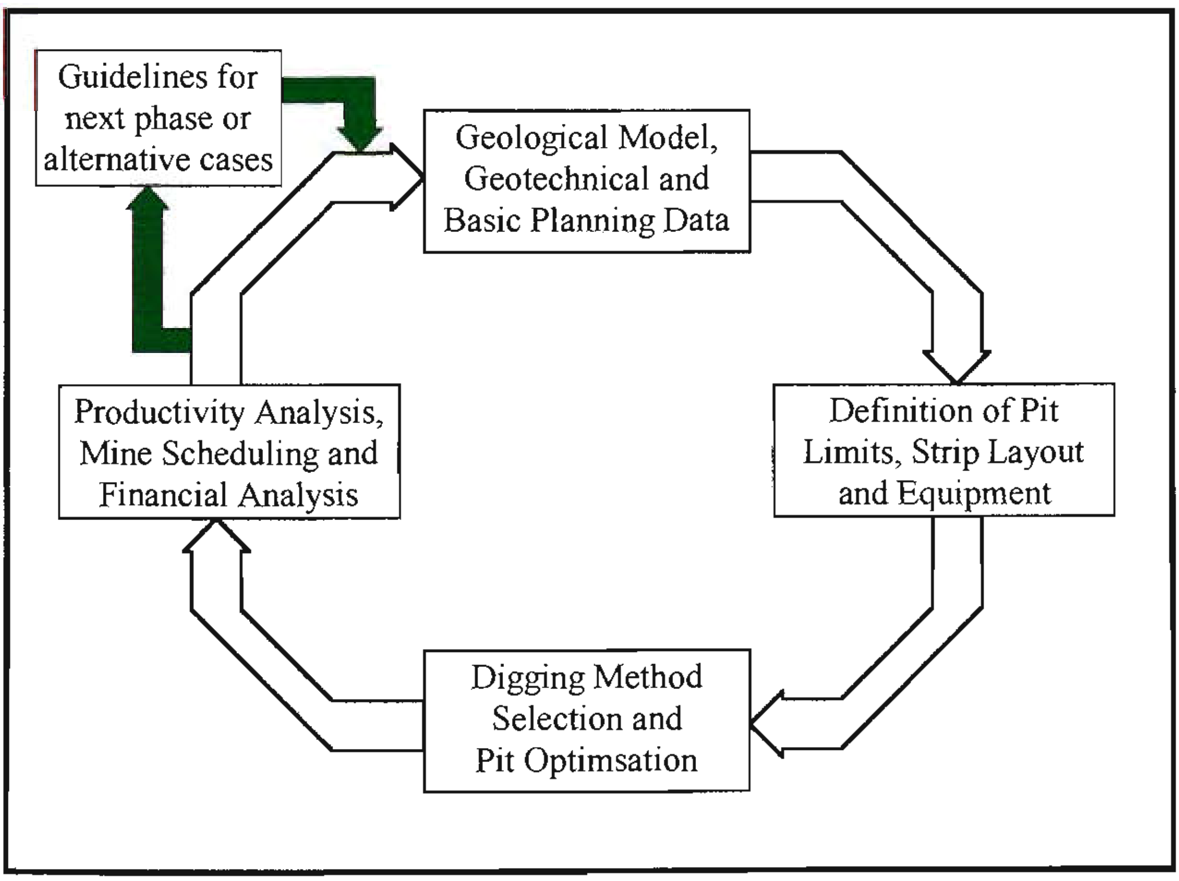


Figure 3.2- The mine planning process.

The most important problems associated with planning of a strip mine are:

1. equipment selection and sizing,

2. selecting the appropriate and most productive system for the digging method from the various alternatives,
3. sequencing of the operations in a suitable manner,
4. optimising the strip layout and pit configurations to achieve maximum dragline efficiency and productivity,
5. mine scheduling to determine the extraction sequence and resource requirements over the mine life, and
6. capital and operating cost estimation.

The planning tasks, such as mine design and equipment selection, should be repeated until an "optimum" solution is found. As stated by Westcott, Ryder and Thrift (1991), the process of mine planning is not a simple and straightforward task. In general, mine planning is an iterative process since it starts with an alternative scenario which provides sufficient information for refining the previous input data to generate a "better" alternative plan.

In the initial iteration, conceptual and long term planning processes are undertaken on incomplete data. Usually it is not cost effective to obtain a complete set of data at the initial stages of a mining project. Even with a limited amount of information, the planning process requires a complete analysis to justify the viability of future investment. The mine planning cycles only require more detailed stages as more data become available. Each planning cycle must provide guidelines for the direction of the next step. A mine planning cycle must also give valuable information on the type and level of data required for further steps, for example location and the pattern of the subsequent drill holes.

There are separate classes of mine planning process, each involving the same tasks but with different levels of detail. Mine planning starts at the pre-feasibility stage of a project and continues right through to the end of the project life. There are therefore several stages of mine planning including: conceptual mine plan, various intermediate design phases, final design for feasibility study and detailed scheduling for optimum equipment usage and production. There is no clear definition or standard on the terms used in classifying the different types of mine planning. This is understandable since the mine planning process is unique to a particular mine and cannot be generalised (Asmady, 1993).

3.3 ELEMENTS OF STRIP MINE DESIGN

The basic requirement of any strip mine planning is a fundamental understanding of the deposit. There must be sufficient information on geology and nature of the coal seam(s) including the topography of the area. This information is further used to develop a geological model of the deposit. The geological model is developed, usually with an aid of a computer, to provide a clear three dimensional representation of the deposit which describes the relationship of the various features of the deposit. The geological model must be easy to use and accurate since it is used as a basis during different stages of the mine planning process.

3.3.1 Assessment of Mining Boundary and Limits

Starting with a geological model, the next step in a strip mine planning process involves an assessment of the mining limits and boundaries. This requires development of an initial conceptual mine plan based on estimated reserves within the defined area. The principal purpose of establishing mining limits is to estimate the total mineable ore reserve. A knowledge of the final pit configuration is also important for the planning of tail dumps and surface facilities. Definition of the final pit limit is a function of physical and economic constraints. Parameters defining mine limits can be categorised into two major groups, physical and economic. The economic based limits are not easy to define compared with the physical limits (Runge, 1992).

3.3.1.1 Physical Limits

The physical limits are constant and once defined do not change over time. The process of assessment of the physical mine limits starts from the most readily definable limits and proceeds to the more difficult ones.

Some typical physical limits as defined by Runge (1992) are:

- lease boundary,
- limits associated with the local infrastructure such as adjacent towns, railways, and main roads,
- clear geological limits such as outcrop or subcrop zones and boundary faults,
- clear topography limits such as major watercourses, and
- environmental, historical and social constraints such as aboriginal sites.

3.3.1.2 Economic Limits

The purpose of the economic limit assessment is to determine which components of the potential coal property, lying within the physical limits, can be economically mined. The economic factors affecting open cut mine design are variable and change over time. Most of the economic factors are not clearly definable in the early years of the project and must be estimated over the life of the mine.

The most important factors affecting the economics of a surface coal mining project are:

- waste removal costs including operations such as drilling, blasting, etc.,
- coal mining costs including haulage costs,
- preparation costs,
- overheads and administration costs,
- out of mine costs such as rail and port charges,
- financial factors such as interest rate, taxation, royalties, and
- coal price and revenue per tonne of product.

Conventionally, all the above economic factors are condensed into a simple formula generally referred to as the *Strip Ratio* as used in this thesis. This defines a minimum acceptable profit per tonne of coal recovered. An iso-line method is normally used to develop stripping ratio maps. The iso-line method involves construction of contour lines of equal value of overburden and coal thickness (Hrebar, 1992). A stripping ratio contour map can be easily developed from structural grids in a geological model. This map is originally a grid which can be generated using simple mathematical functions between the grids. Assuming three grids representing topography (*TOPS*), coal seam roof (*SR1*), and coal seam floor *(SF1)* the thickness grids can be developed first by subtracting the surface grids (Figure 3.3). The next step is to generate a stripping ratio grid *(STRATIO)* by dividing the overburden thickness grid over the coal thickness grid as follows:

$$OBT = TOPS - SR1 \quad and \quad COALT = SR1 - SF1$$

$$STRATIO = \frac{OBT}{COALT}$$

where: *OBT* = overburden thickness grid,

COALT = coal thickness grid, and

STRATIO = grid representing the stripping ratio values.

This stripping ratio grid may be plotted in the form of contours or colour shading maps for planning purposes.

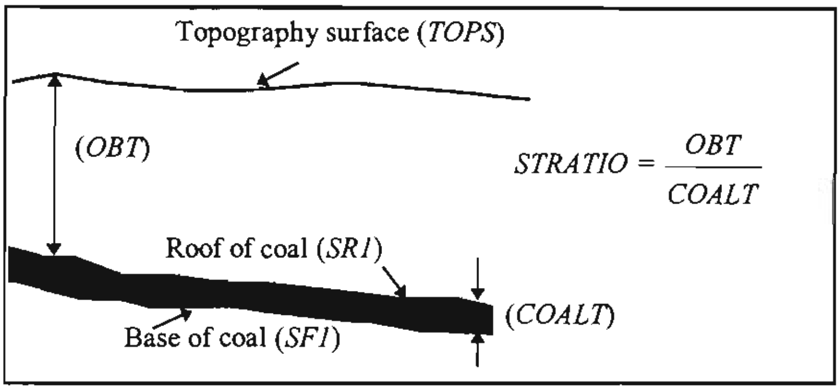


Figure 3.3 - Concepts used to develop a stripping ratio grid.

Stripping ratio maps help the mine planner to appreciate both the economic and physical limitations of the mining area. Such an approach defines the maximum allowable stripping ratio and defines the area which can be mined economically (Figure 3.4). The northern and eastern boundaries in Figure 3.4 define the mining lease while the western side is limited by a major fault. In this example the southern side of the area is characterised by relatively high stripping ratios in excess of 12:1. In other words if a maximum stripping ratio of 12:1 is used as the economic limit for an open cut mine using a dragline stripping method, only the northern part of the lease is viable.

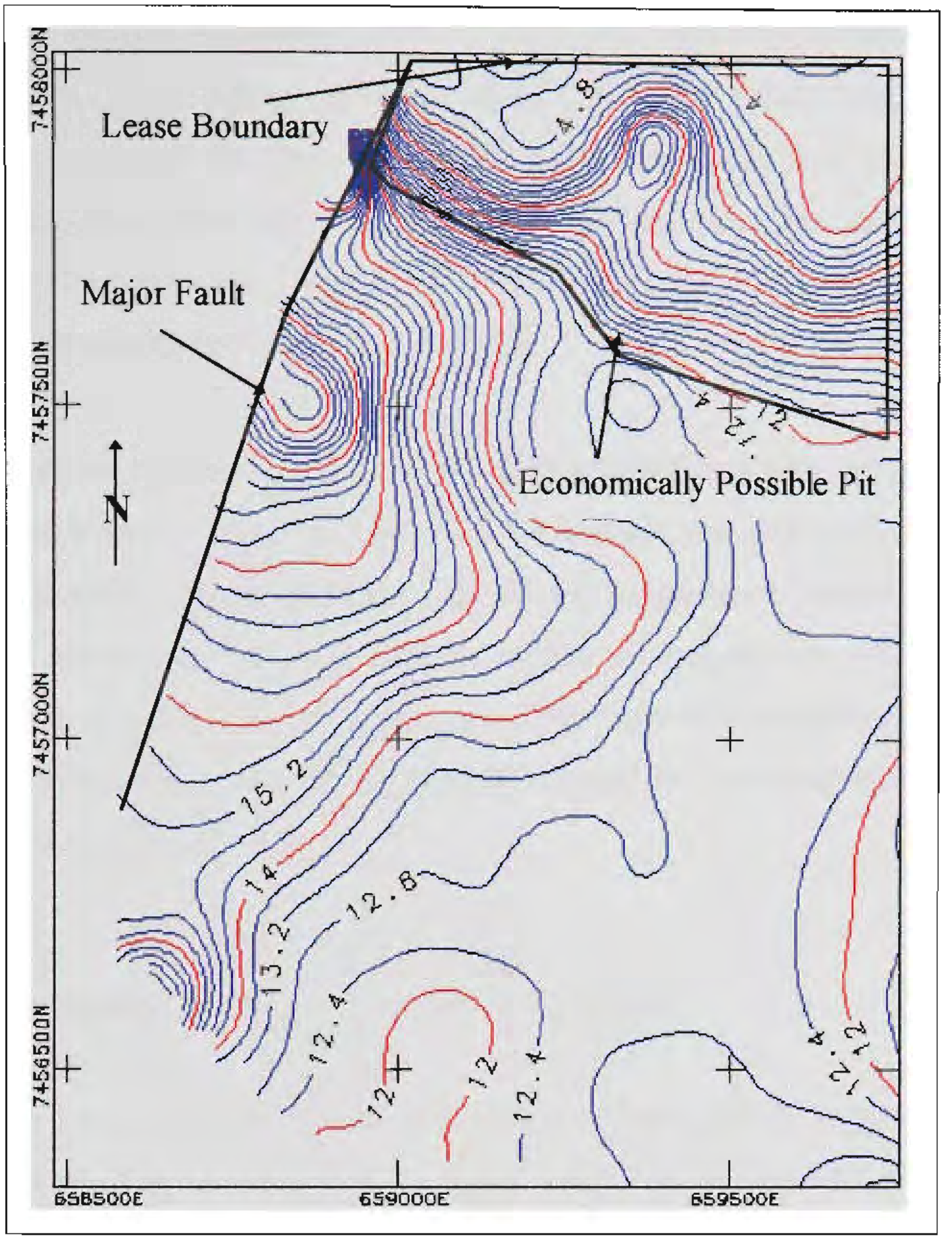


Figure 3.4- A stripping ratio map.

3.3.2 Pit Layout and Orientation

In a normal strip mine operation, the dragline removes the overburden in strips to uncover the coal. The common layout of dragline pits is a series of parallel trenches that are relatively long and narrow. The starting point for a detailed design of a dragline mining system is to generate a layout of strips for the duration of the mine life. This layout is initially a function of the ultimate pit limit, the geology of the coal and economic constraints. The basic aim of a pit layout design is to maximise rate of coal exposure while considering safety and environmental aspects. In practice, the initial pit starts from where the coal is shallowest, within the limits imposed by the lease boundary. From a financial point of view it is also preferred to start mining from where the coal is shallowest (Runge, 1992). However, some operations have reported advantages for an averaging approach in hilly environments (Chironis, 1986). This is an effort to "average" the overburden depth within each strip and keeping a constant average value through the life of the property. The equipment selected must be capable of handling the maximum design depth from the initial phases of the operation to the completion of mining activities.

Most strip mines commence their coal mining operation near an outcrop, if applicable, and progress down on the dip side. Using a comparative cost study, Hrebar (1990) showed the advantages of this approach. Such an approach causes an incremental increase in overburden depth for each successive pit and perhaps additional stripping units, such as a truck and shovel system would be required in the later years of the mine life. With this design the initial cut, termed "boxcut", is laid out along the strike of the subcrop of a coal seam.

3.3.2.1 Comparison of Straight and Curved Pit Shapes

When strip mining is started along a coal outcrop and within an area with a rolling topography, the dragline pit may be designed so that the pit follows a uniform contour. As a result, this type of design may result in pit being developed in a curved shape. Where curved pits are designed, a series of inside and outside of curves are usually encountered. The major problem with this kind of design is the difference in available area for spoiling between an inside and outside of the curve. Outside curves, where the spoil side arc length is greater than advancing highwall arc length, provide more spoil room. This extra spoil room can be used in cases where overburden depth increases significantly in the mining advance direction or in the vicinity of coal access ramps. On the other hand, the inside curves may cause significant spoil room problems, depending on the depth of overburden, strip width, radius of curvature, and operating parameters (Morey, 1990). Insufficient spoil room increases both the dragline cycle time and the volume of bench rehandling, hence a decrease in the dragline productivity. In practice, curved pits are difficult to lay out in the field and to implement from an operating standpoint. Owing to the problems associated with the curved pits, many operators prefer to develop straight pits. A curved pit can be straightened by designing a series of short pits on the cord of an outside curve. In many cases, auxiliary equipment such as a scraper fleet may be used to strip these short pits (Hrebar, 1992).

Once the pits are oriented, the next step to be considered in the design of dragline pit geometry is to determine the pit length. Normally longer pits are preferred as they provide a better coal inventory. Dragline efficiency also increases as pits get longer due to the less walking requirements and reduced rehandling around the ramps. Coal haulage systems may be affected adversely by the use of long pits due to increased truck travelling time. If spoil stability is a function of time, shorter pits are preferred to limit spoil failures. In the Australian context typical pit lengths are between 600m to 3000m.

3.3.3 Dragline Size Selection

Morey (1990) categorises key parameters influencing mine design into three distinct groups: geological, equipment specifications, and operational parameters. In this classification, the geological parameters governing strip mine design are overburden and interburden depth, thickness of coal seams and partings, swell factor, and repose angles of bench and spoil material. In practice these parameters cannot be controlled and the strip mine must be designed to meet the geological constraints. However, the mechanical parameters of the major equipment being used must meet the constraints of the geological parameters. The design of mine equipment is generally based on standard specifications with limited flexibility.

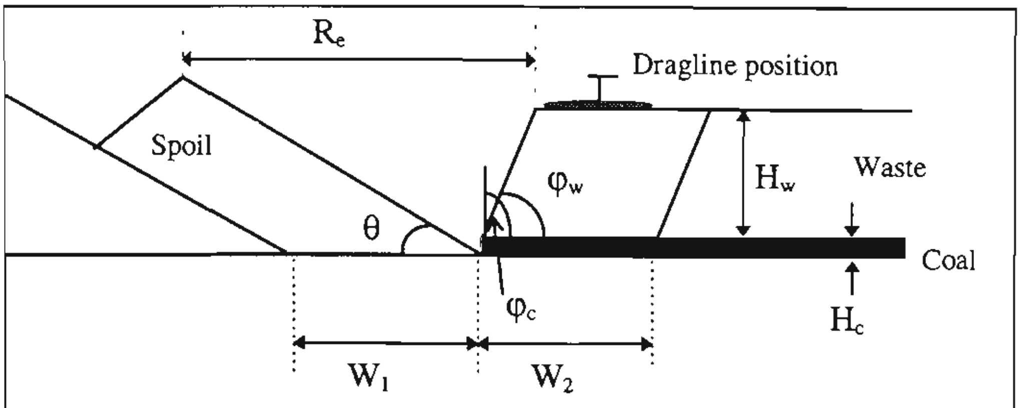
There are two different approaches in the design of an open cut coal mine using a dragline as the major overburden removal unit. The first approach is to set a production target and then select a dragline to satisfy production requirements. However, the majority of operations may already have their equipment in place. In this case both the strip geometry and the digging method would be highly dependent on the dragline available (Hrebar, 1990). Typical cases are expansion of an existing operation using the same dragline, moving an existing dragline from another company site and purchasing a dragline from a worked out mine. In such environments, the dragline specifications will affect the sequence and configuration of the strips.

Draglines have more design flexibility than shovels. Dragline boom length and boom angle can be varied within specific limits. However, the maximum suspended load will vary as boom length and boom angle are varied. When selecting a new dragline, the selection process is based on the maximum dragline dimension required for the excavation and spoiling operations and the production requirements of the mine. The primarily dimensions which must be considered when selecting a dragline are (Sweigard, 1992):

1. dragline reach,
2. maximum dig depth capability,
3. maximum dump height,
4. bucket capacity, and
5. swing speed, hoist and payload speed.

The first three factors are dependent on the stripping method, geological and geotechnical parameters and the proposed pit configuration, such as final pit depth and strip width. The last two factors are controlled by the target production rate. Since the geological factor, and hence the digging conditions, are variable during the dragline life, the required dragline specifications must be matched to a "worst case" scenario.

Traditionally, there are two main methods to estimate the required dragline size. The first method uses 2D range diagrams. The required dimensions can then be measured directly from the scaled diagrams. This method is relatively flexible and provides a basic understanding for the combination of pit configurations and the size of the dragline being selected (Sweigard, 1992). The second method uses an analytical approach to compute the required dimensions of the dragline, particularly the effective dragline reach. In this method, mathematical relationships for relevant variables have been developed for the particular stripping method. Such mathematical relationships have been described by many authors (Stefanko, Ramani, and Freko, 1973; Charles et al, 1977; Hrebar and Dagdelen, 1979; Stefanko, 1983; Humphrey, 1990). For example, the effective dragline reach (R_e) can be calculated for a Simple Side Casting method as shown in Figure 3.5.



$$R_e = \frac{H_o}{Tan\varphi_w} + \frac{H_c}{Tan\varphi_c} + \frac{W_2 \times H_o \times SF}{W_1 \times Tan\theta} + \frac{W_1}{4}$$

SF = Swell Factor, (normally 1.3)

Figure 3.5- Dragline effective reach calculation for a Simple Side Casting method.

The bucket capacity required can be determined by considering the annual coal production requirements and an estimation of the coal uncovering rate. A simple method to determine the bucket capacity is the use of historic production indices. The required bucket capacity is then the product of the average regional index and the annual stripping requirements. The volume of overburden removed by each cubic metre of bucket for Australian draglines averages 2.0 - 2.5×10^6 (Runge, 1992).

Another method for determining bucket capacity is a mathematical approach which uses the standard excavator sizing equation with an adjustment for rehandle (Hrebar and Dagdelen, 1979):

$$B = \frac{O \times (1+R) \times (1+S) \times C}{BF \times SH \times M \times J \times 3600}$$

where: B = Bucket capacity (bank cubic meters),

O = Annual stripping requirement (bcm/yr),

R = % Rehandle, (typically varies between 20 to 60%)

S = % Swell, (varies between 20 to 35%)

C = Cycle time (sec), (around 60 seconds)

BF = Bucket fill factor, (varies between 0.9 to 1.05)

SH = Scheduled hours (hrs/yr), (varies between 6500 to 7000 hrs)

M = Mechanical availability, (varies between 75 to 90%) and

J = Job factor (varies between 0.8 to 1.2).

In the above equation the annual stripping requirement is dependant on the stripping ratio. For example with an average stripping ratio of 8:1 (bcm of waste per tonne of coal) 16 (Mbcm) must be removed annually to produce 2 Mt of row coal.

Using either of the above methods, the computed bucket size must be converted into a factor termed *maximum suspended load* (MSL). The MSL specifies the maximum allowable weight of the loaded bucket considering the weight of the bucket and the material being dug, which varies with overburden type. MSL is expressed as:

$$MSL = required\ bucket\ capacity\ (m^3) \times [bucket\ density(t/m^3) + swelled\ material\ density\ (t/m^3)]$$

For an example, for a calculated bucket capacity of 50 m^3, loading material of loose density of 1.3 t/m^3 and bucket density of 1.7 t/m^3, the MSL is calculated as follows:

$$MSL = 50 \times (1.7 + 1.3) = 150\ tonnes\ (\approx 330{,}000\ lbs)$$

The calculated dragline size and MSL must be compared with the specifications of existing models provided by the manufacturers (Bucyrus Erie Co., 1977 and Humphrey, 1990). In these charts various dragline reach factors are graphed against the MSLs for all the existing models (Figure 3.6). Using the dragline selection chart for the example presented above (330,000 lbs), the appropriate model can be machine number 45 if a reach factor of at least 250ft (76m) is required. Table 3.1 is a part of the Bucyrus Erie standard machine selection table and shows the dragline specifications for the various sub-models of a 1570-W walking dragline.

In addition to the above considerations, there are other factors which must also be considered such as the maximum dragline digging depth and dump height, swing angle and hoist times for each specific dragline. Generally, these factors are not as critical as the bucket capacity and the dragline reach factor. Other aspects worth considering include the manufacturer's ability to provide support on repair services, variable boom and bucket size features for a specific model, gradability and walking capabilities (Pundari, 1978). For example, availability of the various boom and bucket sizes for a specific model allows a mine to change its dragline specifications as the geological conditions change. Also a dragline with higher walking gradability is preferable since it needs less earthworks for access ramps.

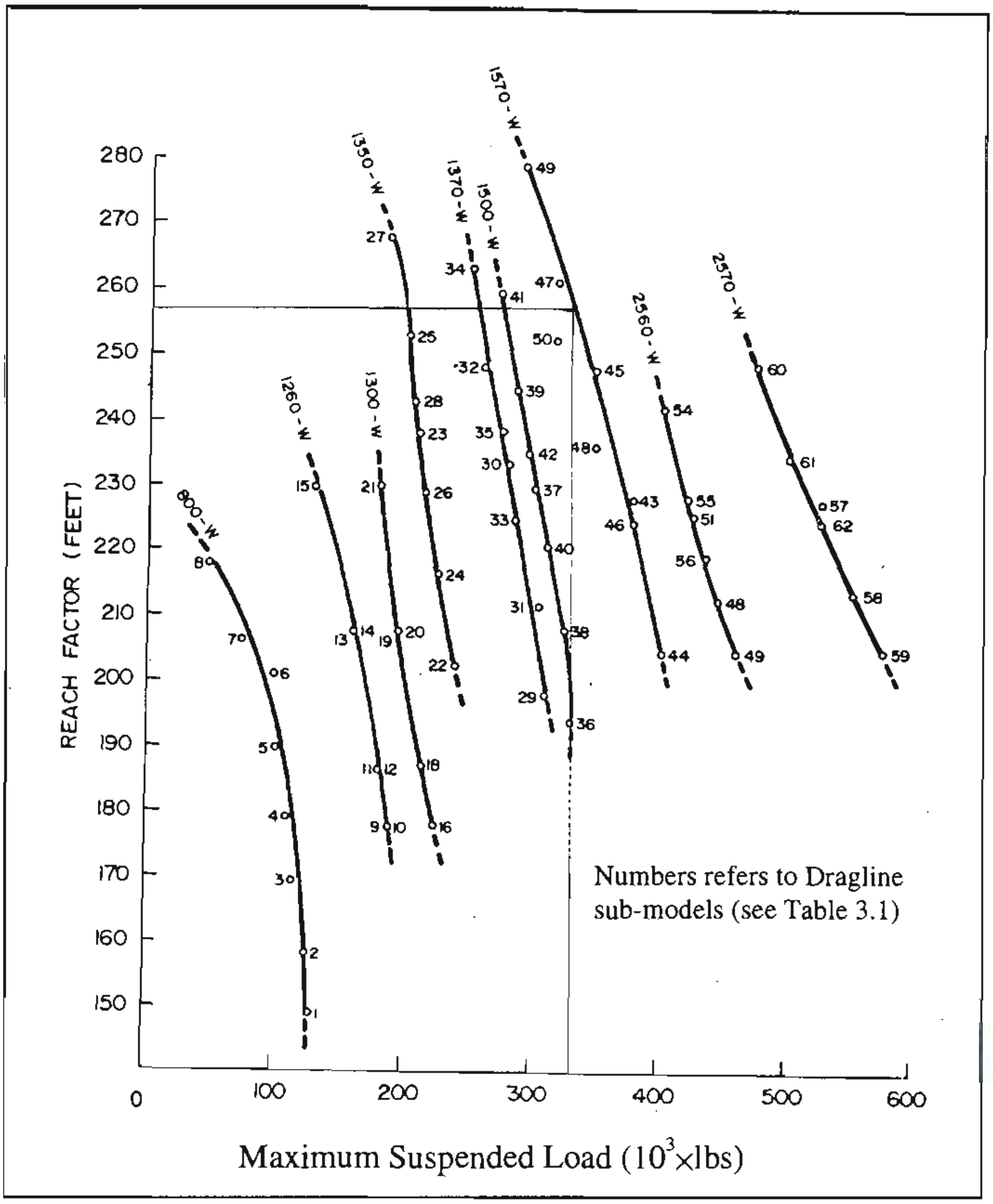


Figure 3.6- Dragline standard machine selection chart (after Bucyrus Erie Co., 1977).

Table 3.1- A part of the standard machine selection table (after Bucyrus Erie Co., 1977).

Reference Number	*Boom Length (ft)*	*Boom Angle (deg)*	*Operating Radius (ft)*	*Reach Factor (ft)*	*MSL ($10^3\times lbs$)*
43	285	30	277	227	375
44	285	38	254	204	400
45	310	30	298	248	345
46	310	38	274	224	375
47	325	30	311	261	315
48	325	38	286	236	345
49	345	30	329	279	285
50	345	38	302	252	315

3.4 COMPUTERISED DRAGLINE MINE PLANNING SYSTEMS

Large walking draglines are very capital intensive pieces of equipment. A new medium size dragline may cost up to $A60×10^6. To maximise the return on investment and to improve the performance of a dragline, its mode of operation and influencing parameters must be fully understood and analysed with the view of optimising the process. Finding the normal working ranges for a given dragline and optimising its operation usually requires that various possible mining scenarios and pit configurations be assessed and compared with each other. The optimising process is normally both difficult and time consuming due to the a large number of parameters involved. This process requires a repetitive arithmetic and analytic solution which is ideally suited to the application of computer simulation methods. For the last two decades various software packages have been developed to simulate different aspects of a dragline operation. A computer software package aims to facilitate the tedious and repetitive aspects of the dragline based mine planning process. Usually computer models use mathematical, graphical and analytical techniques to solve two common problems in the planning and design of a dragline operation. These problems are the selection of a suitable dragline for a given digging method and pit geometry and the selection of a suitable digging method together with optimised pit geometry for a given dragline and geological condition. Some of the simulation models and commercial software developed for an open cut dragline operation are reviewed below.

3.4.1 Computerised Dragline Simulators

The selection of a dragline is a major and critical decision during the strip mine planning phase. As emphasised by Humphrey (1990), when sizing or selecting a dragline for a new operation an important concept to keep in mind is to select the dragline for the strip mine plan, not the strip mine plan for the dragline.

One of the first attempts to derive the equations required for a dragline selection procedure was carried out by Rumfelt (1961). He used MUF (maximum usefulness factors) in conjunction with the geometry of the pit to select the most suitable dragline for overburden stripping. MUF for a dragline is defined as the product of the nominal bucket capacity and the dumping reach of the dragline. He also developed computer programs to analyse the relationship between the pit geometry (ie. overburden depth, strip width, etc.) and MUF.

Hrebar and Dagdelen (1979) reported a computerised simulation method for dragline selection. In their simulator, they took the conventional approach for reach and bucket capacity determination and modified it using a three dimensional approach to correctly calculate the dragline selection requirements. The study showed that dragline reach requirements are underestimated when using a conventional two dimensional approach.

Chatterjee (1980) developed a computerised strip mining model which could be used in selection of a suitable dragline and pit geometry for a standard Extended Bench method. The model incorporated a volume concept (rather than area calculations in a two dimensional approach) and spoiling was computed on the basis of dragline position and the available void in the spoil area. Chatterjee's model considered the digging position of the dragline and included operational factors such as the digging depth and the obliquity of the boom with respect to the general face line in evaluating the digging efficiency of the dragline.

Gibson and Mooney (1982) applied a mathematical programming (non-linear programming) technique for selection of a suitable size of dragline. The objective functions were to minimise the time for overburden removal and the cost per tonne of coal removed. The constraints of the non-linear programming equations were dragline reach, working space requirements and interactions of dragline size and pit geometry characteristics. The program was designed to be incorporated in a more comprehensive surface mine and reclamation planning package, "SEAMPLAN". They used a CAD (Computer Aided Drafting) approach to the traditional dragline sizing problem. Sharma and Singh (1990) also reported developing a computerised model to select a suitable dragline for a given mining operation. Their computer model used the conventional reach and bucket size calculations and expanded these formulas to a greater depth to consider factors such as the effect of blasting on the swell factor and repose angle.

Hrebar (1990) developed a comprehensive computer based costing model to cope with the dragline selection problem. The model incorporated capital and operating costs to select the most cost effective dragline for a given operation. The model, unlike the previous models, considered the changes in production rate as a result of changes in the depth of overburden. Using overburden production rate versus depth data for each dragline, Hrebar's model calculated machine requirements and costs for a series of draglines. The most cost effective dragline for a given mining sequence could then be selected on the basis of present worth of after-tax cost. Using a case study, Hrebar showed that the dragline productivity may vary by 50% or more over a typical range of machine-digging depths.

Method selection and pit optimisation processes have traditionally relied on graphic and analytical approaches. One of the fist attempts to simulate a dragline operation was reported by Nikiforuk and Zoerb (1966). They reported developing an "analog" computer simulation model which could be used for investigation of performance of different movements of the dragline and its bucket, e.g. swinging, hoisting and dragging actions.

During the 1970s, the US Federal Government funded computer programs and simulation models, including costing, equipment selection and pit optimisation software (Stefanko, Ramani, and Freko, 1973; Ford, Bacon and Davis Inc., 1975; Ramani, Igoegbu and Manula, 1976; McDonnell Douglas Co., 1978; Sadri and Lee, 1982). All of these computer programs were written to simulate single seam operations and could be run only on main frame computers. Due to their hardware restrictions and lack of graphical interfaces, these programs are not widely used (Hamilton, 1990).

White and Jones (1984) reviewed and compared seven of these programs and modified them to run on the IBM 370 personal computer. Among these computer programs Fluor's program was the most useful and versatile one for modelling a dragline operation. Fluor's simulator required much less input data and included a geological model in a form of topography and coal seam data. The software required three elevations for each X, Y point in the topographic surface, and top and bottom of the coal to be entered one at a time. The program could also generate a preliminary 3D output of the simulated pit (Sadri and Lee, 1982).

Chatterjee, Rowland and Siller (1976) developed a dragline simulation model which was able to establish a cut-off point of the cost-depth ratio and predict the most economic method of overburden stripping for a given geology. The advantages of the model, compared with the previous ones, were that it considered the operation as a three dimensional procedure for the first time and also eliminated the assumption of a ninety degrees swing angle to spoil material. Varcoe (1984) reviewed Chatterjee's work and modified the output of the simulator to be presented in graphical form.

After Chatterjee's work, the next generation of computerised simulation models was developed by a number of authors (Bandopadhyay and Ramnai, 1979; Mooney and Gibson, 1982; Sadri and Lee, 1982; White and Nesz, (1983), Williams and Shanks, 1984; Lee, 1988; Michaud and Calder, 1988; Stuart and Cobb, 1988). These simulators applied a wide range of analytical and graphical techniques to facilitate dragline mine planning. One of the latest simulators developed for dragline strip mine planning was reported by Michaud and Calder (1988). They applied computer technology developments in the areas of three dimensional graphics and geological modelling to develop an interactive computer graphic software to assist with dragline strip mine planning.

Recently several authors have attempted to use expert system technology to select dragline mining method. Two high-level languages, "LISP" and "PROLOG" are commonly used to design expert system applications to mimic human thinking (Chironis, 1987). Stuart and Cobb (1988) developed an expert systems using a "TURBO PROLOG" programming language to help the user to find multiple feasible solutions. The program lists all possible mine design scenarios that meet the criteria which are selected by the user. Wu (1990) developed an expert database which could be used in selection of the optimal dragline mining method. A hybrid approach was applied to combine a discrete simulation model and a knowledge database. Instead of the rule-base expert system, a logic-base expert system was chosen for consultation to select the optimal method of stripping by dragline. The current trend toward development of expert system applications is likely to continue, but a danger exists that an expert system may actually limit the creativity of an engineer when designing a modified or new type of digging method (Hamilton, 1990). Besides, as every mining situation is physically unique, it is quite difficult to gather all feasible methods in one software package.

Strip layout and pit geometry optimisation is an important area in dragline mine planning. Feasibility analyses, long-range planning and the assessment of the capital costs are influenced by the results of the optimisation of the pit configuration. Operation research techniques have been used by some authors to determine the best dragline and pit configuration (Dunlap and Jacobs, 1955; Gibson and Mooney, 1982; Mooney and Gibson, 1982; Rodriguez, Berlanga, and Ibarra, 1988). Because of the complexity of the dragline operations and the number of parameters involved, a realistic "best" solution cannot be obtained with only the use of mathematical techniques. *Heuristic optimisation technique* is one of the methods most widely used in simulation models to optimise various pit design parameters and dragline specifications. The technique is simply to repeat the simulation runs for the alternative configurations, say width of the pit. To distinguish the most desirable configuration of each parameter the changes in the design or operating policies of the system are compared for each of the simulation runs.

3.4.2 Commercial Dragline Simulation Software

Numerous commercial computer packages are available to the mining industry providing comprehensive integration of various modules to handle all aspects of the strip mine planning process. Most of the integrated mining packages have special modules to simulate the dragline operations. Almost all the dragline simulation packages available provide some sort of volume calculations and rehandle estimations. Some of these systems also compute the preliminary machine productivity for the simulated operations. Optimising procedures in these systems are based on a trial and error process.

Two mathematical approaches may be taken in a 2D range diagram calculation. These are trigonometric and CAD approaches. While a trigonometric calculation is easy to program, it has limitations in terms of access to real geology data and the geometry of the cut and spoil profiles is generally oversimplified. Most of the recently developed software uses the CAD approach as this provides more flexibility for the dragline simulation model to be part of an integrated strip mine planning system. In a CAD based software, closed strings are used to generate polygons of cut and spoil areas. The area of the closed strings can then be computed to provide the required volumetric calculations.

3.4.2.1 DAAPA

The *DAAPA* (Dragline Analysis And Productivity Assessment) software is a product of Runge Mining (Australia) Pty Ltd. The basic application of *DAAPA* is to assess dragline rehandle, operating limits and productivity for a simple dragline operation. *DAAPA* uses the dragline specifications and the average geological data to establish the pit configuration for a standard Extended Bench with an advance chop operation. Up to three coal seams can be simulated by the software and lowwall pass operations are allowed from the second pass. The software displays a 2D range diagram of the sequence of operation, and calculates theoretical dragline rehandle quantities (Figure 3.7). A preliminary estimation of the machine productivity is also calculated by the software.

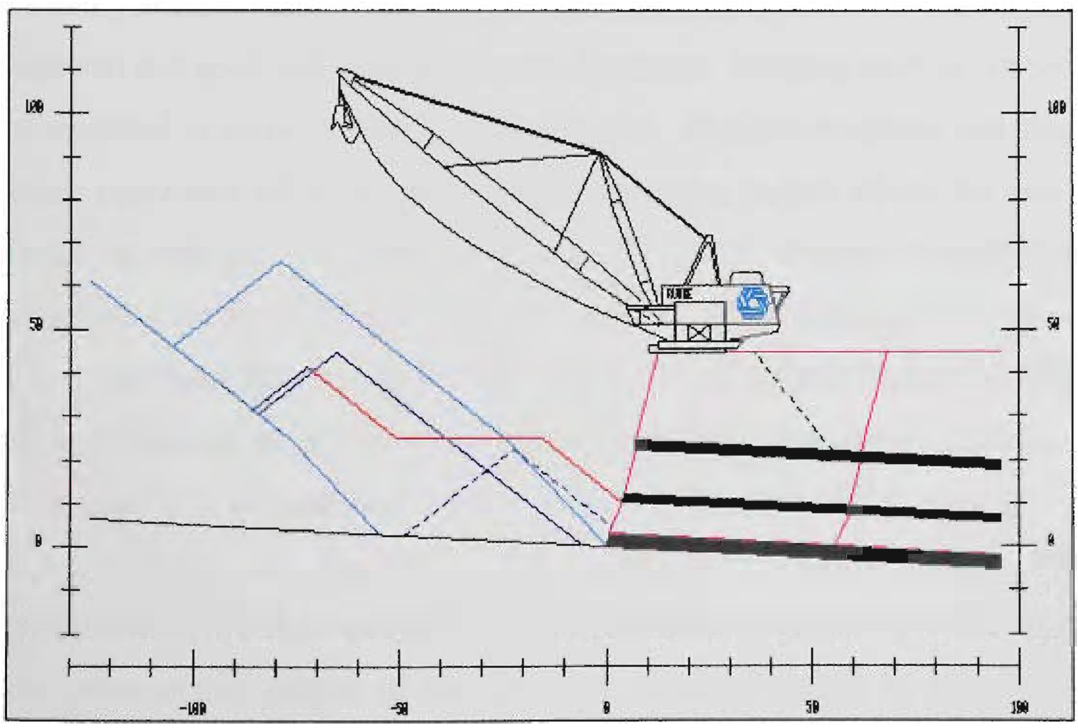


Figure 3.7 - A sample 2D range diagram output of *DAAPA* (V-3.1).

The software *DAAPA* is only suitable for rapid analysis and comparison of very simple dragline operating alternatives in the early stages of strip mine planning process. *DAAPA* can not optimise the operational parameters such as strip width or dragline working level from a single run.

3.4.2.2 DRGX

DRGX is a product of Engineering Computer Services International Pty Ltd (ECSI) and is a part of an integrated open cut mine planning package called "*APOLLO*". The package is run on graphic workstations and UNIX-PC environments. The software computerises 2D range diagrams in cross sections. *DRGX* allows the user to define the geology of simulation sections from three different sources as follows:

1. digitising (from screen or a digitising table),
2. using a template option which has been pre-designed for up to six coal seams, and
3. accessing information from a geological modelling system.

DRGX is a multi-section and multi-strip dragline. Unlike *DAAPA*, the software works interactively from the screen and the user can define the position of the dragline so that the highwall and spoil side methods can be simulated. Existing spoil or cut profiles can be incorporated to allow simulation of active pits. Multiple draglines and simple truck and dozer operations are supported by *DRGX*. A replay facility allows the user to save a particular cut and spoil sequence for later screen replay. Outputs from *DRGX* include cross-section plots at any stage of the operation, volumetric and rehandle calculations, and swing and hoist information reports. Simulation of cut and fill procedures in *DRGX* cannot be automated, so it is not suitable for optimisation purposes particularly when the whole deposit is to be simulated. Any optimising process in *DRGX* must be carried out through trial and error. The simulation results must be further analysed using other computer packages such as spreadsheets to estimate the productivity of the operation. A sample cross-section output of the software, showing details of the cut and spoil procedures, is presented in Figure 3.8.

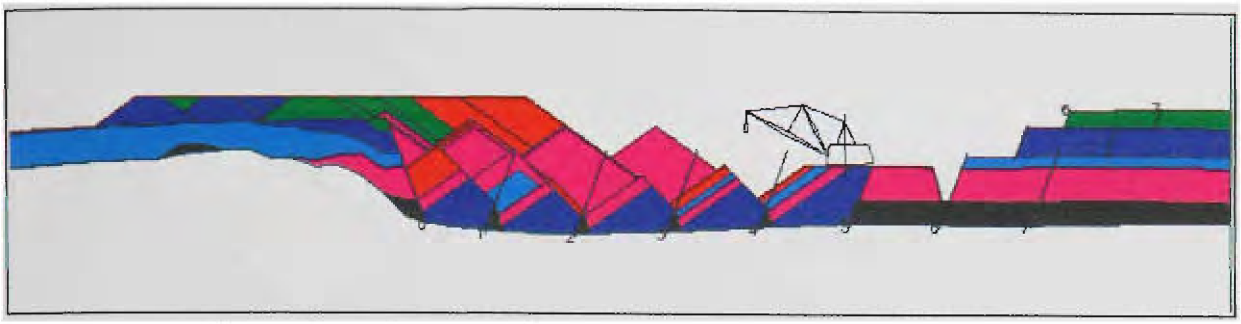


Figure 3.8 - A sample 2D range diagram output of DRGX.

3.4.2.3 Dragline (MINESCAPE - Dragline Modelling)

Dragline is dragline modelling module under an integrated mine planning system called MINESCAPE which is a product of Mincom Pty Ltd. It is a CAD-oriented module enabling engineers to define and test dragline excavation methods on a cross sectional basis. The module functions include simulation of normal cut and fill dragline processes and also simple throw blasting and dozing operations. *Dragline* accesses the topographic and stratigraphic surfaces of the deposit from MINESCAPE geological model. Sections approximate the geology determined by both pit surveys and drilling through geological models. The material strength characteristics can be assigned to each stratigraphic unit. Working in cross-section, the user interactively defines the dragline digging method as a sequence of steps through CAD functions that simulate the dragline and material movements.

Dragline produces reports of prime and rehandled material moved by a productive unit as base data for production scheduling. It also generates 3D surfaces that form the starting point for rehabilitation planning, and produces standard range diagram sections which can be optionally annotated with volumetric details. Simulation of dragline operations in *Dragline* is interactive, similar to *DRGX* software. The entire process of the dragline simulation cannot be automated, so it is not suitable for optimising purposes particularly when the whole deposit is to be simulated. A cross-section output of the software, showing details of the cut and spoil procedures, is presented in Figure 3.9.

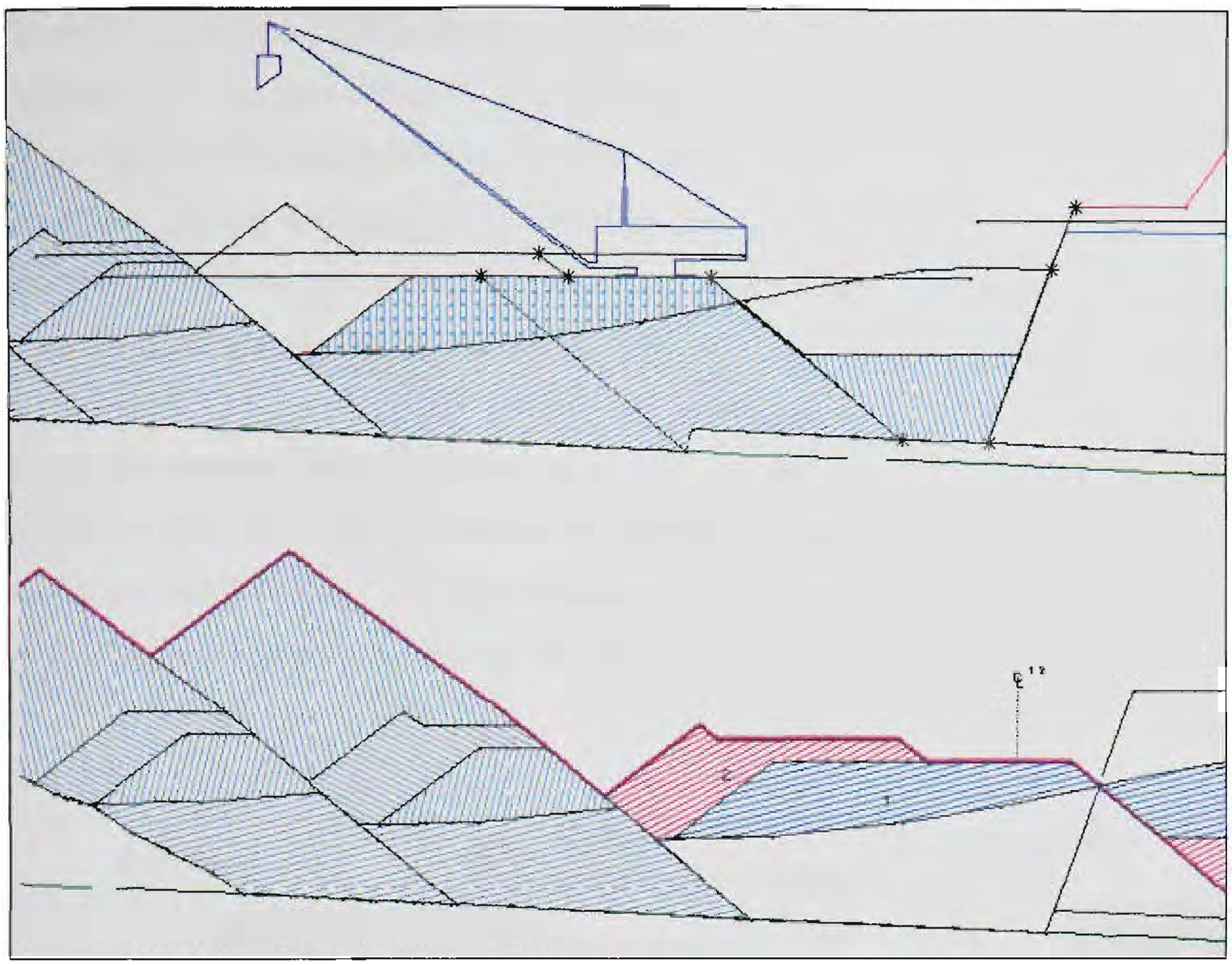


Figure 3.9- A sample 2D range diagram output of *Dragline*.

3.4.2.4 3D DIG

3D DIG is a product of Earth Technology Inc. of Australia. The package is a PC based software which interactively simulates dragline operations. *3D DIG* works with a topography grid as a reference surface and this surface is updated as cut and fill procedures progress along the strip. The software inputs include a digital terrain model of the existing topography, coal seam roof and floor data, and string data describing pit limits, toe lines, roads and any other relevant features. *3D DIG* also simulates the dragline geometry as well as swing and hoist speeds.

Simulation involves removing the cut profile in layers and dumping spoil in discrete parcels. As each parcel is dumped it is allowed to rill out to the repose angle, thus modelling the three dimensional topographic effects of dragline spoiling. The simulation of the dragline involves digitising each new position of the dragline from the screen and performing cut and spoil processes. For an example, for a dump procedure the user is required to digitise the dump location. If the digitised point is outside of the dragline reach, the user will be prompted with an error message and a new point must be defined. *3D DIG* is suitable for short term planning where the details of the dragline sequences and pit configurations are known and set already. For an optimisation process as well as running a simulation on a full deposit, *3D DIG* is very time consuming and to some degree inefficient.

While the software provides a very good 3D visualisation of the dragline pit and operations, it is not suitable for analysis of alternative mining scenarios. The simulation results are highly sensitive to the knowledge and experience of the user. A sample 3D view of the dragline pit generated by *3D DIG* is presented in Figure 3.10.

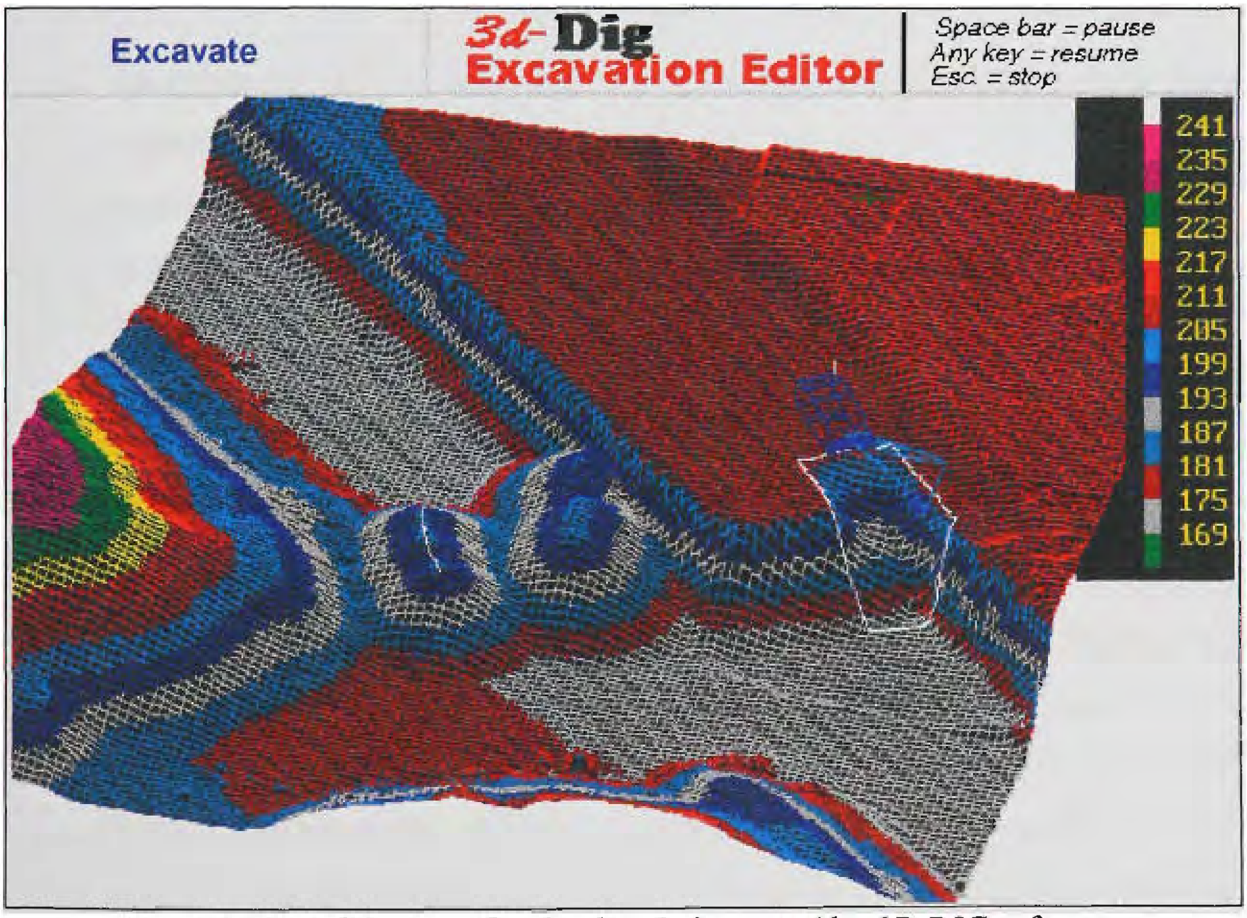


Figure 3.10 - A 3D view of a simulated pit created by *3D DIG* software.

3.4.3 Blasting Computer Modelling

Computer modelling packages which are capable of handling blasting parameters and predicting various blasting results are presently available to industry. Computer modelling in conjunction with dragline strip design analysis and cost evaluation can provide a valuable insight into potential cost savings in overburden removal, without the need for expensive and risky field trials (Sengstock, 1992). Computer based blasting models provide a scientific approach to predicting blast performance. Computer models can be used to quickly evaluate alternative blast designs, reducing the need for costly and time consuming full-scale trials. However the need for field trials cannot be avoided completely by computer models as there are always unknown factors which may alter the result of a blasting operation. Typical applications of a computer blasting model are to:

- determine the effect of changes to blast hole diameter, blast hole pattern, explosives type and initiation sequence,
- determine the effect of deviation from design (e.g. incorrect blast hole location),
- determine the effect of different rock properties including mechanical strength and structure, and
- design a blasting technique to provide a specified result such as improved fragmentation, reduced blast damage or greater thrown muckpile.

In Australia, since the late 1960's the dominant software used by the mining companies has been ICI Explosives' SABREX. SABREX (**S**cientific **A**pproach to **B**lasting **R**ock by **EX**plosive) is an integration of previous individual modules including BOBCAT, KURAN, DCRACK, SCRRACK and MICBLAST. The SABREX computer model predicts blast performance by modelling the blasting process and its interaction with the rock mass for a specified blasting geometry and design (ICI Technical Services, 1996).

3.5 SUMMARY

Mine planning of complex coal deposits using dragline stripping technique is an iterative process. In this process each phase provides some information and allows the next phase to be undertaken with a greater degree of refinement and confidence. Because of the complex nature of dragline operations, a large number of alternative digging methods are used in practice, depending on the nature of the deposit. Both the selection of the most efficient digging method and determination of an optimum working geometry of a dragline operation require a repetitive arithmetic and analytic solution. This is ideally suited to the application of computer simulation methods.

Most of the conventional dragline simulators are restricted to the simulation of standard dragline digging methods. They also require average shapes and thicknesses of overburden and coal to represent the geology. These dragline simulators ignore any changes in the geology over the mine life. In practice, the material around coal access ramp should be carried along a strip. This extra material, often rehandled, affects the volumetric calculations of the adjacent six to ten mining blocks in the vicinity of the ramp. Such an option cannot be handled by most of the available dragline simulators.

Another limitation with most of the current computer packages is that their outputs are limited to the volumetric and rehandle calculations. To select a dragline, pit geometry and a digging method the decision must be made on the basis of maximising the coal uncover rate and more importantly an analysis of costs associated with each option. To accomplish this it is necessary to consider many tasks including geological, geometric, productivity and cost calculations.

In this thesis an innovative dragline simulator has been developed which accesses the mine geological model and uses a multi-section approach for simulation to overcome the problems of the available dragline simulators. The development philosophy of this simulator is discussed in the subsequent chapters.

CHAPTER FOUR

DEVELOPMENT OF A DRAGLINE SIMULATION MODEL

4.1 INTRODUCTION

To evaluate different mining scenarios for complex geological situations, the dragline simulation must be carried out on a full set of closely spaced sections that are not necessarily similar in characteristics. Also an automatic process is preferred rather than an interactive one for the optimisation purposes where repetitive computer runs are required. The automation of the process allows more sections of closer spacing to be simulated, hence more accurate prime and rehandle volumes can be calculated.

Most commercially available dragline computer packages are limited to regular geological structures where the total mining area can be represented as a simple generalised cross section. Besides, many of the currently available packages are limited to the standard digging methods or specific mining conditions and a "black box" approach is used to determine the "best" mining parameters. This means that the user cannot follow the logic of the package and thus has no means of improving the software limitations. A dragline simulation model (**CADSIM**: **C**omputer **A**ided **D**ragline **SIM**ulator) was developed during the course of this thesis which avoids the "black box" approach.

4.2 CADSIM MODELLING APPROACH

A **C**omputer **A**ided **D**ragline **SIM**ulator (**CADSIM**) was developed during the course of this thesis. A general flow chart of the CADSIM's modelling process is shown in Figure 4.1.

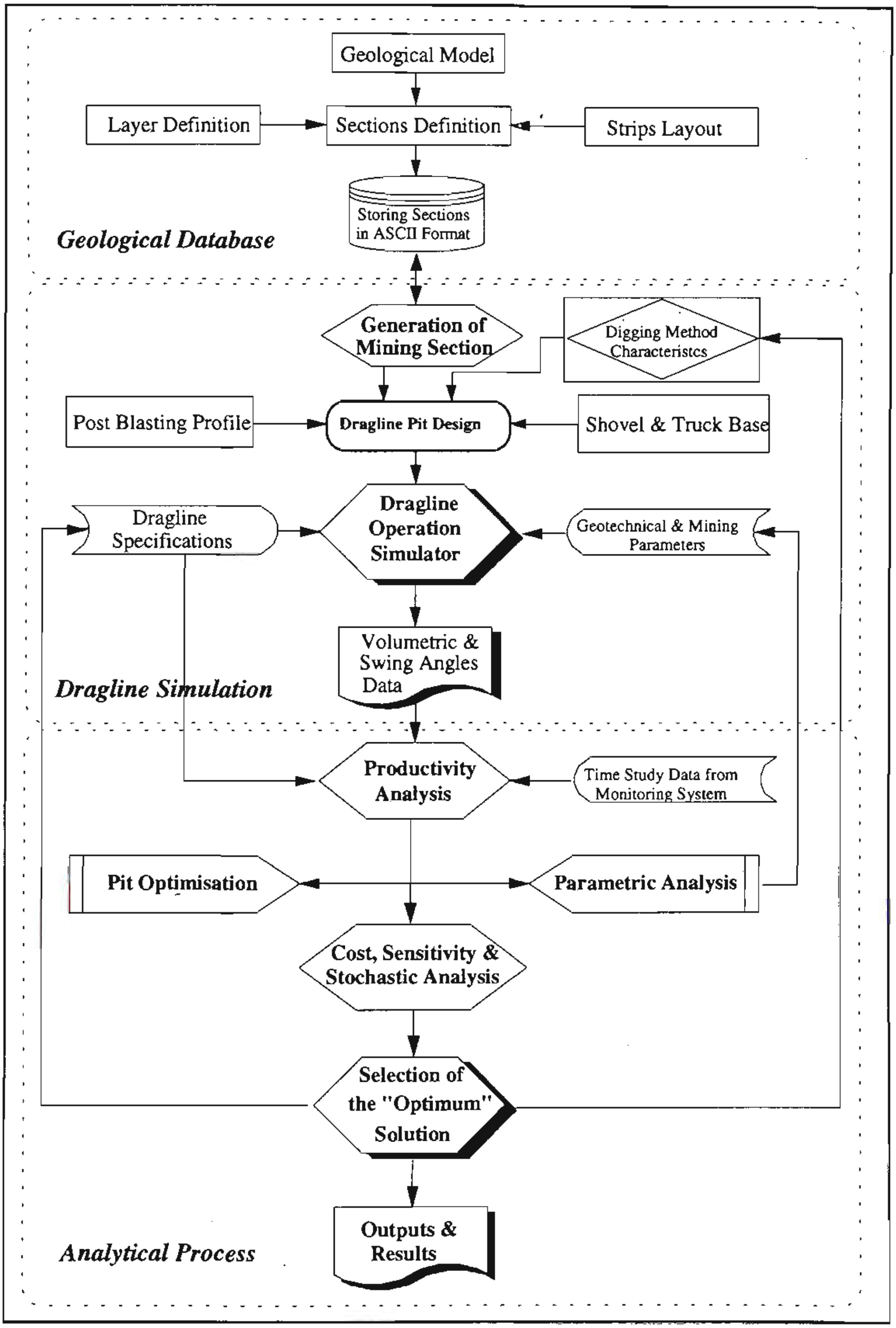


Figure 4.1- The modelling flow diagram for the CADSIM system.

Because of the iterative nature of the mine planning process, the CADSIM model was made flexible to handle different alternatives, including variations in strip layout and pit design, machine characteristics and dragline digging methods. During the development of the CADSIM model a number of criteria and design requirements were identified. The factors which were considered in meeting the original thesis objectives are:

- Making all the aspects of the planning process transparent to the user and hence avoiding a "black box" approach.
- Linking the dragline simulation model to a three dimensional (3D) geological model to deal with data representing different geological conditions.
- Applying the process of dragline simulation to the whole deposit rather than using average conditions.
- Taking a modular approach so that the user is able to get the appropriate result at the end of each module. For example, a user may wish to estimate only productivity of an operation while another user may like to compute alternatives based on available cost data.
- Automating most of the dragline simulation processes using the capabilities of macros to avoid unnecessary program interruptions and speed up processes such as strip design optimisation procedures.
- Taking advantage of an existing simulation language (**DSLX: D**ragline **S**imulation **L**anguage for **X** Windows) instead of general purpose languages to reduce the time and effort required for programming.
- Using 3D CAD graphic facilities to provide visual displays of the design process.
- Using data recorded by dragline monitoring systems to calibrate the model as well as providing data required for productivity analysis.
- Applying sensitivity and risk analysis to evaluate the effects of uncertain parameters on the final results.
- Developing a cost model to allow the decision making processes to be based on the costs associated with the simulated options.

The simulation process in the CADSIM model is integrated from three major distinguishable phases. The first phase deals with development of a geological database and preparation and transferring data in a form which can be used in the next two phases. Data required during the simulation of a dragline operation are geological and geotechnical parameters, strip layout, digging method characteristics, dragline specifications, time study and cost data.

The second phase of this thesis is to develop the logic of the dragline simulator of the CADSIM system. This design stage is an iterative one and uses the input geological and dragline data to simulate different strip and dragline operations. Outputs from this stage are then used in the final phase which is a productivity data processing and analysing stage. The final phase of the simulation model is the decision making stage where the various alternative simulation results are compared and optimised.

The basic steps involved in the development of the CADSIM system are:

- **Geological data input:** Inputting the borehole data or digitised data from contour maps;
- **Grid generation:** Converting the raw data to produce gridded surfaces of the topography and the top and bottom of the coal seams;
- **Strip and section definition:** Building up the coordinates of the sections and design of the strip layout;
- **Layer definition:** Generating different layers from the gridded surfaces by intersecting with the defined sections;
- **Dragline specification data input:** Inputting the necessary dragline parameters required for simulation and productivity analysis;
- **Dragline pit design**: Includes design of the initial cuts, post-blasting profiles, dimensions of the strips, original surface and spoil shapes;
- **Dragline operation simulation:** This includes simulation of the digging method, sequencing the different operations, removal and placement of the pre strip and dragline overburden material, volume calculations, swing and hoist angles calculations, and the dragline walking pattern;
- **Output formatting:** Generating and formatting the output files to be used in other software such as spreadsheets;
- **Reading data to a spreadsheet:** Importing output from the dragline simulator and from input time study data;

- **Productivity analysis:** Development of a spreadsheet model to estimate the productivity of the simulated operations;
- **Optimisation process:** Optimising pit configurations, the dragline dimensions and strip design using a heuristic approach;
- **Risk and cost analysis:** Conducting a risk and cost analysis to identify the possible range of outcomes and comparing the simulated options based on the costs associated with each alternative.

4.2.1 Dragline Simulation with DSLX

One of the major objectives of this study was to develop a computer model to evaluate various operating functions such as different digging methods and pit configurations. This required a flexible and more general approach rather than the approach used by most of the available computer software. **DSLX** (**D**ragline **S**imulation **L**anguage for **X** Windows) software meets most of these needs as its flexible simulation language allows the user to develop different mining scenarios (DSLX Getting Started Manual, 1996). It also allows the process of strip mine design and dragline simulation to be automated. Simulation of a dragline operation using DSLX is based on a language concept which uses predefined functions to build spoil piles, working benches, blast profiles and strip geometry.

The simulation language of DSLX is defined as a non-procedural language. A non-procedural language is one which anticipates what the programmer is trying to do with each command and therefore considerably shortens the computer program (Harison and Sturgul, 1989). In contrast, FORTRAN is known as a "procedural" language. This means that FORTRAN statement can do only one operation at a time in a sequential manner. Whenever the programmer wants to have the computer perform a task, it is necessary to write the computer codes in a step by step manner. A non-procedural language often contains commands that result in the computer performing certain tasks without the programmer resorting to many detailed programming steps. Using a non-procedural language can be thought of as writing a program with only sub-programs.

The simulation language of DSLX contains all the elements of any high level computer language, such as line labelling, GOTO statements, loops (ie. DO, REPEAT, WHILE), nested loops, conditional statements (ie. IF-ELSE-ENDIF) and basic arithmetic operations (ie. +, -, *, /, etc.). In addition, the DSLX language is a CAD based language and contains a number of functions to allow strings, points and scalars to be manipulated and edited. These functions include intersection of strings, creation of points by intersecting of strings, creation of points by moving existing points by bearings and distances, maximum and minimum functions on points, concatenation of points into strings, concatenation of strings into new strings, etc. For an example, using "PNTINTS" function, a new point can be created by projecting an existing point to a string as illustrated below. This function is frequently used during the pit geometry definition.

PNTINTS Function:

PNTINTS (STR1,P1,ANG,PN) calculates a new point PN which is the intersection of a ray projected from point P1 at angle ANG with string STR1. Figure 4.2 illustrates concepts used in the PNTINTS function. The four arguments required by PNTINTS function are:

STR1 = *Intersecting String*
P1 = *Existing Point*
ANG = *Projection Angle*
PN = *New Point Found on STR1*

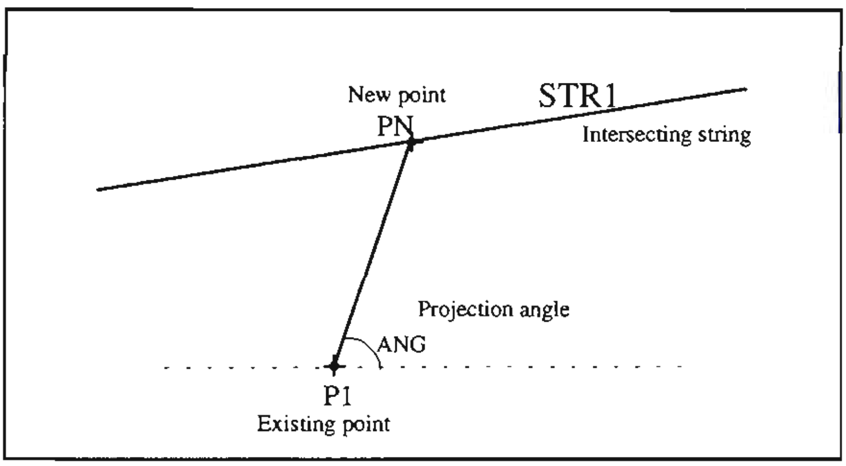


Figure 4.2- Concepts of the "PNTINTS" function.

The most frequent design procedures such as volumetric calculations or spoiling a given volume at the defined repose angle are coded in the more powerful functions. A useful function to perform volumetric calculations is the "VOLCOMP" function. The function is described below. This function has five arguments, the first three of them are input and the last two arguments are calculated and returned by the function.

VOLCOMP Function:

VOLCOMP (SPSTR,CUTS,CUTN,VOLP,VOLS) calculates prime and rehandle volume for a proposed cut. Referring to Figure 4.3 five arguments required by VOLCOMP function are:

SPSTR = *Current spoil string.*
CUTS = *Current cut string.*
CUTN = *New cut string.*
VOLP = *Volume, prime.*
VOLS = *Volume, spoil.*

The VOLCOMP function uses the three input strings to form closed string for volume calculations. The prime volume (*VOLP*) is the volume calculated from the area between the new cut string (*CUTN*) and the current cut string (*CUTS*). In a similar way the area between the spoil string (*SPSTR*) and the current cut string (*CUTS*) is calculated as spoil volume (*VOLS*). The strings are progressively updated as the mining advances down dip.

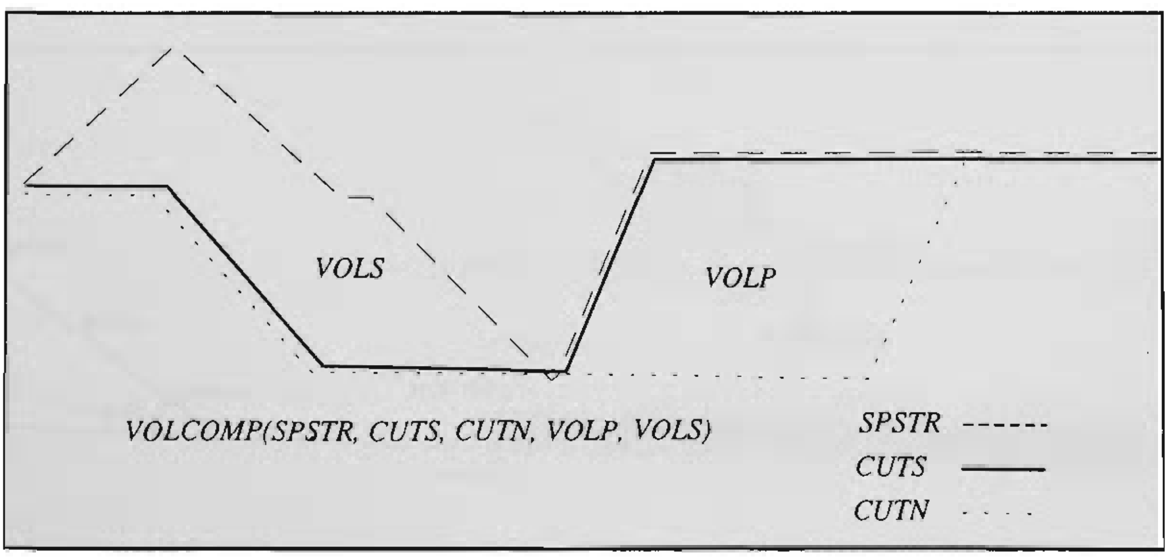


Figure 4.3- Concepts of the " VOLCOMP " function.

A simple example of a macro coded in DSLX language is provided below and the important DSLX functions are described in Appendix A.

4.2.2 An Example of a Macro in DSLX language

Figure 4.4 shows the process followed to simulate a cross section of the dragline pit. This is a single seam operation with a flat topography. The coal seam thickness is 5 metres and it dips at 1 degree. The overburden depth in this example is 30 metres. As with any computer program, the variables must first be defined. Here "GLOBAL" command defines three types of scalar, point and string variables used in this example.

global swidth, obdepth, coalthick, dip, repose, batter, wb
global_point start, end, ctoeold, ccrestold, otoeold, ocrestold, ctoenew, ccrestnew
global_point ocrestnew, tmp1, tmp2, dlpos, otoenew, tmp1
global_string roof, floor, tops, oldhwall, newhwall, spoil, cut, key

After definition of the variables, some variables are given the default values:

swidth = 60; cthick = 5; obdepth = 35; dip = -1; repose = 37; batter = 75

The following set of codes create the coal seam floor string. A string can be created simply by joining two points which define start and end of the string. First an original point *start* is created by giving coordinates X and Z equal to zero. The next point *end* is then generated by offsetting from *start* at a specified angle such as *dip*.

start = 0,0
end = start + {dip} 500*

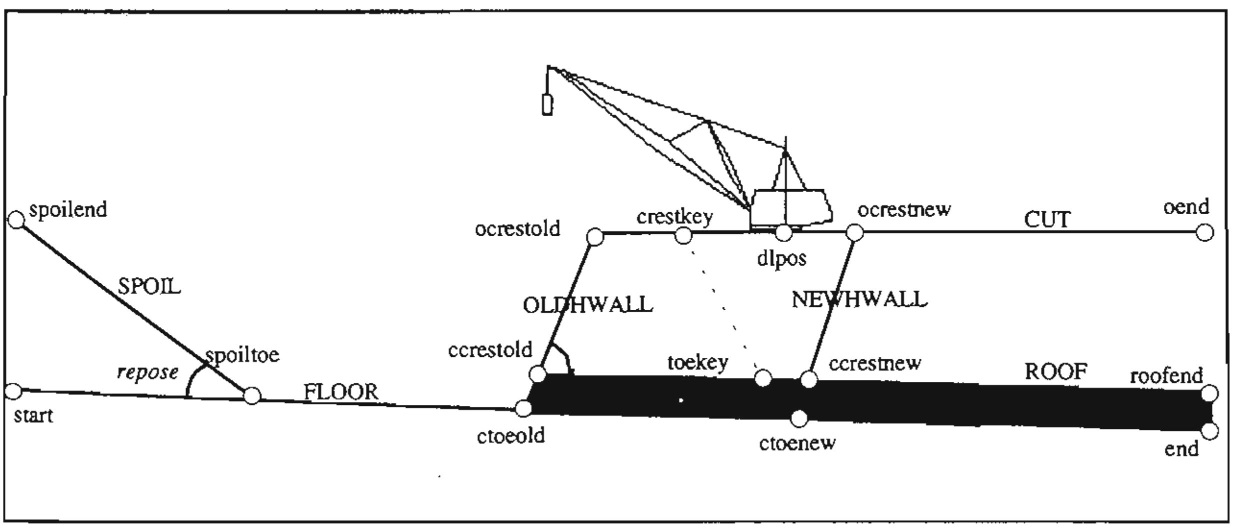


Figure 4.4- Points and strings used to construct the dragline pit for the example.

Once the start and end points are created, the string *floor* is a joint between these points. Operator "//" is used to concatenate points when creating a string.

floor = start // end

A starting point for design of the dragline cut can be the toe of the coal seam at the old highwall side (ctoeold). This starting point is created on the floor at the specified angle of *dip* and a distance of 200m.

ctoeold = start + {dip} 200*

Referring to Figure 4.4 all required points can then be generated from the starting point *ctoeold* to establish strings of old and new highwall, roof of coal, cut and spoil by the following relationship:

*ccrestold = ctoeold + {batter} * cthick*
*roofend = end +{90} * cthick*
roof = ctoeold//ccrestold//roofend//end
*ocrestold = ccrestold + {batter} * obdepth*
*oend = ocrest +{0} * 300*
oldhwall = ctoeold//ccrestold//ocrestold
cut = end//oldhwall//oend
*ctoenew = ctoeold + {dip} * swidth*

The new highwall points can be created by the projection of the toe of the coal, *ctoenew,* to the existing strings *roof* and *cut.* The PNTINTS function can be used for this purpose.

PNTINTS(roof,ctoenew,batter,ccrestnew)
PNTINTS(cut,ccrestnew,batter,ccrestnew)
newhwall = ctoenew//ccrestnew//ocrestnew

The old spoil string can be created by joining two points *spoiltoe* and *spoilend.* But first the points must be defined as follow:

*spoiltoe = start + {dip} * 200 - swidth*
*spoilend = spoiltoe + {180 - repose} * 70*
spoil = spoitoe//spoilend

The next step is generating a key cut inside the new pit. The width of key cut at the bottom is defined as *wb* and it is assumed that the left hand angle is the same as the batter angle.

*toekey = ccrestold + {dip} * stwdith - wb*
PNTINTS(cut,toekey,180-batter,crestkey)

key = toekey//ccrestkey//ocrestnew//ccrsetnew

Similarly, a point can be defined for the dragline position (*dlpos* point) on the *tops* string. The dragline position here is defined at the middle of the key cut. A temporary point *tmp1* is used for an equivalent *dlpos* point on the *roof* string.

*tmp1 = ccrestold + {dip} * swidth - (wb/2)*
tops = ocrestold//oend
PNTINTS(tops,tmp1,90,dlpos)

The final task in this example is to draw created strings and points. The DSLX drawing functions are available for drawing the dragline at a specified scale and for drawing strings and points at a specified colour and type such as dashed or filled.

DRAWSTR(cut,2,1)
DRAWSTR(spoil,3,0)
DRAWSTR(roof,4,0)
DRAWSTR(floor,5,0)
DRAWDL(dlpos,1,1)

4.2.3 Simulation of Dragline Digging Methods in CADSIM

All the processes involved in a dragline operation in the CADSIM system can be coded into a series of linked macros using the DSLX's functions. The macros developed in the CADSIM model are sub programs that have been coded and arranged in a logical sequence to simulate various dragline digging methods. These sub programs are called within a main program which controls the entire process. Each main program simulates a specific digging method such as Extended Bench or In-Pit Bench digging methods. The main program also controls the number of strips and sections which are being simulated and repeats the process for each new section. Seven modules have been developed in the CADSIM model to simulate various digging methods currently used by Australian strip mines. For example, module EXTBENCH consists of a main program and twenty-three sub-programs. Figure 4.5 shows the relationship of the main program and subroutines in module EXTBENCH that simulates the sequence of the dragline operation for a standard Extended Bench method. A brief description of the subroutines' functions is also provided in Figure 4.5.

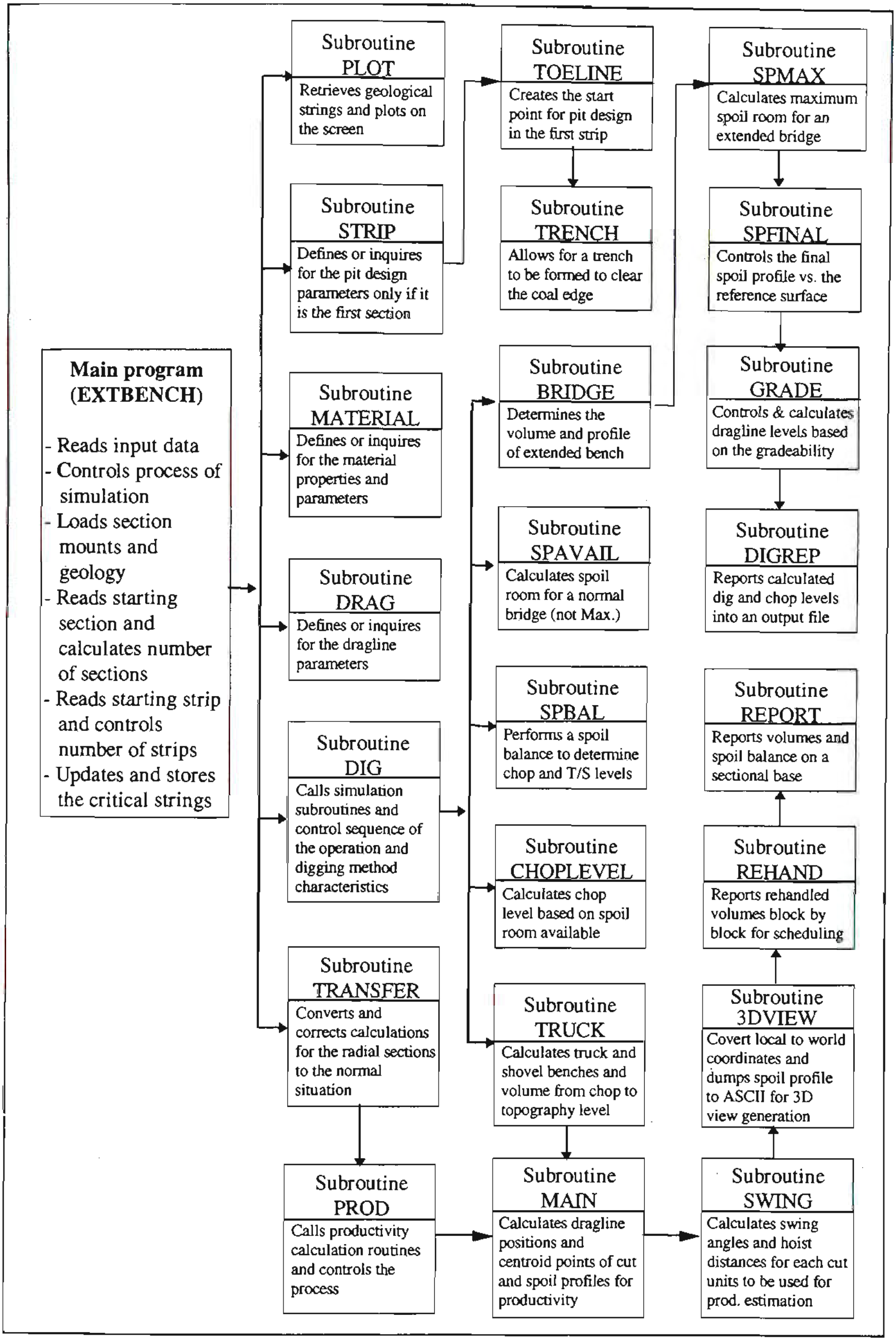


Figure 4.5- Relationship of the main program and subroutines in the EXTBENCH module of the CADSIM model.

The same approach as used in EXTBENCH module was followed for other modules. Some sub-programs that are used for simulation setup and formatting of program inputs and outputs may be shared between different modules. The degree of similarity among different modules depends on the similarity among the digging methods. For example, modules simulating multi-pass digging methods can share most of subroutines. Usually sub-programs DIG, PROD, SPBAL, SPMAX and SPFINAL that are the core of each module must be changed to suit a specific digging method. As it is also shown in Figure 4.5 a CADSIM module may have several levels of sub-programs with each sub-program calling another level of sub-programs.

With this approach the user can control the planning procedure, dragline movements and positions, the cut dimension and spoil placements. This special language approach lends itself to automation of the process of simulation, so that different options can be quickly tried through a number of geological conditions across a full deposit. The only limitation of this approach is the need to acquire the skills necessary to write a logical program in DSLX language. A relatively long time is also required to learn the specific functions of DSLX including coding and debugging of the operational procedures. In other words the flexibility of the software in handling exotic procedures is at the expense of more time and effort expended by the user.

4.2.4 Using the CADSIM Model as a Strip Mine Planning Tool

The CADSIM dragline simulator is linked to a geological modelling system to access the topography and coal seam structural data for simulation. Using a powerful 3D CAD tool, *VISTA,* which supports all the basic features for string editing, gridding, and triangulation, the CADSIM model constructs simulation sections from the geological model by intersecting vertical sections and a series of 2D grids of the topography surface and the roof/floor structural surfaces of coal seams. Strip mine design with the CADSIM is carried out by the appropriate cut and fill procedures coded in DSLX language as a series of commands and then complied into an executable model.

Using the CADSIM model the mine designer can define various practical criteria such as checking the maximum of spoil room and finding the shovel and truck base. For example, CADSIM provides the flexibility to check if the thickness of the partings at a particular area exceeds a pre-defined value (e.g. partings greater than 5m in thickness). When such a condition is satisfied the CADSIM modules can change the mode of operation to suit the new geological condition. These specific features of the model allow the user to evaluate different scenarios such as the changes in dragline dimension or the effect of mine design parameters. Output from the CADSIM dragline simulator consists of a series of user definable reports. For subsequent analyses such as productivity, sensitivity and cost analysis, output data are formatted in a manner suitable for input into a standard spreadsheet such as EXCEL. These procedures are detailed in the subsequent three chapters with each chapter addressing a major phase of the CADSIM system.

4.3 SUMMARY

Strip mine planning process is a combination of several engineering and decision making steps which must be linked together logically. For a detailed computerised analysis of a dragline operation the whole process must be first broken down into a series of individual modules to make the whole process more manageable. By this means each module can be made to address a major design aspect of the strip mine planning process. The major modules employed in this thesis are:

1. the geological interface module which provides basic geological and pit design data for the simulation phase,
2. the dragline operation simulation phase in which geometric and volumetric calculations are performed to provide required data for the analysis phase, and
3. the analysis stage in which productivity and cost analysis are carried out to provide the basis for selection of the best option.

These modules plus a 3D graphical tool can then be integrated to create a total strip mine planning system called CADSIM.

CHAPTER FIVE

DEVELOPMENT OF A GEOLOGICAL DATABASE

5.1 INTRODUCTION

Before commencing simulation of a dragline operation, the geology of the deposit must be modelled and presented in a format suitable for use in the simulation process. In most available computerised dragline simulators, geological data such as the coal and overburden thicknesses for a representative section are input manually. Such an approach becomes tedious and inefficient as the geology becomes increasingly complex and the number of simulated mining blocks increases. Various geological modelling techniques such as regular block models, irregular block models, cross sectional models and grid seam models may be used depending on the nature of the deposit and the modelling objectives. The gridded seam model was found to be the most efficient technique for modelling coal deposits for the purposes of this thesis. Simulation of dragline operations employed here uses the geological data from a gridded seam model to build a series of geological sections. The geology of the coal seams and the topography of the surface is represented by a set of strings in each section. These strings are generated by intersecting 2D grids from the geological model and the planes of a series of section. The relevant information associated with each section is then stored in ASCII format to be accessed during the simulation phase.

5.2 GEOLOGICAL MODELLING

The major purpose of geological modelling is to develop a three dimensional picture of the geological features of a deposit. This starts with the gathering of soft and hard geological information, including drill hole data, geophysical logs, topographical maps, cross sections and surveying data. The next step in modelling is to develop a data base to organise the available data into appropriate categories such as quality, structural and thickness data. A drill hole database usually consists of a set of information that defines drill hole location and geological thickness as well as assay data. A geological database should at least contain following information:

1. Borehole name, collar elevation, easting and northing coordinates, drill hole deviation, seam identification, seam top and bottom intercepts, coal quality, lithology and rock type.
2. Additional information, such as driller's logs, geophysical logs, special codes for geological conditions, and water table.

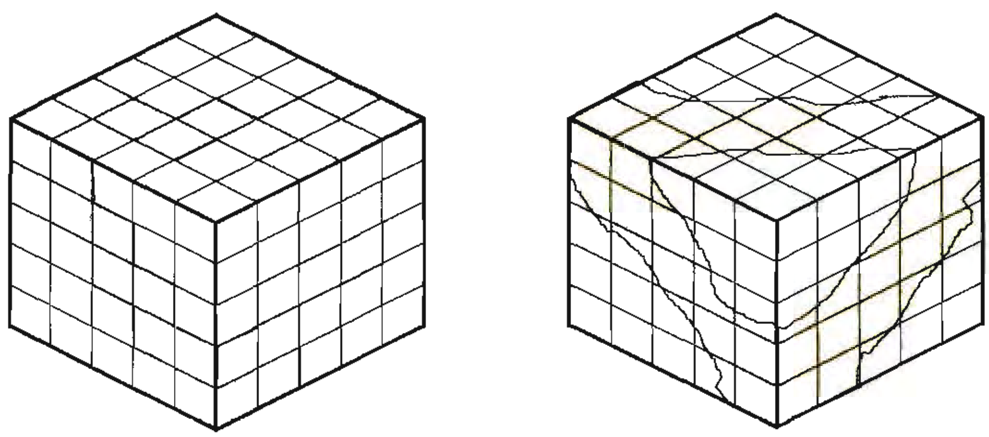
In addition to the borehole data, digital map data may also be entered into the database from the existing maps. Topographic and survey data are frequently provided using photogrammetric means or by digitising existing maps. Once a database of geological data is established the next step is to interpret this information. The interpretation process is highly dependent on the knowledge and experience of the geologists. The interpretation stage is one of the most critical steps toward developing a sound geological model. Any misinterpretation of the geological data in development of the model can lead to large errors in resource estimation and other mine planning processes.

5.2.1 Geological Modelling Techniques

The geological modelling process combines the power of imagination with mathematical formulations to arrive at a satisfactory model of a deposit. Almost all of the modelling techniques require a computer for storage, organisation of the data as well as mathematical formulation. Common modelling techniques include regular and irregular block models, grid models, cross-sectional models, solid models, surface models and string models (Williams, 1993). Among these modelling techniques, block, grid and cross-sectional and string models are the most common methods used for modelling coal deposits (Kim and Wolff, 1978; Michaud, 1991).

5.2.1.1 Block Model

A block model is the collection of a three dimensional set of regular (or irregular) blocks of given X, Y and Z dimensions. The dimensions of the blocks are primarily dependent on the mineralisation geometry, data availability and the model objectives (Brew and Lee, 1988). The blocks are considered to be contiguous in all directions and no gap is allowed in the model. Each block is identified by the X, Y and Z coordinate at the centre of the block and is assigned a series of attributes such as grade, rock type, dollar value, etc. Block modelling is more suitable for grade estimation techniques and is commonly used for vein type deposits and irregular massive and disseminated deposits such as porphyry copper, uranium and gold. Block modelling can also be applied to steepy dipping strata type deposits and very thick coal seams (Badiozamani, 1992). An example of a geological unit interpreted into a block model is shown in Figure 5.1.



Blocks contain mineral *Interpreted model*

Figure 5.1- A regular set of 3D blocks used to model a deposit.

5.2.1.2 Cross Sectional Model

Cross-sectional modelling uses a series of parallel and radial sections to describe a deposit (Figure 5.2). In this way each section contains a geological description of the deposit along the section line. A geological model can be developed from a set of parallel planes which are a combination of plans and sections applying concepts of the cross-sectional model (Williams, 1993). The technique assumes that each section has a width of influence which is normally half way between adjacent sections. The model of deposit can then be constructed by connecting these sections to one another by linear interpolation and assuming a gradual change between the sections.

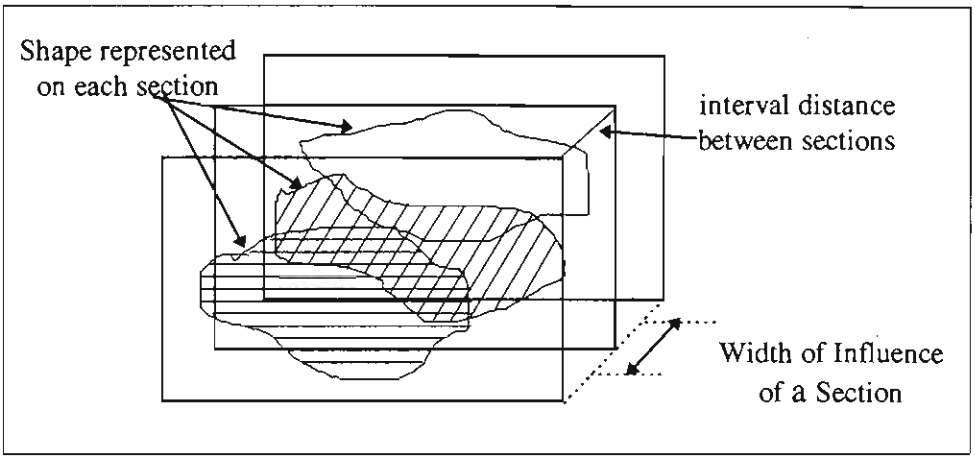


Figure 5.2- An example of a Cross-sectional model.

Although the cross-sectional model provides a relatively easy and quick method for the modelling of a coal deposit, the accuracy of this method decreases as the complexity of the geology increases. Also irregularities in the topographical surface and seam geology cannot be accurately represented unless very closely spaced sections are used.

5.2.1.3 String Model

String models consist of a sequence of X, Y and Z coordinates with one or more additional attributes or values at each point. Points may be unique or joined in a continuous series. Joined points form lines which may be either closed or open (Figure 5.3). Closed lines have an area which can be calculated. The string concept is frequently used in cross-sectional models for data manipulation and volume calculations. Most of the calculations and mine design procedures in the dragline stripping model can be carried out using strings and points. The string structure provides less data storage devices and high speed and efficiency during the calculations required. Although there are inherent disadvantages in the use of strings for irregular orebodies, the technique is quite adequate for modelling bedded deposits such as coal (Williams, 1993).

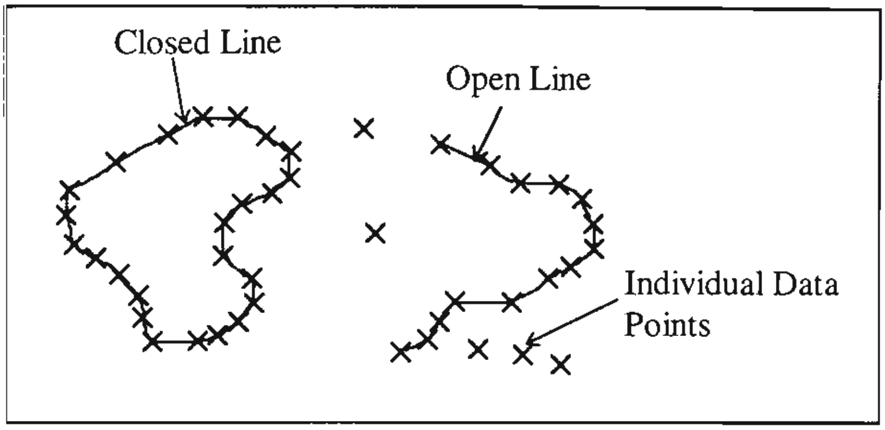


Figure 5.3- An example of a String model.

5.2.2 Gridded Seam Model

The Gridded Seam Model (GSM) is extensively used to model the geology of multiple coal seams (Williams, 1993; Michaud, 1991; Boyd, 1990; Brew and Lee, 1988; Sadri and Lee, 1982). The GSM uses a series of computer programs to transform topographic elevations, drill hole data and geologic knowledge into a three dimensional model of the deposit. This model of the deposit can then be displayed using maps, cross sections and 3D graphic views. The gridded model is a set of two-dimensional matrices, each representing a surface or value. These surfaces or values are the results of interpolation from a set of irregularly spaced data to a regular and fixed matrix called a "grid".

One of the main advantages of using a GSM over other irregular modelling techniques is a reduction in the disk space required by eliminating the need for storing all easting and northing coordinates. Given a matrix and the coordinates of its starting position along with the X and Y increments, the location of any other point on the matrix can be established. This is an important advantage especially when numerous seams are involved and each seam may have a number of attributes associated with it. Another major advantage of using GSM to model bedded deposits is ease of use and manipulation of the data. The new set of data and attributes can be easily generated using mathematical operators (e.g. adding, subtracting) or logical operators (e.g. "OR" and "AND" statements). For example, the bottom structure of a seam can be subtracted from its top structure to arrive at the seam thickness (Badiozamani, 1992).

In addition to the above advantages, the gridded seam method can be easily integrated with cross sections to generate cut and spoil profiles. The intersecting plane of a section with different grids such as topography surface and the coal seam's structural roof and floor surfaces results in the generation of strings of data. These strings can be used to construct the geometrical structures of the dragline cut and spoil profiles for the calculation of the volumes.

The advantages inherent in a grid based approach outweigh the disadvantages. Tasks such as drawing contour lines and volumetric calculations of map modifications are much faster using this approach. Under most circumstances, there are few problems when using a gridded model to produce a contour map. One potential disadvantage to gridding is the possibility that the original data points might not be honoured in the grid nodes. Contour maps are drawn from the interpolated grid rather than the original input data points.

5.2.2.1 Grid Estimation Techniques

The data gathered to model the geology of a deposit or estimate a reserve are often irregularly spaced. To perform calculations and to present the model in the forms of contour maps or 3D surfaces, the user must process the data to generate a grid matrix. The grid matrix consists of rectangular meshes with the surface elevation estimated for each grid node. The term "irregularly spaced" implies that the points are randomly distributed over the extent of the map area meaning that the distance between the data points is not consistent over the map. When the XYZ data is randomly spaced over the map area, there are many "holes" (missing points) in the distribution of the data points. A gridding technique fills in the holes by extrapolating or interpolating Z values in those locations where no data exists.

The estimation process is common to all modelling techniques and it involves the application of a series of algorithms or mathematical methods to convert a set of irregularly spaced data to a regular pattern such as grid nodes. In a gridded model, the grid matrices can be generated by a variety of techniques, including (Hays, Betzler and Canton, 1990):

1) Polynomial regression,

2) Geostatistical methods such as 2D and 3D kriging, and

3) Various distance-weighted functions with a variety of search procedures.

Polynomial regression is used to define large scale trends and patterns in the data. Polynomial regression is not really an interpolator because it does not attempt to predict unknown Z values. The method requires that all the data points be used when calculating the grid.

Kriging is a geostatistical gridding method which has been found to be very useful in many fields. Geostatistical methods are commonly used for grade estimation. Kriging attempts to express trends that are suggested in the data set. The variogram is fundamental tool used in kriging and other geostatistical methods, and is a graph of the average variability between samples against the distance between the samples (Nobel, 1992).

The Inverse Distance Weighted (IDW) method is one the most widely used techniques applied in computerised modelling for grid generation (Badiozamani, 1992). The principle concept of an inverse distance technique is that the data points further away from a node have lesser effect on the node than the closer ones. Usually a weighting factor is assigned to the data points, controlling how the influence of the data points declines as the distance increases. The greater the weighted power, the less the effect given to the remote points from the node being estimated. The basic equation used by the weighted average methods (Michaud, 1991) is:

$$V(X,Y)=\frac{\sum_{i=1}^{n}\frac{r_i}{(a^k + d_i^k)}}{\sum_{i=1}^{n}\frac{1}{(a^k + d_i^k)}} \tag{5.1}$$

where: $V(X,Y)$ = the estimated parameter at coordinates of (X, Y),

r_i = value of the parameter at coordinate of (X_i, Y_i),

d_i = distance between (X, Y) and (X_i, Y_i),

n = number of data points,

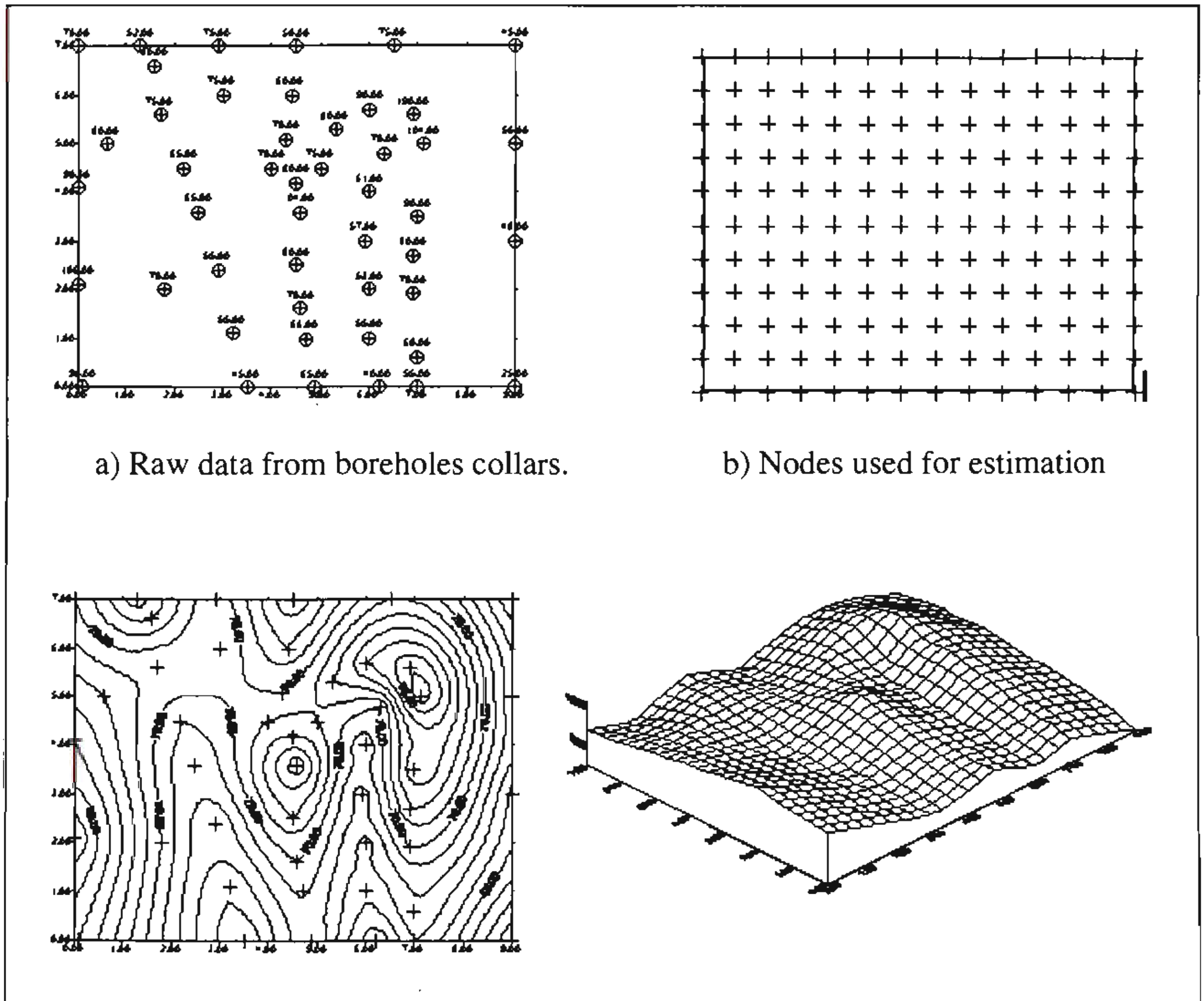
a = a constant which is a function of the local variability of the parameter, and

k = a constant which is a function of the regional variability of the parameter.

For most of the gridding exercises and for estimating the elevation of a grid node "$Z_{(X,Y)}$", the local variability function is set to zero (a = 0). Then Equation 5.1 is reduced to an equation based on a distance relationship as follows:

$$Z_{(X,Y)} = \frac{\sum_{i=1}^{n} \frac{z_i}{d_i}}{\sum_{i=1}^{n} \frac{1}{d_i^k}} \qquad (5.2)$$

In Equation 5.2, the variable k usually varies between 1 to 2 depending on the nature of the geology. However, the best value for k must be found over several trials on a given data set. Figure 5.4 shows the process of grid generation from a set of borehole collars. The contour map and 3D surface created from the grid generated using an IDW technique (k = 2) is also shown in the figure.



a) Raw data from boreholes collars.

b) Nodes used for estimation

c) Contour map after grid modelling.

d) 3D view of the gridded topography.

Figure 5.4- Grid modelling of a topography surface from borehole collars.

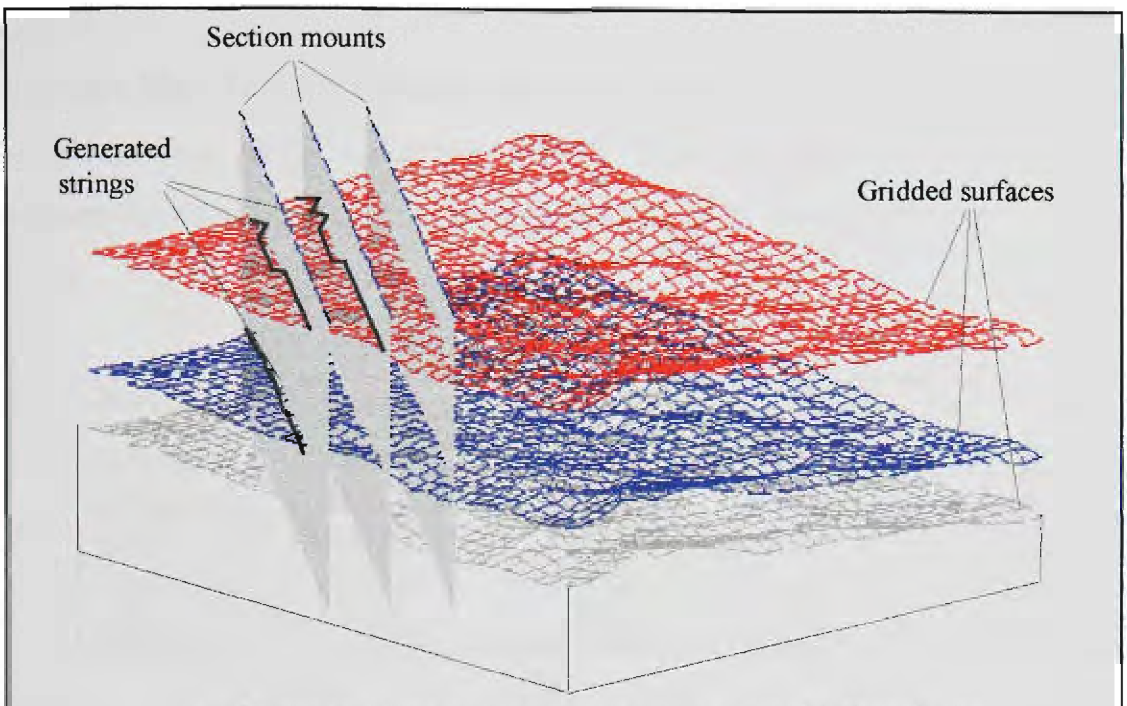
The differences between the gridding methods are essentially in the mathematical algorithm used to estimate the sample weightings used during grid node interpolation. Each method can result in a different representation of the data. It is advantageous to apply the different methods to the data set to determine the gridding method that provides the most appropriate results which satisfy the geologist.

5.3 DEVELOPMENT OF A GEOLOGICAL DATABASE

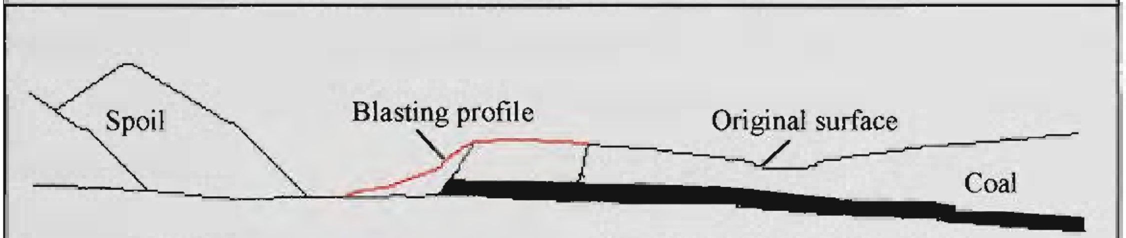
The methods of transferring data from the geological model to the mine design modules varies from computer based system to system, with varying degrees of flexibility. The most successful approach in an integrated system is to use a unique structural approach that provides the designer with the flexibility to move freely between the system modules. To achieve this, most integrated mining software build their various modules based on a common "data structure". This common data structure approach allows relatively easy interaction and communication between different modules within a system. One data structure which is being extensively used in integrated mining software is the "string" approach (Brew and Lee, 1988).

One of the main features of the CADSIM system which was developed as part of this thesis, is the total integration of all aspects of geological modelling, the strip mine planning process and volumetric calculations. This integration was made possible through the use of strings as the base for the modelling process. The initial strings are made up by the intersection of the plane of the cross-sections and the gridded surfaces in the geological model. The geology of the simulated sections is merely a set of strings that represent the cross section of different material types at different easting and northing locations. Various material types can be defined using a pair of strings. For instance, the area between the topography string and the coal roof string in a section represents overburden material.

The geological information required for the simulation of dragline operations is built up by interrogating the structural surface grids along the simulation section. As schematically shown in Figure 5.5, by intersecting gridded surfaces and plane of sections (section mounts) it is possible to create strings which represent each surface (e.g. topography surface) in a section. Before the geological sections can be generated, the two major steps of generating grids and definition of section mounts must be performed.



a- Example of intersecting sections and gridded surfaces.



b- The resultant strings from an active pit with thrown blasting.

Figure 5.5- Example of intersecting sections and girded surfaces.

5.3.1 Generation of Grids in the Geological Model

Prior to generating grids in the geological model, a database of drill hole data must be established. The following three data files were used to build the borehole database:

Collar data file: This file is initially used to build the database and has fields such as borehole name, X,Y and Z coordinates of the collar, final depth of the hole and azimuth and dip of the borehole at the collar. Table 5.1 is an example of a part of a collar data file.

Table 5.1- An example of a collar data file.

Borehole ID	Easting	Northing	Elevation	Depth	Azimuth	Dip
DDH016	204993.06	156305.34	1185.33	198.48	265.0	-89.30
DDH023	205058.53	156258.42	1184.86	220.00	295.2	-89.35
DDH024	204990.70	156642.56	1176.19	208.48	287.5	-89.20
DDH027	205080.05	156369.03	1185.74	225.60	290.0	-89.50
DDH032	205206.98	156503.56	1179.91	255.68	288.4	-89.90
DDH038	205347.47	156337.14	1183.75	279.28	284.6	-89.75

Data type file: This file is used to add down hole data to the database with each data type assigned a name. The typical data types are lithological data, quality and geophysical sampling data. An example of a data type file is given in Table 5.2.

Table 5.2- An example of a data type file.

Borehole ID	From	To	Ash (%)	Density
DDH016	0.00	135.84	94.0	2.21
DDH016	135.84	136.56	12.5	1.43
DDH016	136.56	137.76	56.7	1.87
DDH016	137.76	138.96	8.9	1.41
DDH016	138.96	139.76	72.6	2.02

Pick interval file: This file includes a list of interpreted seam intervals. The seam intervals are stored in a stratigraphic sequence list as interpreted by the geologist. This file is frequently used to load data for generating structural grids during the grid generation phase. Table 5.3 is an example of a pick interval file for an area covering two coal seams coded as HM1 and LM1.

Table 5.3- An example of a pick interval file.

Borehole ID	From	To	Seam Code
DDH016	137.76	144.08	HM1
DDH016	165.93	168.45	LM1
DDH023	203.20	208.54	HM1
DDH023	236.40	239.32	LM1
DDH024	224.61	227.28	HM1
DDH024	259.20	263.80	LM1

Before a database can be used to generate gridded surfaces in the geological model some modifications are required. In the first step, a seam correlation must be performed between drill holes using cross sections. This is normally performed by the computer, however manual correlations may be required due to the complexity of the geology.

Next, the coal intercepts have to be "composited" for each seam. For instance, if a seam split has more than one intersection in a drill hole, the total thickness of the coal and the total partings must be calculated as one unit.

The topographic surface is normally the first grid which is created in the model. The data required to create a grid of the topographical surface are normally obtained by digitising surface contour lines obtained from existing maps. The topographical surface can be also developed from survey data or drill hole collar elevations. The data is then processed to generate a grid matrix consisting of a rectangular mesh with the surface elevation estimated for each grid node. In this thesis the estimation technique used for gridding is based on an inverse distance weighted average method. The mesh size used for the topographical grid matrix is generally used for all subsequent grid generations. However, as the coal seam structural surfaces are usually smoother, a greater mesh size may be used for the coal seam grids.

The gridded surfaces of the roof and floor of all mineable coal seams are created using the composited data from the borehole database. Two common methods used to generate seam surfaces are triangulation and weighted average method. The triangulation method provides an easy and quick method for creating a surface when enough data are available. As shown in Figure 5.6 the triangulated surface of a seam floor can be created by connecting the end points of the seam intervals obtained from the borehole database. This triangulated surface can then be converted to a grid representing the floor surface of the coal seam (Figure 5.7).

When creating floor and roof grids of a thin coal seam, in some cases the roof and floor grids may cross each other (ie. negative thickness). To avoid this problem, it is preferable that only one grid (e.g. floor) be created in the first stage. It is also possible to create a thickness grid for each coal seam using data from the pick interval file. The roof grids can then be produced by summation of the two grids of floor and thickness. A 3D gridded seam model (GSM) is then generated by combining the information from all gridded surfaces including topography, depth of weathering and various coal splits and seams into a binary file called MODEL.GRD.

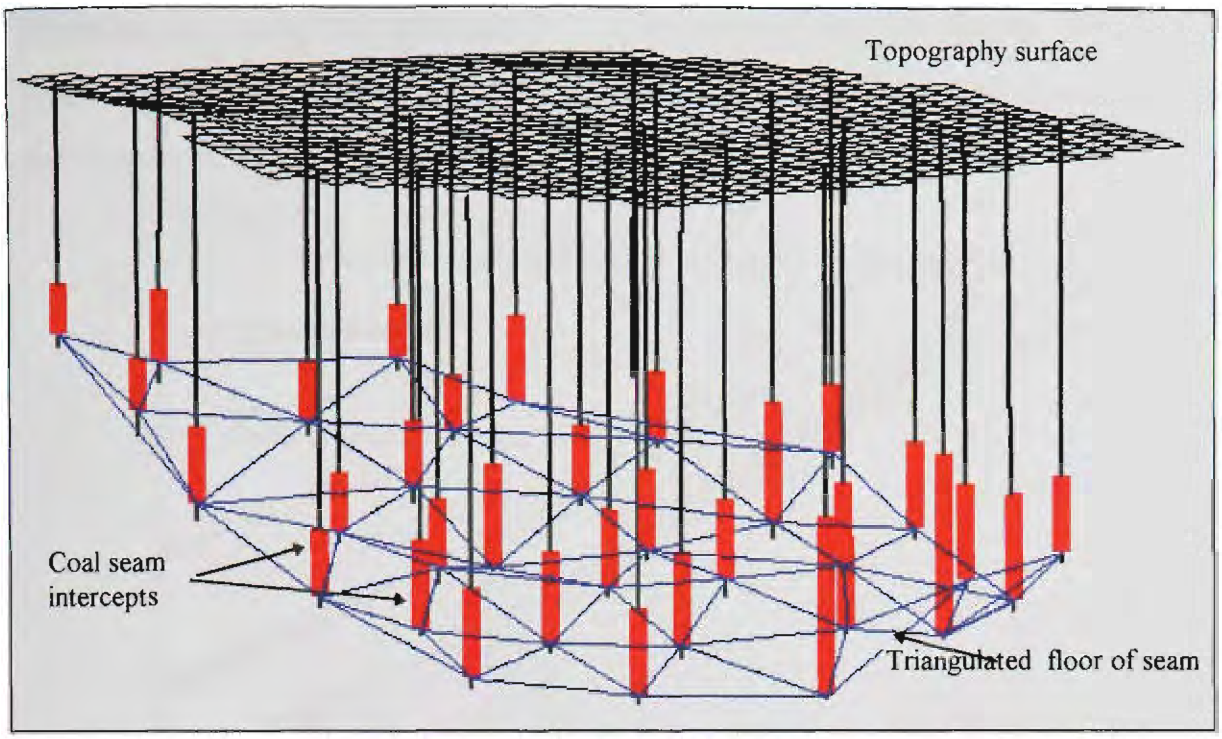


Figure 5.6- Triangulating a seam floor surface using borehole data.

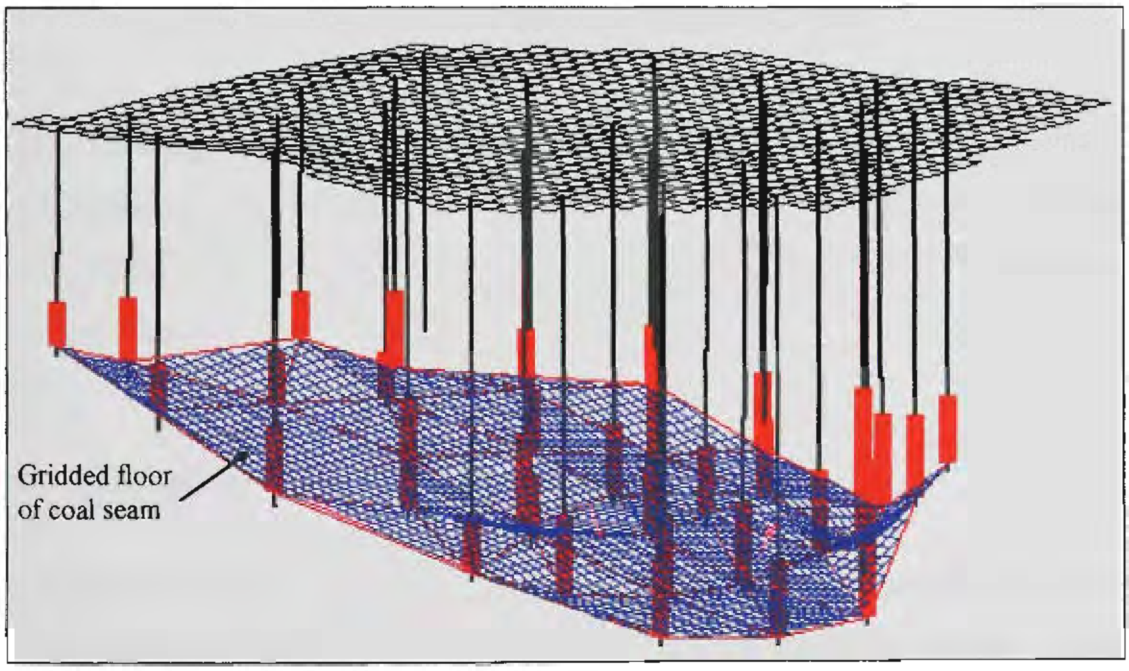


Figure 5.7- Converting the triangulated surface of the seam floor to a 2D grid.

The last step in preparing grids in a format that can be used by the simulation process is merging the grids to fill the undefined parts of the roof and floor grids. The concepts of merging grids is shown in Figure 5.8. The merge grid file takes all of the structural information of the modelled grids and merges all of the undefined parts of a surface grid vertically upwards to the superior surface. For example, in a stratigraphy of seam 1 and seam 2, seam 1 is merged up to topography where it is undefined and seam 2 is then merged up to seam 1 where seam 2 is undefined. In general, this causes the seams to run along the topography grid where they are terminated by weathering or faults. This will not affect the volumetric calculations as the thickness of each layer remains zero in the undefined area.

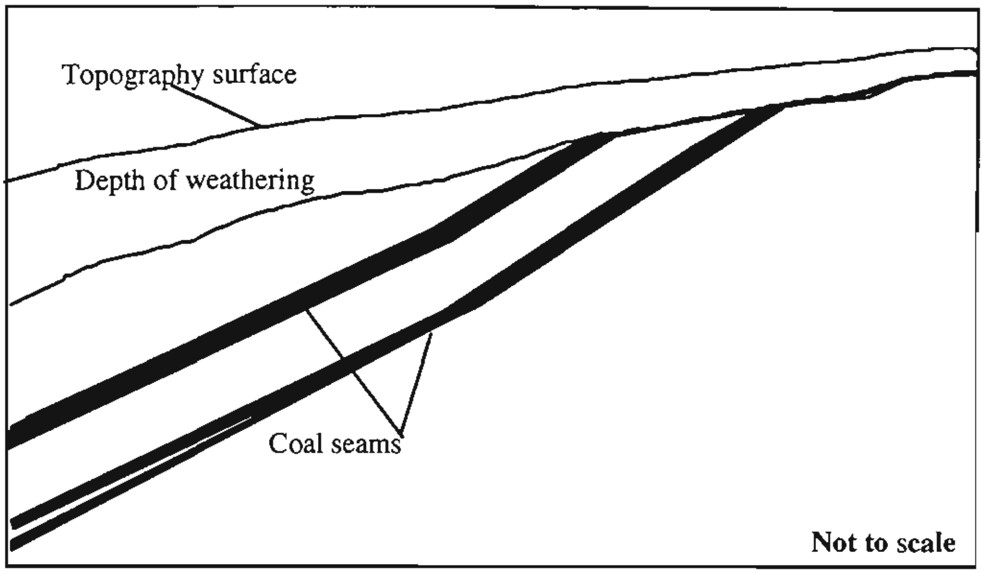


Figure 5.8- Concepts of the merged grids.

After generating and editing all of the gridded surfaces of the total mining area required for simulating the dragline operation, the grid information is combined into MERGE.GRD file. This file contains the elevations and mesh size of each grid stored in binary format.

5.3.2 Definition of Sections

The original sections used for the dragline simulation are frames with real coordinates with no geological information; they are normally called section mounts. The section mounts used for the simulation are vertical and are defined by start and end Cartesian coordinates. A section mount also has a vertical maximum and minimum range which together with start and end coordinates define a window in 3D space. Section mounts are generated by defining the reference coordinates points either via the text input or by digitising from the screen. Once the section mounts are created, they are stored in a binary file called *Geometry File* for later integration with the geological data. The sections are named with a common prefix (e.g. S<1>to S<n>) to facilitate access by the computer routines developed for the dragline simulation.

In order to simulate the dragline operation in a given pit which may include several strips, a set of parallel and radial sections is required. The radial sections are normally used for curved strips (Figure 5.9). The number and type of sections are determined by the strip length, pit geometry, topography irregularity and the accuracy of available geological information. For example, in a highly variable topography and faulty coal area, a closely spaced set of sections is required to obtain accurate and reliable results.

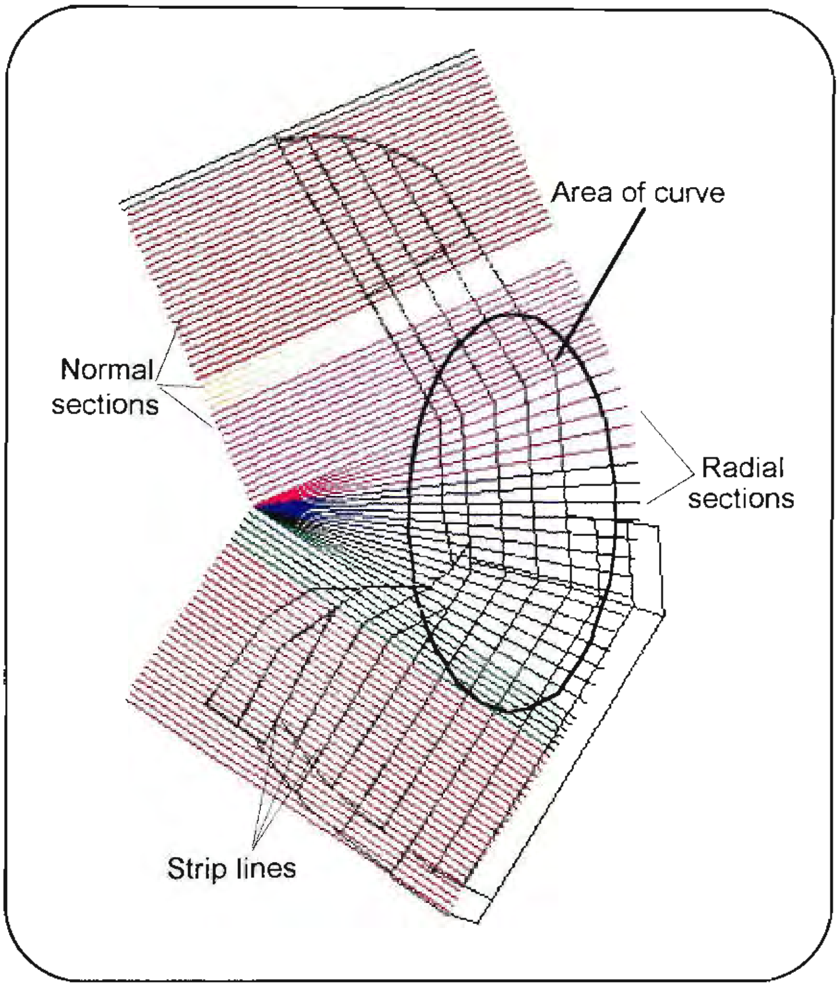


Figure 5.9- Plan view of the radial and parallel sections.

5.3.3 Creating Output Files from the Geological Model

ASCII files are the key element of transferring the data between the geological, the dragline simulation and analytical phases. The use of ASCII files as a means of data integration also enables the dragline simulation to access data from other sources and databases. Another advantage of this type of data storage is that the data can be read and any error is relatively easier to delete.

To create ASCII files for the simulation of the dragline operation a list of grids from the geological model and sections mounts must be first defined. The grids must be defined based on their original order from top to bottom. Once the list of sections and grids are defined, the CADSIM model then processes each section by intersecting the section through all the grids defined in the list. As a section is processed various output files are generated for the coordinates of the section mounts, intersecting strings of geological layers (ie. original topography and roof and bottom of the coal seams), cut, spoil and reference surfaces. The resultant string for each section can also be reviewed on the screen while processing. Seam elevations are taken along the section axis at every defined interval. The distances of these intervals depend upon the consistency of the gridded surface and the mesh size used for grid generation.

5.3.3.1 Section Coordinates File

For ease of calculation and data storage the real easting-northing coordinates of the string points are generally not used in the dragline simulation. Rather local coordinates are developed for X and Y values with the Z value remaining unchanged. However, after simulation the designer may wish to dump the section data to real world coordinates for use in mapping or plotting. A reverse procedure can be developed to transform section coordinates (offset, elevation) to real world coordinates. The section coordinates file keeps the original data of each section to meet this need. An example of a part of the section coordinates file (SnCOO.STR) is shown in Table 5.4, where column 1 is the chainage along the section and columns 2 and 3 are easting-northing coordinates of the chainage.

Table 5.4- An example of a coordinate (SnCOO.STR) file content.

0.00	2156434.25	899889.19
5.00	2156424.25	899897.52
10.04	2156415.50	899905.62
15.06	2156405.50	899912.44
20.08	2156396.75	899919.05
25.10	2156384.75	899926.56
30.12	2156375.25	899932.16

5.3.3.2 *Layer Surface File*

The resultant strings from intersecting layers and section mounts are stored in a single ASCII format file for each section. This file is called the *layers file* in which each string addresses a grid surface in that particular section. The strings are stored corresponding to the grid sequences from top to bottom. Typical grids are original topography, roof and floor of coal seam(s). The number of columns in this file depends on the number of surfaces in the section. An example of a layer file (SnLAY.STR) contents for three surfaces is shown in Table 5.5.

Table 5.5- An example of a layer file (SnLAY.STR) content.

0.00	234.94	207.18	200.42
5.02	234.00	206.73	200.16
10.04	233.03	206.30	199.90
15.06	232.70	205.76	199.63
25.10	230.93	204.81	199.14
30.12	229.94	204.39	198.93
35.15	229.05	203.93	198.70

In this file column 1 is the chainage along section and columns 2 through 4 are elevations of the three intersected surfaces (ie. topography, roof and floor of coal) at that chainage.

5.3.3.3 *Cut, Spoil and Reference Surface Files*

During the dragline simulation process, the coded routines read string information from the input files and after processing them writes to the output files. The most frequently used strings are existing spoil, current excavated cut and reference surface strings. The elevation data on these strings are stored separately into the single file for each section. An example of a cut surface file (SnCUT.STR) content is shown in Table 5.6.

Table 5.6- An example of a SnCUT.STR file content.

0.00	234.94
5.02	234.00
10.04	233.03
15.06	232.70
20.08	231.90
25.10	230.93
30.12	229.94
35.15	229.05

In Table 5.6 the first column displays the intervals along the section and the second column contains the elevation information on the excavated surface for each interval.

Reference (or permit) surface is used to define any limitation for spoiling during the dragline simulation process. The reference surface shows the maximum possible elevations for the spoil surface during and after the simulation. For example, in the case of a coal access ramp there is a limitation for spoiling in the area around the ramp and also there may be a limitation on spoil height imposed by stability requirements. In order to create a reference surface the following five stage process may be used:

1. The final ramp design for the whole mining area must be generated by defining two parallel strings to delineate the toe lines of the spoil pile at both sides of the ramp. The distance between these strings is equal to the ramp width. This can be done by either digitising from the screen or by accessing data from mine planning maps. The parallel strings are then fitted to the base of the bottom coal seam grid.
2. A temporary grid is created at a level which is determined by maximum spoil height or a higher level when there is no limitation for spoiling. This temporary grid must be large enough to cover the whole area. If there is a limitation of spoil height for stability reasons (e.g. maximum of 90m), the temporary grid can be generated by adding the maximum height to the base of the coal grid.
3. The strings created in stage 1 above are projected to the temporary grid at the spoil repose angle. The result of this projection would be two new strings (Figure 5.10).

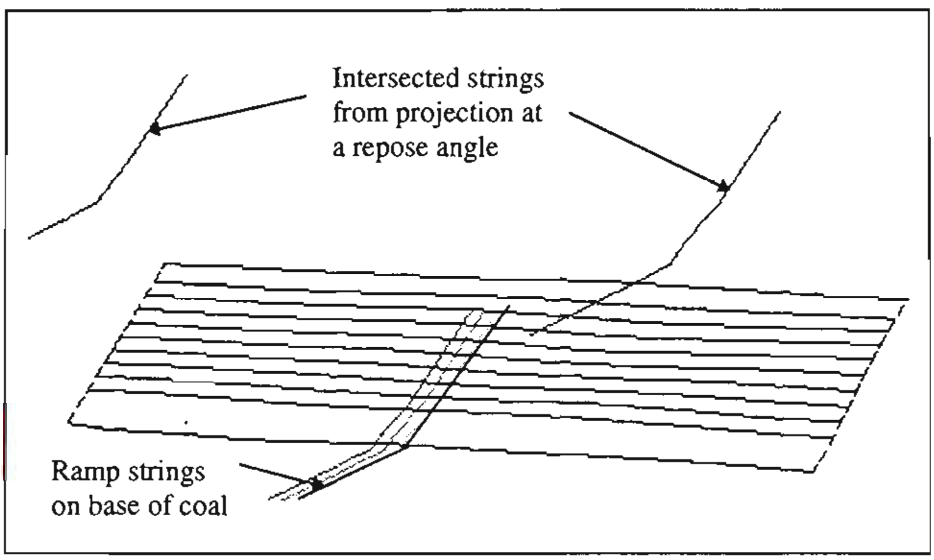


Figure 5.10- Generation of the access ramp strings.

4. The ramp related strings are used to generate a new surface, applying a triangulation process. This represents the base of the ramp and spoil faces at both sides of the ramp. The triangulated surface is then converted to a new grid called "RAMP" (Figure 5.11).

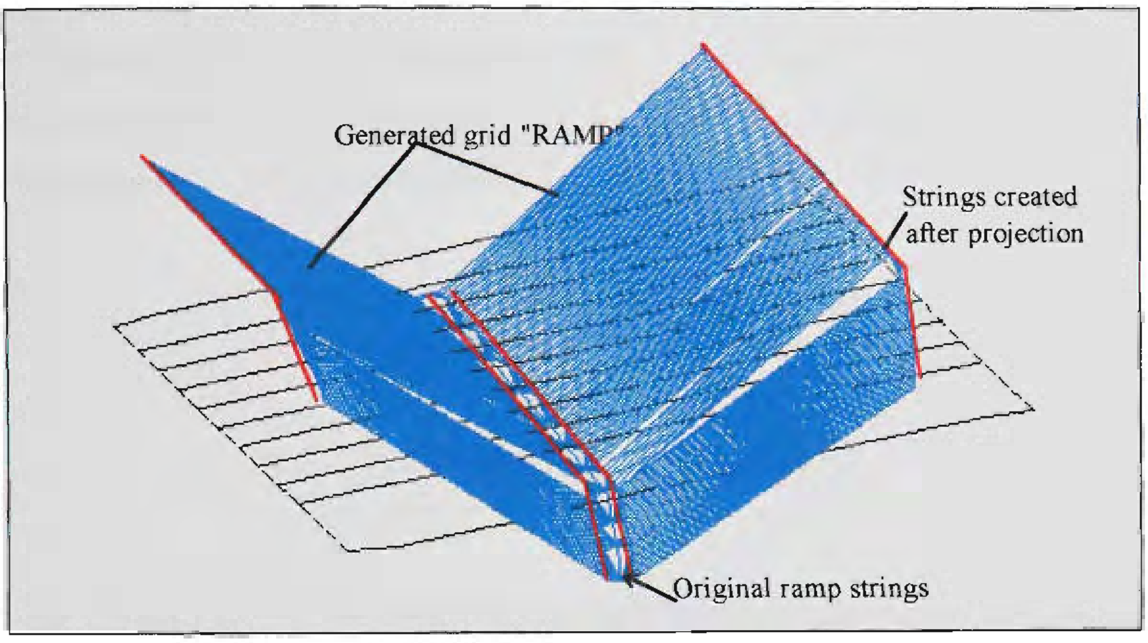


Figure 5.11- Creating the RAMP grid by triangulation of the digitised ramp strings.

5. Finally the RAMP grid and the temporary grid are merged together to generate the reference surface (Figure 5.12).

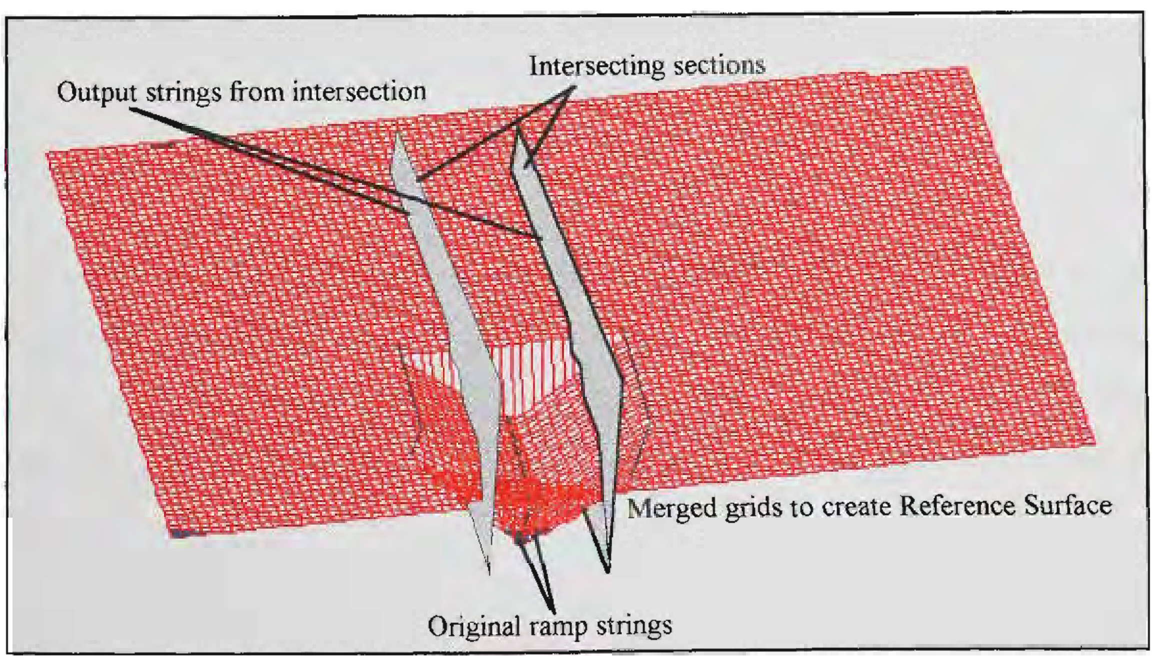


Figure 5.12- Merging grids to generate final reference surface.

5.3.4 Definition of Strip Layout

To simulate an existing layout of strips, the strings of strip lines are accessed from either an existing file or are digitised on the screen. These strings are then intersected with the current section mounts from the defined section list. The intersection points for each strip line are stored in a file. The information stored in the file includes the section mount name, the horizontal distance from the section origin to the point of intersection and the elevation at the intersection. An example of the content of a strip file for four section mounts is illustrated in Table 5.7.

Table 5.7- An example of a strip file content.

S1	417.51	91.721
S2	417.01	90.532
S3	416.21	90.123
S4	416.12	91.243

In this file the first column represents the intersecting sections, the second column is the distance of the intersection point from the section origin and the last column is the angle between the section line and the strip line. The intersecting angle is used to determine whether a section is radial or not. The radial sections are a result of the intersection of curved strips and section lines which have intersecting angles either less than 87^{o} or greater than 93^{o}. As a result of this oblique intersection, volumetric calculations and reach parameters of radial sections must be treated differently.

5.3.5 Width of Influence

In order to calculate the volumes of the calculated areas for radial and parallel sections, a band width sampling technique can be adopted. This allows automatic sampling along a width located on either side of the section showing the average data for that band width. Figure 5.13 shows two types of sampling bound for the radial and parallel sections.

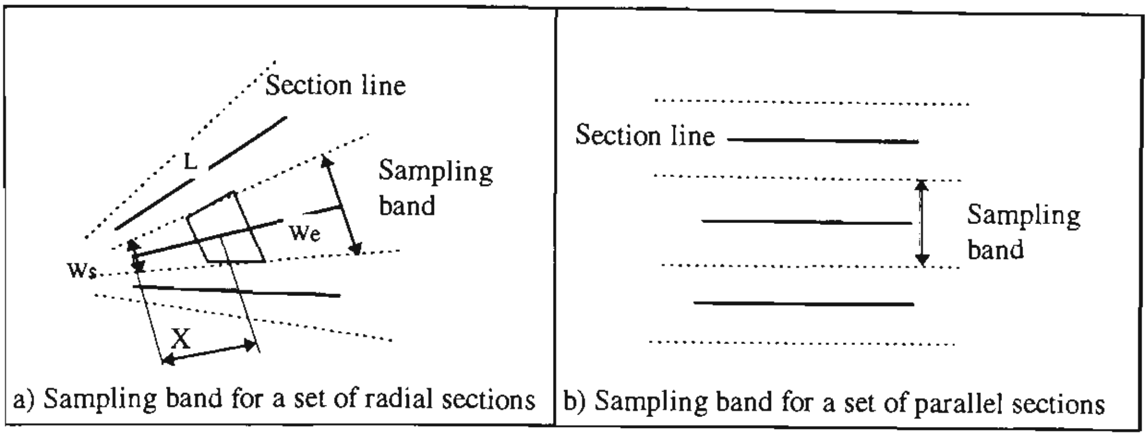


Figure 5.13- Concepts used for sampling band for different types of sections.

Once an area is calculated in cross section, it can be converted to volume using the related width for that specific area. This width is called the width of influence. For the parallel sections, the width of influence has a constant value and is defined as the summation of two half widths from both sides of the section. In the case of radial sections the width of influence is variable and its value is a function of the distance of the calculated area from the section origin and the information which is stored into a file called WIDTH.TAB. An example of such a file for two sections S1 and S2 is given in Table 5.8.

Table 5.8- An example of a width file (WIDTH.TAB) content.

S1	0.00	5.71	308.01	30.532
S2	0.00	6.12	312.12	31.243
S3	0.00	6.15	309.51	30.532
S4	0.00	6.02	310.42	31.243

The contents of a WIDTH.TAB file are as follows:

column 1 = section name,

column 2 = the starting point offset from the section origin,

column 3 = width of sampling band at the starting point (*Ws* in Figure 5.3a),

column 4 = distance from the section origin for the end point, and

column 5 = width of sampling band at the end point (*We* in Figure 5.3a).

The following formula is then used to calculate the width of influence for an area at distance X from the section origin by accessing the information stored in WIDTH.TAB file.

$$Wx = \frac{X \times (We - Ws)}{L} + Ws \qquad \text{(Radial Sections)} \qquad (5.3a)$$

$$Wx = Ws = We \qquad \text{(Parallel Sections)} \qquad (5.3b)$$

where:

Wx = width of influence in X distance,

X = distance from the section origin,

Ws = width of the sampling band at start point,

We = width of the sampling band at end point, and

L = length of the section.

5.4 SUMMARY

For geological data first a gridded seam model was developed which used digitised data and data from boreholes to establish a geological database for topography surface and roof and floor surfaces of the coal seam(s). These gridded surfaces were then integrated with cross sections to generate strings which were required during the simulation phase. The intersecting strings are stored into the ASCII formatted files to be loaded later by the computer routines for the simulation of dragline operations.

CHAPTER SIX

SIMULATION OF DRAGLINE OPERATIONS

6.1 INTRODUCTION

The productivity of a dragline operation is the result of two types of calculations, calculation of the quantity of waste being removed and estimation of the time required for its removal. The main purpose of the dragline simulation phase, described in this chapter, is to address the volumetric calculations and also to provide the data required for the time estimation process. The basic information required for the productivity calculation from the dragline simulation model is the volume of cut units, swing angle and the hoist distance required to move that particular volume, and the dragline walking distances both within and between the blocks. The calculations of these parameters are used as the basis in development of the dragline simulator.

6.2 ELEMENTS OF DRAGLINE SIMULATION MODEL

In a dragline operation removal of a block of overburden is carried out as a planned sequence of digging and dumping operations, with the machine walking to a new position between operations. To simulate removal of a mining block, first the volume of a single block is divided into sub-volumes or units. The next step is to simulate the dragline actions and the sequence of its operations in a logical manner. A dragline operation can be defined as the removal of a specified volume of overburden from a particular dragline position and dumping the material into the spoil area. There are several interrelated tasks essential to remove and dump a block of overburden. The dragline simulator developed in this thesis performs five basic tasks which serve as the core of the dragline simulation model. These tasks are the following:

1. design of initial pit and definition of the dragline mining blocks,
2. division of the bulk volume of the blocks into the unit volumes (sub-volumes), and calculation of the optimum positions of the dragline and the walking patterns for removal of a unit,
3. calculation of prime or rehandle volume and centroid point of the cut unit,
4. calculation of the final shape and centroid point of the spoil, and
5. calculation of the swing angle and hoist distance for each unit volume.

6.2.1 Initial Pit Design

After constructing the geology of the sections in the form of initial strings, simulation of the overburden removal operation commences with establishment of the initial strip mine design. This involved the development of numerous computer routines in order to automate the whole strip mine process. The process of the pit design in the CADSIM model includes the design of a truck and shovel level, overhand chop and main dig depth calculations from spoil balance, the design of post blasting profiles, definition of the dragline passes, and determination of mining blocks.

The CADSIM model allows two types of calculations for the pit design. The design parameters such as shovel and truck working level, chop depth and location of the main cut toe points can either be input as data or be calculated by the simulator. A combination of the two methods is also possible. For example, a toe point can be read from an input file as a starting point for the pit design, while the calculation of the total dragline depths including main dig and chop depths may be calculated by the simulator.

The location and geometry of the strips can be generated by digitising the strip layout as strings on the plan of the mining area. The toeline points are then determined from the intersection of the strip lines with plane of the section mounts. The intersecting points for each strip are stored into a single file for easy access by the simulator. The intersecting angles are also stored with the toe points to determine whether a section mount is oblique to the strip line or not. During the simulation the toe points are used as starting points to produce the pit geometry. After calculation of the position of the toe points, the dragline cut details are generated by the simulator.

6.2.2 Dragline Positions

The bulk volume, or the volume of a single digout is divided into subvolumes to provide more accuracy in the required calculations. For each unit volume, it is necessary to determine the optimum dragline location and optimum position for spoil dumping in order to calculate the swing angle and hoist distance. The main purpose of dividing a mining block into subvolumes is to simulate the overburden removal operation accurately. The number and shape of divisions are governed by local geology, digging method and different positions of the dragline. For the purposes of this thesis, the divisions are made based on particular dragline methods and considering the number of lifts in each pass. An example of the division of units for a two-pass split bench method is shown in Figure 6.1.

The positions of the dragline during removal of a block greatly influences the swing angle and filling time. The swing angle and hence cycle time is reduced if the dragline is correctly positioned at the specific cut units of a block. The dragline often starts removal of a block by walking near to the new highwall in order to make the key cut. From there it gradually walks toward the edge of the pit or the bridge to dump the remaining parts of the block. Moving the dragline to the edge of the pit increases the dragline reach for spoiling, but it may also increase the swing angles. Many operators move the dragline more often to achieve better operation efficiency and smoother dumping of the spoil (Wu, 1990).

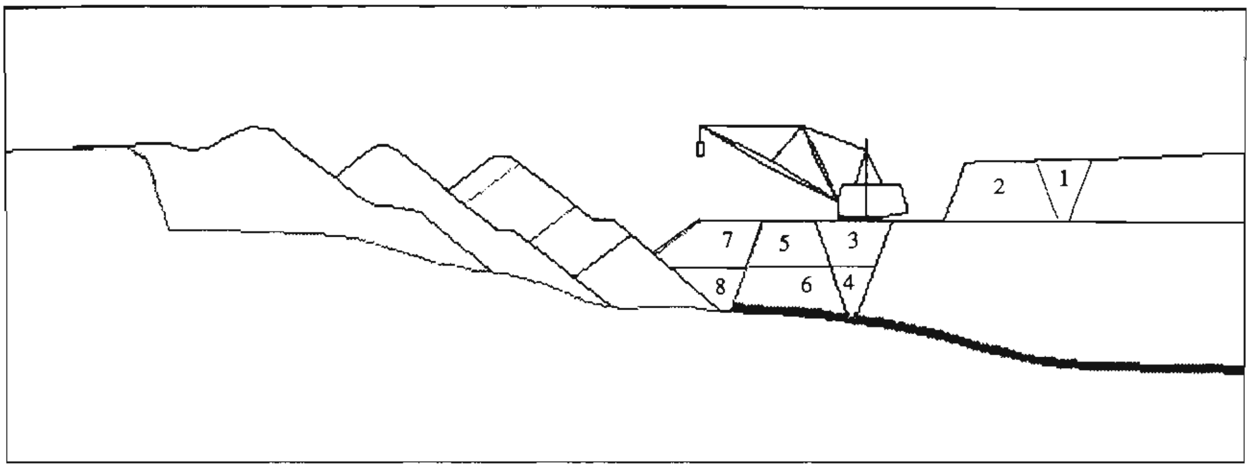


Figure 6.1- Subdivision of the mining blocks.

In order to determine the dragline positions during the excavation of a cut, it is necessary first to establish the desired shape of the cut. Different dragline positions are calculated corresponding to the cut shape and the criteria set in the specific routines while running the simulator. This is because the critical points such as crest and toe points are not known until the simulator is run. For an example, the position of the dragline while excavating a key cut is calculated in a multi stage process as described below. The concepts used during these stages are shown in Figure 6.2.

1. Toe point coordinates are read by the simulator and stored as a reference point, TOEOLD, to start the pit design.
2. A temporary point, TMP1, is created by offsetting from the original point TOEOLD at a strip width distance to the right hand side.
3. TMP1 is projected down (or up if no intersection is found) vertically to intersect top of coal string. The intersecting point is set to the new highwall toe, TOENEW.
4. A new temporary point, TMP2, is defined by offsetting from TOENEW on top of coal to the left hand side at a half bucket width distance (BW/2). TMP2 determines the middle point of the key cut at the bottom.

5. TMP2 is then projected up vertically to intersect the dragline working level (e.g. original topography). The intersection point is set to the dragline position for excavating the key cut, KEYPOS.

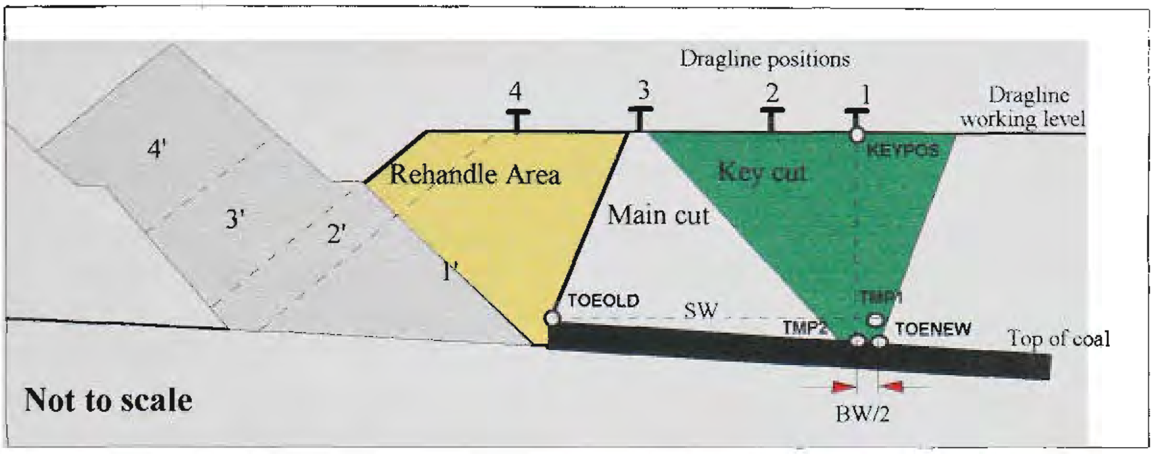


Figure 6.2- Concepts used to locate a dragline position.

For the purpose of walking calculations, a two dimensional Cartesian system is first chosen on the plan for defining of the dragline locations. As shown in Figure 6.3, the "X" coordinate is measured along the section advancing in the mining direction and "Y" is measured in the direction of the strip line.

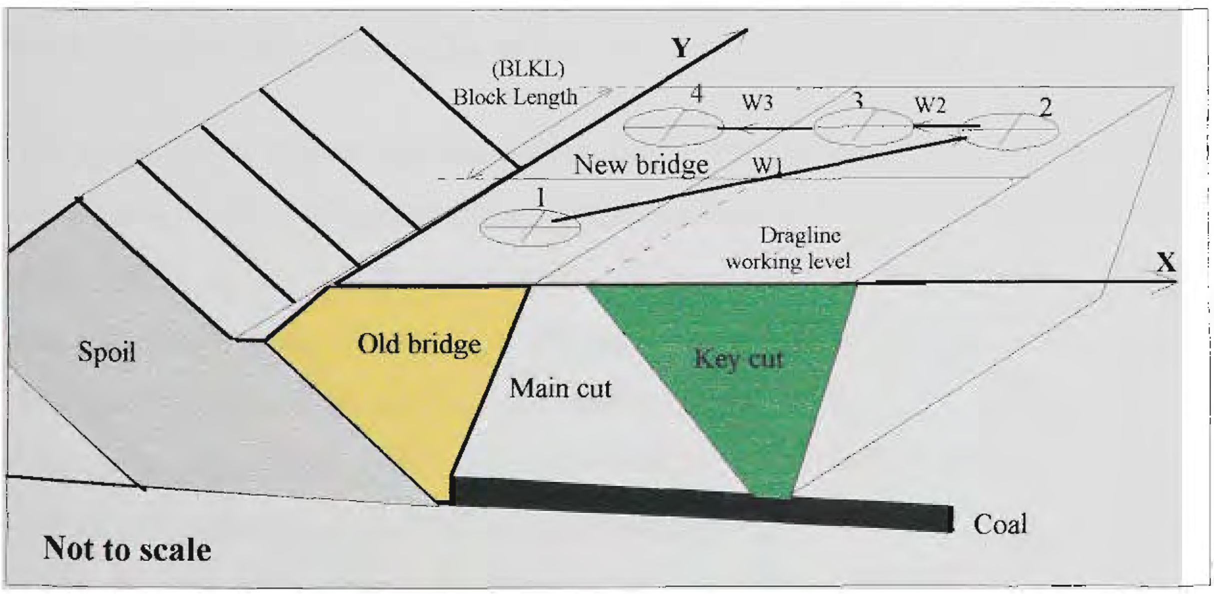


Figure 6.3- Calculation of the dragline walking pattern.

The following formula can be used for calculation of walking distances:

$$W_t = \sum_{i=1}^{n} \sqrt{\left(X_i - X_{i+1}\right)^2 + \left(Y_{in} - Y_{i+1}\right)^2} \qquad (6.1)$$

where:

W_t is total walking distance and X_i, Y_i are coordinates of the dragline's i^{th} positions.

For example in Figure 6.3, the total walking distances can be calculated as follows:

$$W_t = W_1 + W_2 + W_3$$

$$W_t = \sqrt{(X_1 - X_2)^2 + (Y_1 - Y_2)^2} + \sqrt{(X_2 - X_3)^2 + (Y_2 - Y_3)^2} + \sqrt{(X_3 - X_4)^2 + (Y_3 - Y_4)^2}$$

In the above example $(Y_1 - Y_2)$ is equal to the block length and $(Y_2 - Y_3)$, $(Y_3 - Y_4)$ are equal to zero.

$$W_t = \sqrt{(X_1 - X_2)^2 + BLKL2^2} + ABS(X_2 - X_3) + ABS(X_3 - X_4)$$

Where BLKL is the block length and the (ABS) function calculates the absolute value of the arguments.

6.2.3 Volumetric Calculation of the Cut Units

The original cut shapes and excavation limits are generated from strings which are determined by the simulator routines from the criteria set by the user. For instance, a cut shape may be decided on the basis of the volume of material needed to build the bridge. Unlike a trigonometrical approach, the use of strings as a base for the modelling and design of the various cut and spoil profiles is not limited to the regular structures. The basic geometry of the cut profile is determined from the location of toe and crest points and the projection of these points at a nominated angle such as slope angle. The critical corner points may be generated by the intersection of the strings.

Normally, the design of an excavation profile begins from an input point such as a toe point. This initial point is then moved on a particular string (e.g. top of coal) at a specified distance and projected to the top or bottom surfaces in order to make part lines and the new points. A new string is then generated by joining these points and part lines together to generate the profile of a cut. For the purpose of volume calculations the profile of the cut must be combined with existing strings to form a closed string of the cut (Figure 6.4).

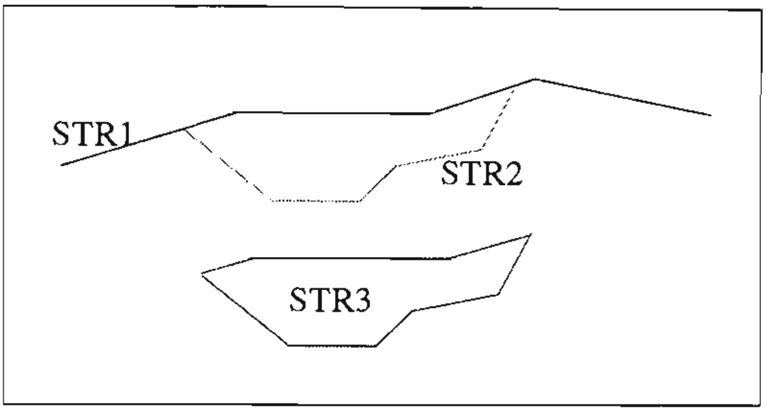


Figure 6.4- Intersecting two open strings (STR1 and STR2) to generate a closed string (STR3).

Referring to Figure 6.5 the area formed by a closed string can be calculated using the following standard equation.

$$Area\ ABCD = area\ EABF + area\ FBCH - area\ GDCH - area\ EADG$$

$$2 \times Area\ ABCD = [(Y_A + Y_B)(X_B - X_A)] + [(Y_B + Y_C)(X_C - X_B)] - [(Y_C + Y_D)(X_C - X_D)] - [(Y_D + Y_A)(X_D - X_A)]$$

Multiplying these values and rearranging the results yields:

$$A = \frac{X_A(Y_D - Y_B) + X_B(Y_A - Y_C) + X_C(Y_B - Y_D) + X_D(Y_C - Y_A)}{2}$$

Where A is the area of ABCD.

The same rule can be extended for a closed string with n points as below.

$$A = \frac{X_1(Y_n - Y_2) + X_2(Y_1 - Y_3) + \ldots + X_n(Y_{n-1} - Y_1)}{2}$$

$$= \frac{\sum_{i=1}^{n} X_i (Y_{(i-1)} - Y_{(i+1)})}{2}$$

$$= \frac{Y_1(X_2 - X_n) + Y_2(X_3 - X_1) + ...+ Y_n(X_1 - X_{n-1})}{2}$$

$$= \frac{\sum_{i=1}^{n} Y_i (X_{(i+1)} - X_{(i-1)})}{2} \qquad (6.2)$$

It must be noted that n+1 equals 1 and 0 equals n in the above formula.

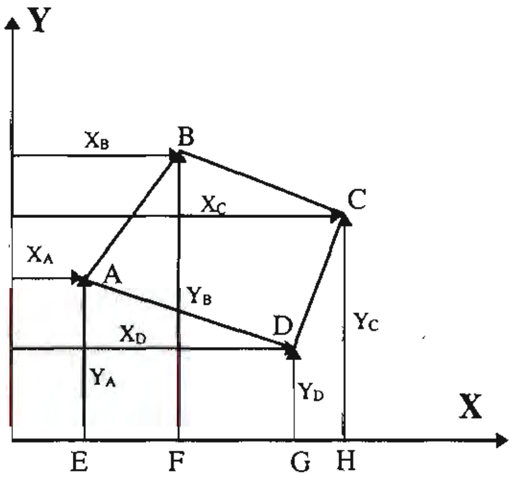


Figure 6.5 - Area calculation for a polygon using coordinates.

At any stage and for debugging purposes the resultant string or points can be displayed using the graphic device of the software to check the position and profile of the generated cut unit.

Referring to Figure 6.6, the centroid of an irregular area is obtained after it is divided first into regular shapes such as rectangles and triangles.

The centroid of an integrated area is determined as follows:

$$\overline{\overline{X}} = \frac{\sum_{i=1}^{n} \overline{X}_i A_i}{\sum_{i=1}^{n} A_i} \quad \text{and} \quad \overline{\overline{Y}} = \frac{\sum_{i=1}^{n} \overline{Y}_i A_i}{\sum_{i=1}^{n} A_i} \qquad (6.3)$$

where: $\overline{\overline{X}}$ and $\overline{\overline{Y}}$ are coordinates of the overall centroid of the area.

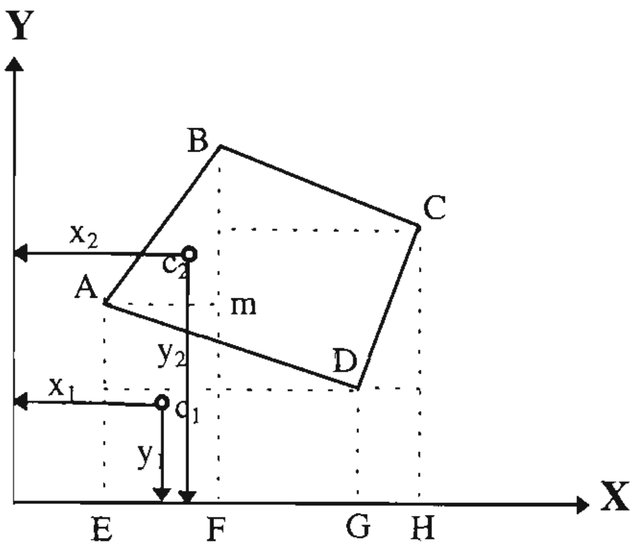


Figure 6.6- Calculation of centroid point of an area.

Multiplying these values (using Equations 6.2 for the areas) for a closed string with *n* points and rearranging the results yields the following equations:

$$\overline{\overline{X}} = \frac{\sum_{i=1}^{n} X_i^2 (Y_{i-1} - Y_{i+1}) \ + \ \sum_{i=1}^{n} X_i X_{i+1} (Y_i - Y_{i+1})}{6 \sum_{i=1}^{n} X_i (Y_{i-1} - Y_{i+1})} \qquad (6.4a)$$

$$\overline{\overline{Y}} = \frac{\sum_{i=1}^{n} Y_i^2 (X_{i+1} - X_{i-1}) \ + \ \sum_{i=1}^{n} Y_i Y_{i+1} (X_{i+1} - X_i)}{6 \sum_{i=1}^{n} Y_i (X_{i+1} - X_{i-1})} \qquad (6.4b)$$

One of the main areas of concern in any dragline operation is the amount of the rehandle that is included in the total overburden removal. Rehandle is the quantity of material that must be handled more than once before being placed in its final position. It is expressed as a percentage of the prime waste. In this thesis the percentage of rehandle material is calculated as follows:

$$Rehandle(\%) = \frac{Equivalent \quad Prime \quad Volume \quad Rehandled}{Prime \quad in\text{-}situ \quad area} \times 100 \qquad (6.5a)$$

where:

$$Equivalent\ Prime\ Volume\ Rehandled = \frac{Volume \quad Rehandled}{Swell \quad factor} \qquad (6.5b)$$

6.2.3.1 Design Limitations

A dragline cut block can be modelled as a geometric entity bounded by up to six planes (Figure 6.7). Normally gridded surfaces are used for the top and bottom planes and the accuracy of these planes is only limited to the mesh size selected in the geological model. The left and right hand side walls can have different angles and may even be folded. Both the front and rear planes are restricted to a vertical angle since they are parallel to the plane of the section mounts. Figure 6.7 shows the planes used to create a key cut.

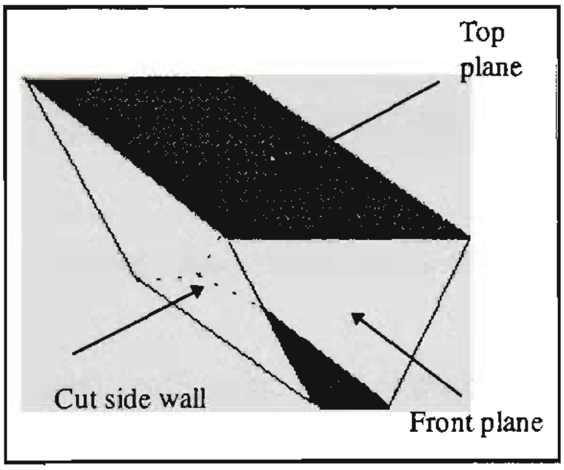


Figure 6.7- Planes used to form a key cut.

6.2.4 Spoiling Calculations

Spoil profiles can be developed by the use of the string structure. The following three major dumping procedures were considered for a dragline operation in the CADSIM model:

1. normal spoiling (side casting),
2. bench filling, and
3. dumping away from a ramp.

Two methods of generating spoil structures were applied to the design of the spoil profiles. In the first method individual slices of a spoil pile are modelled. This offers the flexibility of the specific placement of spoil in a small area and in an area where a high degree of accuracy is required. The geometry of an individual slice of a spoil pile is calculated from the position of the dragline, the predefined angle of repose and the volume of material being spoiled. Generation of the slices begins from where the dragline can dump material with the shortest possible swing angles. The process then continues until the total volume waiting for spoiling is dumped or one of the restrictions for the dragline reach or maximum dump height is met. An example of making a dragline pad using spoiling slices is shown in Figure 6.8.

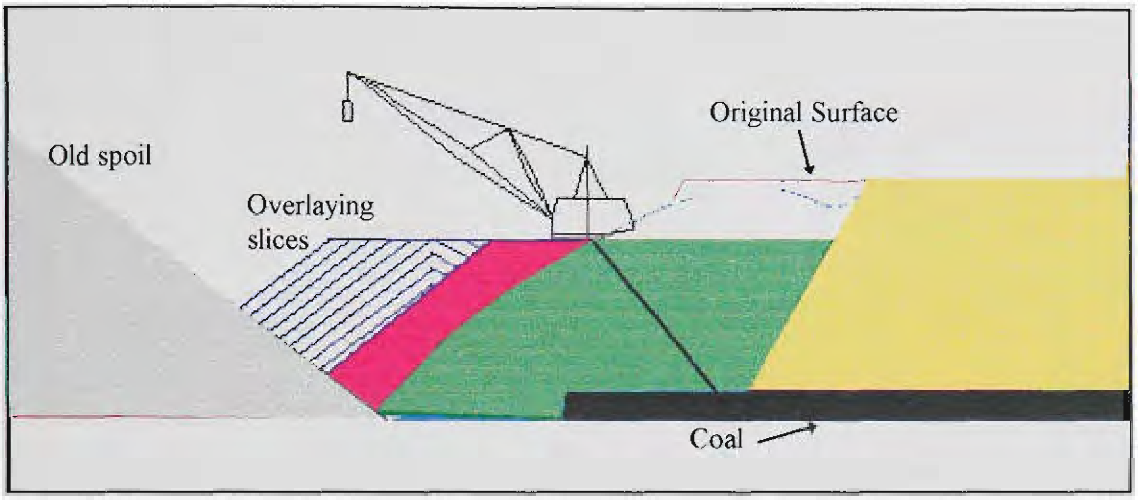
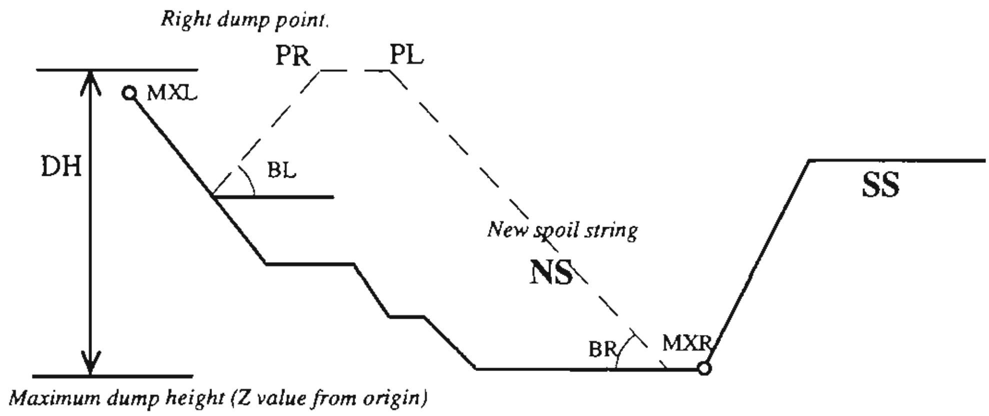


Figure 6.8- Creating a dragline pad using spoiling slices.

Alternatively, the design of spoil profiles can be modelled using the "SPOIL" function in DSLX's language. Because larger volumes of material can be spoiled in each step, this option provides a quicker method of designing dragline spoil placement. The concepts used in the "SPOIL" function are described in Figure 6.9.

The "SPOIL" function is quite complex and has 15 arguments which control the spoiling limitations before calculating the volume of the final spoil profile. All the spoiling restrictions such as maximum dump point, dragline reach, toe limitations, repose angle and original base string are defined through the arguments. Some arguments are also allocated for volumes that include the original volume being dumped, the available volume for spoiling calculated by the function, and the residual volume. For example, if spoiling met one of its restrictions, say maximum dragline dump, the calculation is terminated and remainder of the material is reported as residual volume.

The last four arguments (*VC, VR, HF and NS*) are returned by the function while all the others are required as input. Volume tolerance and dump tolerance are used to control the precision of calculation. For example, if the dump tolerance was set to one, the function would assume that if the spoil dump height was within a metre of the maximum height then no more volume could be added.



SPOIL (SS,PL,PR,VD,DH,BL,BR,MXL,MXR,VT,TD,VC,VR,HF,NS)

SS = Current spoil string
PR = Right dump point.
PL = Left dump point.
VD = Volume to be dumped.
DH = Maximum dump height
BL = Left slope angle
BR = Right slope angle
HF= Final dump height.

MXL= Left spoil toe limit (X value from origin)
MXR = Right spoil toe limit (X value from origin)
VT = Volume tolerance.
TD= Dump height tolerance.
VC = Computed volume.
VR = Residual volume (VR = VD-VC)
NS = New spoil string

Figure 6.9 - Arguments used in "SPOIL" function.

6.2.5 Swing Angle and Hoist Calculations

Swing angle and hoist distance are critical parameters in the calculation of the cycle time. For each of the cut units the related swing angle and hoist distance can be calculated based on the centroid of the cut and spoil profile and the position of the dragline. It is assumed that the centroid point is the average for the bucket positions during either an excavation of a cut or dumping the spoil. In other words, on average, the dragline boom begins its arc of swing when it lies over the centre of gravity of the material which it must dig from that position.

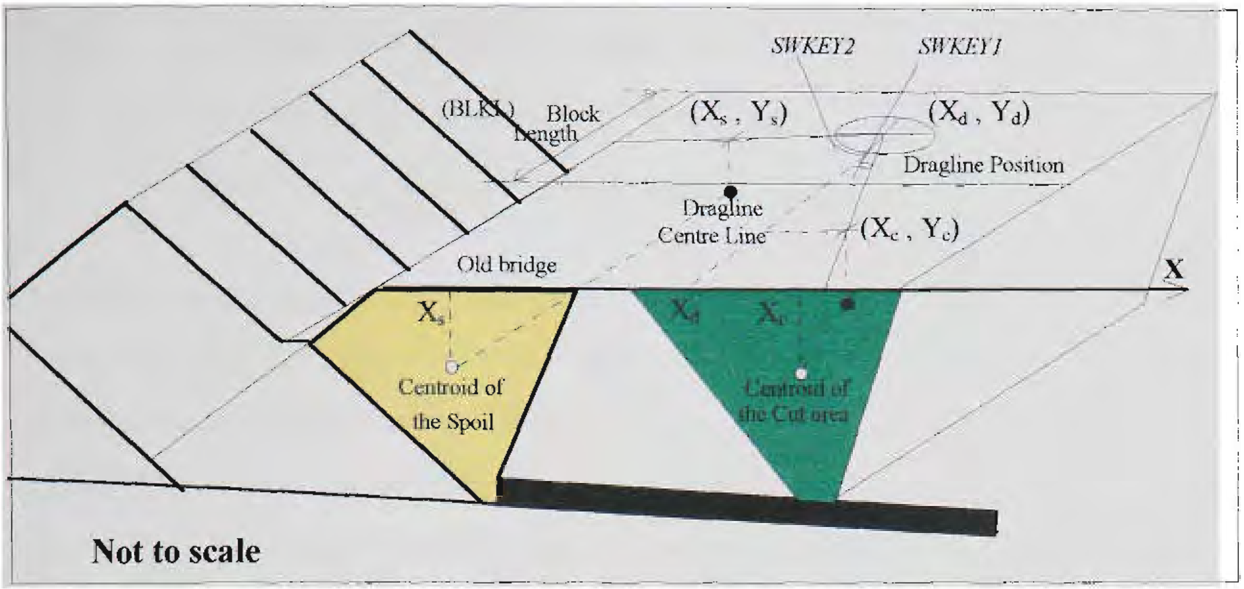


Figure 6.10- Concepts used for the swing angle calculation.

Referring to Figure 6.10, the swing angle is made of two angles *SWKEY*1 and *SWKEY*2. *SWKEY*1 is called dig angle and *SWKEY*2 is dump angle. Assume X_c and Y_c are the coordinates of the centre of gravity of the material to be dug (ie. key cut). These are calculated from the centroid calculation of the area from the cross section and the block length. The same concept can be used for calculation of the spoil centroid coordinates X_s and Y_s. The first angle *SWKEY*1, can then be calculated as follows:

$$SWKEY1 = tan^{-1}\left(\left|\frac{X_c - X_d}{Y_c - Y_d}\right|\right) \tag{6.6a}$$

Similarly, the dump angle:

$$SWKEY2 = tan^{-1}\left(\left|\frac{X_s - X_d}{Y_s - Y_d}\right|\right) \tag{6.6b}$$

where X_d and Y_d are the dragline position coordinates.

The total swing angle is then:

$$SWKEY = SWKEY1 + SWKEY2$$

Calculation of the hoist distance is based on the elevation of the centroid points of the cut and spoil profiles and dragline working level.

$$\begin{aligned} \textit{Hoist Distance} &= (Z_s - Z_d) + (Z_d - Z_c) \\ &= Z_s - Z_c \\ &= 0 \qquad \textit{if } Z_s < Z_c \end{aligned} \tag{6.7}$$

where: Z_d = the elevation of the dragline position,

Z_c = the elevation of the cut centroid, and

Z_s = the elevation of the spoil centroid.

In some situations it may be necessary to separate each swing angle into two separate parts. The first part is the swing and hoist required to clear the bucket from the crest of the cut and the second part would then be the swing from this crest to the final dump point. This is because in removal of a narrow and deep cut (e.g. a key cut in a deep overburden) the dragline makes a short swing, but a long hoist. In this case the time required for hoisting may exceed the swing time. It other words, the dragline cannot start the second part of swing until it clears the bucket from the crest. To include this effect and to compute the hoist dependency of both parts of a complete swing, the related values for swing angle and hoist distances are calculated and reported as first and second swing angles, and first and second hoist distances.

In addition to the usual dragline calculations, several procedures have also been developed in this thesis to solve the common problems associated with simulation of a dragline operation. Four design procedures most significant in the dragline simulation process are described below. These are the design of coal access ramps, curvature strips, dragline walking grade and post blasting profiles.

6.2.6 Design of Coal Haulage Ramps

The calculation of the rehandle percentage from the 2D range diagrams in the conventional approach assumes that the material is placed only once within the rehandle portion of the pit cross-section. In some instances this assumption may not be correct, since it is possible to double rehandle the spoil. The most common case of double rehandling is around haulage ramps where spoil must be carried along the pit to clear the ramp. Even beyond the influence of the ramp additional double rehandle can occur if the pit is very wide and spoiling is tight.

In the vicinity of a ramp a different procedure is used to place extra material on the spoil pile due to the inadequacy of the spoil room. In order to include the effect of access ramps in the simulation, a reference surface is used in conjunction with the other gridded surfaces to determine whether a limitation exists for spoil placement. Any other limitations for spoiling such as maximum spoil height for spoil stability can also be included. The reference surface is a permit surface that indicates the available spoil room in each region. During the simulation, the material should not be spoiled above the permit surface. This surface is represented as a string in each section and there is always a final check for spoiling against this string. For example, if a section is at the centre line of the ramp, the reference surface is almost the same as the original cut string and this means that no spoil can be placed in the void of the old pit.

Figure 6.11 illustrates how the reference surface changes as the sections approach the ramp and pass it. In this example, when sections progress further from the ramp, the reference string changes until it reaches a steady state in regions unaffected by the ramp. The reference surface for each section is also stored with other characteristics of the section. The cumulative extra volume from sections affected by the ramp is carried along the strip until it can be dumped in the sections with more available spoil room. This volume is usually reported as ramp rehandle.

6.2.7 Design of Curvature Strips

When mining is started along the coal outcrop and with a rolling topography, the dragline pit may be designed so that it follows a uniform contour. As a result, this type of design may develop pits in a curved shape. Where curved pits are designed, a series of inside and outside curves are usually encountered. One criticism of traditional 2D range diagram calculations is that they do not work well with curved strips. The major problem with volumetric calculation of a curved strip is the difference in available area for spoiling between an inside and outside curve. A curve causes a variable width between two adjacent sections along the section line and therefore a constant width cannot be used to convert calculated areas to volumes (Figure 6.12). When the strip is curved, the sections are radial and a *curvature correction* must be applied to the volume calculation. The correct volume is determined from where the distance between the sections is taken along the path of the centroid (Uren and Price, 1989).

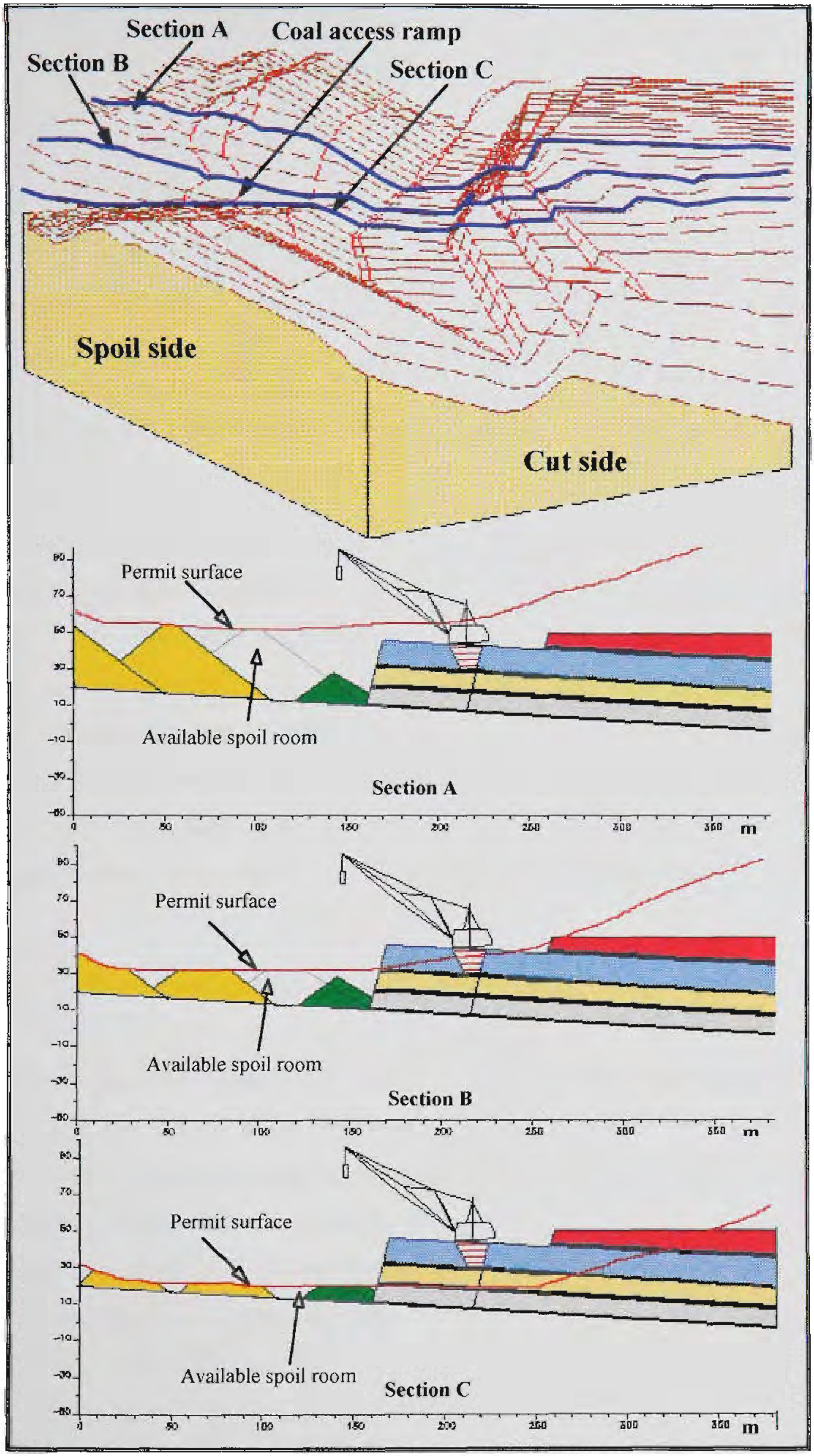


Figure 6.11- Effect of permit surface on the available spoil room in vicinity of a ramp.

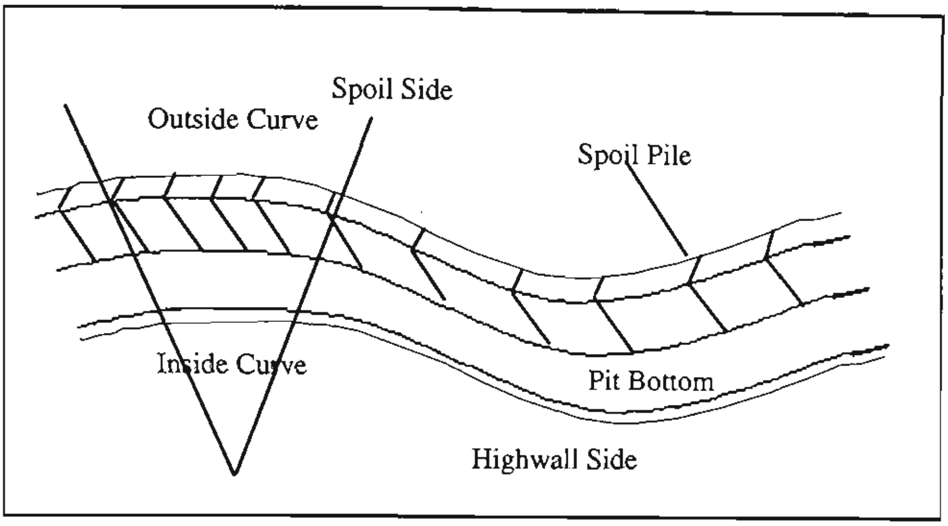


Figure 6.12- Effect of pit curvature on volume inside and outside the curve.

To overcome the problems caused by curves, a variable width along the length of a section is used by the CADSIM model. As soon as an area is calculated, the dragline simulator calculates the centroid of the area as well. The simulator then uses the start and end coordinates of the section lines to create a table of width information. By subtracting the related values of the coordinates and computing the length, a ratio can be calculated for each section. Using the X value of the centroid point and the width ratio for the section, the width of influence can be determined for the calculated area. The width of influence is then used to convert the 2D areas into 3D volumes.

6.2.8 Walking Grade Control Between Mining Blocks

Walking draglines are limited to a maximum walking grade. The dragline gradeability has a specific influence on the minimum ramp length required for any dragline level changes and thus on the amount of pre-strip material for dragline access. A dragline with higher gradeability can be used in rugged topography for pit access with a minimum of earthworks (Seib and Carr 1990). When simulating the dragline operation for an entire deposit, the dragline simulator must be able to measure and control the grade for the dragline movements.

By running the simulation along a strip, section by section, the volumetric information and working levels of a section can be used in designing the geometry of the next section. Maximum grade is then controlled by comparing the average dragline level for adjacent sections and the maximum grade at which a dragline can operate. In certain situations, particularly in rugged topography, the grade between sections may exceed the maximum allowable grade for the dragline. In these cases, either a chop and fill operation or a modification in working level is required to provide the necessary elevation changes between moves.

In the simulation of a full strip it may be found that material carried from simulated blocks and also required working elevation cannot be achieved on subsequent sections due to the insufficient spoil room and gradeability of the dragline. These problems can be solved through changes in working levels. To allow easy modification for dragline working level a feedback mechanism was adopted for running the dragline simulation in the CADSIM model. In this feedback mechanism, first the simulator is run without reading the input information for working levels. In this case the simulator calculates dig levels based on a spoil balance procedure and reports the calculated levels into an output file called DIGREP.TXT. Table 6.1 shows a part of a dig report file which gives information about the dragline working level in both main and chop passes.

Table 6.1- An example of an output dig report file.

Block	Main Pass					Chop Bench				
No	Easting	Northing	R.L.	Grade	Depth	Easting	Northing	R.L.	Grade	Depth
6	245736	7320559	217.5	-2.6	38.7	245776	7320562	235.4	0	0.0
7	245737	7320534	217.0	-2.8	39.1	245777	7320536	236.7	5.0	2.0
8	245737	7320509	214.1	-2.8	39.7	245778	7320511	238.0	5.0	4.0
9	245737	7320484	212.5	-4.4	39.9	245779	7320486	237.5	5.0	6.4
10	245737	7320459	211.1	-5.1	40.0	245780	7320461	236.1	5.0	9.0
11	245737	7320434	209.8	-5.1	40.0	245780	7320437	234.8	5.0	11.6
12	245737	7320409	208.5	-5.1	40.0	245781	7320411	233.5	5.0	14.2

The output file can then be modified by the user to include the desired grade and also to maximise dragline waste, if there is still room for spoiling. The user may also change the working levels so that the ramp rehandle can be minimised. For example, the chop level can be reduced in the vicinity of a ramp and gradually increased as the sections pass the area effected by the ramp. This information can then be used as an input in re-running the simulation. The volumetric calculations are repeated and the results again written into an output file. This process may be repeated several times to arrive at the best solution for a specific pit design and geological condition.

6.2.9 Design of Post Blasting Profiles

A dragline simulator must be capable of handling blast parameters and predicting the thrown percentage in the final spoil position. Simulation of throw blasting results can be carried out by the CADSIM model in two ways. First an existing blast profile can be measured by survey techniques and converted to a triangulated surface. This surface is then used to generate post blasting profile strings for simulation on each section. Alternatively, when the blasting profile has not been recorded, specific routines of the CADSIM model predict the blasting results and fit the profile to the current pit based on pit geometry, swell factor and input thrown percentage. In either case, the simulator then measures blast performance relative to the dragline operation.

Relevant data calculated from pre-blast and post-blast profiles include:

1. percentage throw (moved to final spoil),
2. vertical and horizontal heave,
3. changes in dragline dig depth,
4. rehandle volumes, and
5. pad preparation requirements by dozer.

In order to establish the post blasting profiles for case study simulations, the results from a blasting prediction computer package (ICI Explosives SABREX) were used for various drilling patterns and powder factors. Once the final desired profile was determined the profile was read into the program as a string for simulation of the different dragline operations. Figure 6.13 illustrates the simulated profile provided by the CADSIM model and Figure 6.14 is a photograph of the actual throw blasting profile.

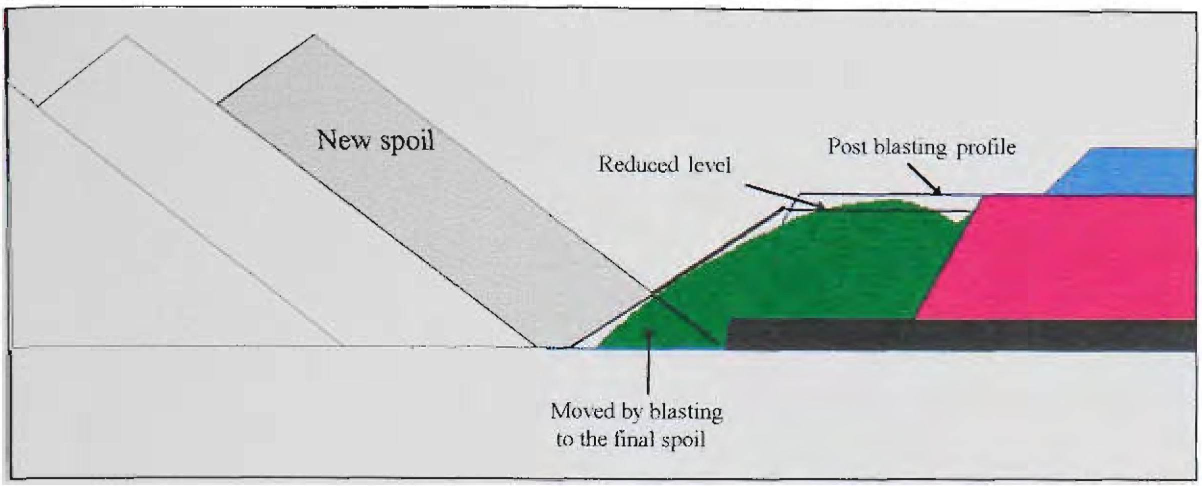


Figure 6.13- An example of a simulated post blasting profile.

Thrown percentage is used to show what proportion of prime overburden is removed by blasting. The thrown expressed in percentage of prime material can be calculated as follows:

$$Thrown\ (\%) = \frac{Equivalent\ prime\ volume\ moved\ by\ blasting\ into\ final\ spoil}{prime\ in\text{-}situ\ volume} \times 100 \quad (6.8a)$$

where:

$$Equivalent\ prime\ volume = \frac{Volume\ \ moved\ \ into\ \ final\ \ spoil}{Swell\ \ factor} \quad (6.8b)$$

Figure 6.14- A throw blasting profile.

6.3 THE CADSIM DRAGLINE SIMULATOR

To implement the procedures described in the previous sections, a number of computer programs were designed, coded and debugged. The computer routines read all of the relevant input data generated in the geological phase to simulate the digging, spoiling and walking patterns of the dragline operation, to write the final reports and to provide data for 3D outputs. The simulation of the various stripping methods includes the extensive use of data gathered from a survey of digging methods used in Australian strip mines. Based on the information gathered in the digging method survey, seven major modules were developed, each one addressing a specific dragline digging method. A listing of the computer programming codes is provided in Appendix B. All of the routines include the use of comment statements (lines start with "!") to aid in clarifying the logic and calculation procedures.

A modular programming approach was employed (Figure 6.15). This enables the use of common routines (e.g. calculation of spoil available) in different programs. Each program consists of a main routine and several major subroutines called from within the main routine and a number of additional subroutines at the next levels. Most of the inputs to the program are in a batch mode although an interactive input mode can also be selected. This will allow more automation in running the program.

All the programs developed to simulate the dragline operations were written in DSLX language which runs under UNIX operating system on workstations such as Sun, Silicon Graphic and PC Solarise. A minimum of 64M bytes memory is required and any extra memory will speed up the graphics presentation and program compiling and running time. The total time required for compiling and running the programs depends on the program size, number of sections and strips and hardware configurations. For example, the module EXTBENCH, with most executable codes can be run under ten minutes for a medium size mine (20 strips and each strip 60 sections) on a Sun Ultra (or Silicon Graphic Otoo) machine. This time consists of three minutes compiling and less than seven minutes program run.

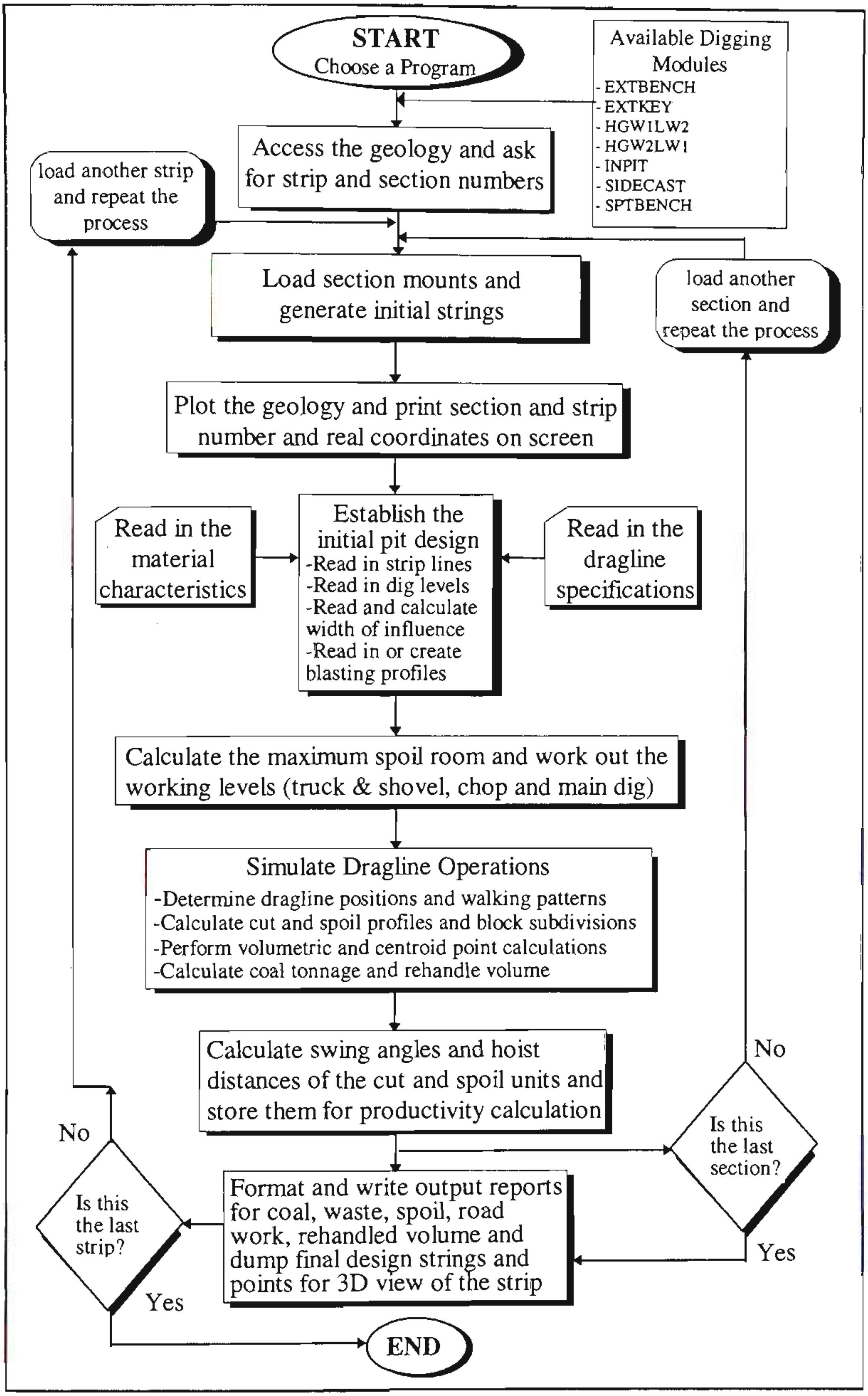


Figure 6.15- The general flowchart of the computer programs developed in the CADSIM model.

Dragline digging methods are simulated using specific sub-programs while the main program accesses the geological information and input data including the material characteristics and dragline operating parameters. The following seven main programs are available to the user:

1. EXTBENCH : simulates a standard extended bench method. This program also includes the use of an advance bench (chop bench).
2. EXTKEY : simulates an extended key cut method for a single seam.
3. HGW1LW2 : simulates single highwall and double lowwall pass method.
4. HGW2LW1 : simulates double highwall and single lowwall pass method.
5. INPIT : simulates an in-pit bench method for a single seam.
6. SIDECAST : simulates a simple side casting method including an advance bench.
7. SPTBENCH : simulates a split bench method in two passes to remove a thick overburden covering a single coal seam. This program can be also used for a two coal seam operation.

6.3.1 Running a Simulation

A simulation starts with loading and compiling a main program (e.g. EXTBENCH) from the disk. All the developed programs start the simulation by accessing the geology through the use of strings. Once the initial strings representing the geology of the section have been retrieved and plotted on the screen, a typical dragline pit design is started by reading the toeline data for the first strip from the input files. For example, in Figure 6.16, the toe point of the old highwall is used as the starting point to build up the pit geometry.

With subroutine "DRAG" (called by the main program), the user can specify the dragline dimension to be used for the simulation. The dragline specifications are read from the input file and a scaled icon of the dragline is drawn at the specified location. Material properties such as spoil repose angle, swell factor, highwall and chop angle are defined by reading subroutine "MATER" into the main programs. The dragline working levels are determined after the initial pit design. Normally, the maximum spoil room is used as a basis for the calculation of the various dig levels, although the pre-designed levels read from an input file can be used as default.

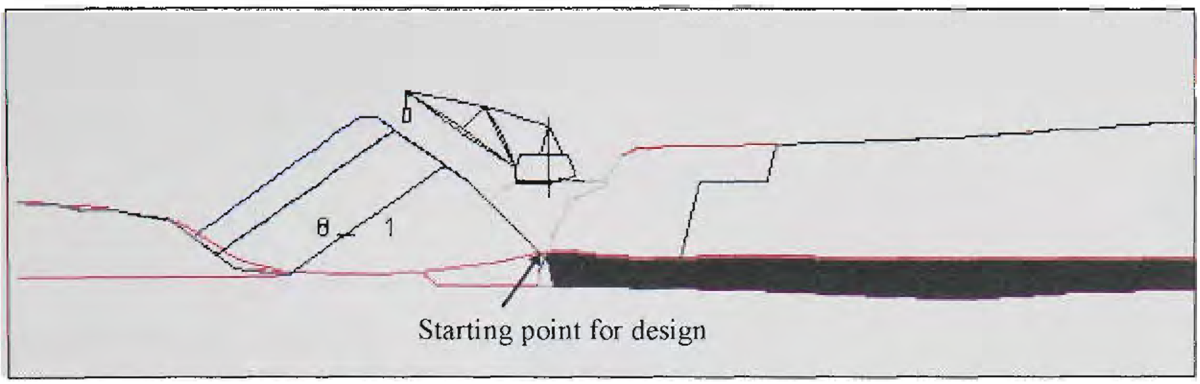
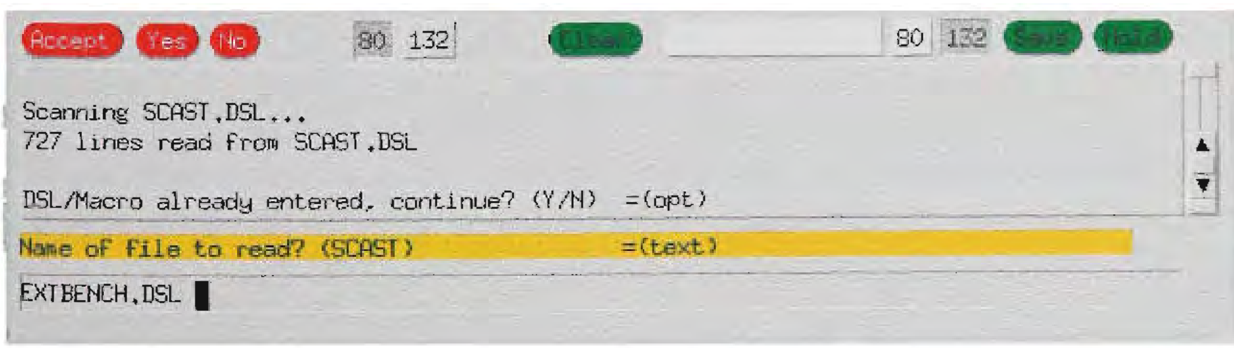


Figure 6.16- The use of the old highwall toe as the starting point for pit design.

6.3.2 User Inputs and Simulation Outputs

One of the goals of this thesis was to automate the simulation of a dragline operation by reducing the number of program interruptions by the user. Most of the input data must be prepared and stored into the ASCII files before a program can be executed. In addition, various options and flags are designed in the program to allow the user to run the program interactively. Below are example dialogue boxes during a simulation run.

- The name of the main program which selects the specific routine for the digging method to be employed:



- The model of the dragline is next entered:

```
 ACCEPT DEFAULT MATERIAL PARAMETERS (1=YES,0=NO) ?
Enter expression for "ians" : (1.00000000) =(text)
WHICH DRAGLINE DO YOU WISH TO USE (1350 or 8750)

Enter expression for "ians" : (8750.00000) =(text)
```

- The number of sections and strips to be simulated during a run must include the starting section and strip number; the simulation can start from an intermediate section:

```
Starting section number ?(0 = read from file)

Enter expression for "isect" : (0.00000000) =(text)
```

```
Starting strip number ?
Enter expression for "istrip" : (1.00000000) =(text)
 If manual control is selected you will be prompted at the end of
 each sequence of sections to determine if another strip is required
 Otherwise processing continues uninterrupted until the specified
 number of strips are completed.
Number of strips ? (default is manual control)

Enter expression for "nstrip" : (0.00000000) =(text)
```

- The name of the control files for the dig levels and width of influence:

```
 If dig levels are read from input file they override the calculated
 dig levels.
Dragline dig levels from file ?(1=yes,0=no,nn=fixed depth)

Enter expression for "ilevel" : (0.00000000) =(text)
```

```
 Volumes can be calculated using fixed length influences for
 all sections or variable influences read from a file
 The file is created using the WIDTH STRIP option under
 Section/Geo_Dump option in DSLX.
 SECTION WIDTH FACTOR (0 = use input file)

Enter expression for "volfact" : (0.00000000) =(text)
```

- The name of the toeline files:

```
The position of the toe of the first strip and the angle of
intersection of sections with the strip lines can be read from
a file. The file is created using the Digitise_Strips option
under Section/Geo_Dump option in DRGSIM. If these values are
not read from a file then 90 degree intersections are assumed
and the toe positions calculated automatically.
**********WARNING**********
In certain situations such as where an in pit bridge and therefore
no previous void exists, the automatic calculation of highwall
toe will fail. In these cases the toe line must be read from an
input file.
TOE LINE CALCULATED(0 = use input file)

Enter expression for "toeflag" : (0.00000000) =(text)
```

Some of the above input data such as the toeline, highwall and dig level files are optional. Input data are stored in memory and used as default values in the subsequent runs.

During the simulation of each section, the volume of material moved from each sub-component (e.g. top of key cut) along with the associated swing angles and hoist distances, coal volumes, spoil carried along the strip and rehandled material are progressively written to report files. The report files are reformatted so that the data can be readily imported to another software such as a spreadsheet, a reserve database or detailed scheduling software for production analysis. This block by block information on the whole deposit may be then used for a variety of applications including productivity analysis and cost estimation. The various types of output reports from the simulation are described below.

Volumetric Report: This report file contains information regarding the volume, swing angle and hoist distance of each of the simulated cut and fill units on a section by section basis. This file is called REPORT.TXT and it is formatted so that it can be readily imported by spreadsheet software (Table 6.2). A summary report can also be created which includes a summary of the volumes input values and a definition of the terms used.

Table 6.2 - An example of part of REPORT.TXT output file.

```
*************** Dragline parameters ****************************************************
Dump height: 30.0;   Dump radius:   87.0;   Dig depth:                  45.0
Hwall clear   25.0   Rear clear     25.0    Crest clear                 6.0
Bucket width  6.0    Tub radius     9.0     Working gradient            5.0
****************************************************************************************

******************* Material parameters ************************************************
Repose angle 35.0   Coal trench angle     45.0   Spoil cut angle  45.0
Swell factor   1.2  Prime cut angle       75.0   Coal rib angle   75.0
****************************************************************************************

******************* Strip parameters ***************************************************
Strip width                  80.0    High wall angle                    75.0
Walk road width              40.0    Spoil bench width                  5.0
Maximum spoil flat top       10.0    Vertical distance to trench base   5.0
Max. overhand depth          15.0
2.0 % extra rehandle allowed for first pass clean up
****************************************************************************************
```

NOTE :- All volumes are in bcm

Sect No.	Str. No.	D/line U/H	D/line Chop	Truck Volume	Spoil Room	Spoil Req.	Incre. Spoil	Cumul. Spoil	Reh. Volume	Reh. (%)	Coal Loss
S1	6	77325	1480	124089	652	78804	78152	78152	92575	141.0	2923
S3	6	79267	1507	121894	50514	214520	30260	164006	38599	57.3	1607
S4	6	79200	1507	120207	72480	244713	8228	172234	35561	52.9	1607
S5	6	71874	808	101132	92402	244916	19719	152514	32591	53.8	1502
S6	6	70427	1313	93819	113826	224254	42087	110427	32951	55.1	1488
S7	6	83887	53043	57903	139473	247358	2543	107884	43278	37.9	1725
S8	6	85740	53857	55597	141764	247481	2167	105717	44184	38.0	1733
S9	6	79612	49997	48941	135033	235326	5424	100293	35508	32.9	1575
S10	6	79663	49817	44830	135660	229773	6180	94113	36048	33.4	1572

Dig Levels Report: This is a report on the elevations of all working levels including chop, main and spoil side bench levels. This file can be modified by the user and read back by the program as an input file for design of working levels.

Coal, Spoil and Rehandle Reports: These are output reports which are written specifically to be imported by a mine reserve database or a scheduling software. The files contain information on coal tonnage, spoil volume and rehandle percentage of each mining block. Examples of all types of output files are provided in Appendix D.

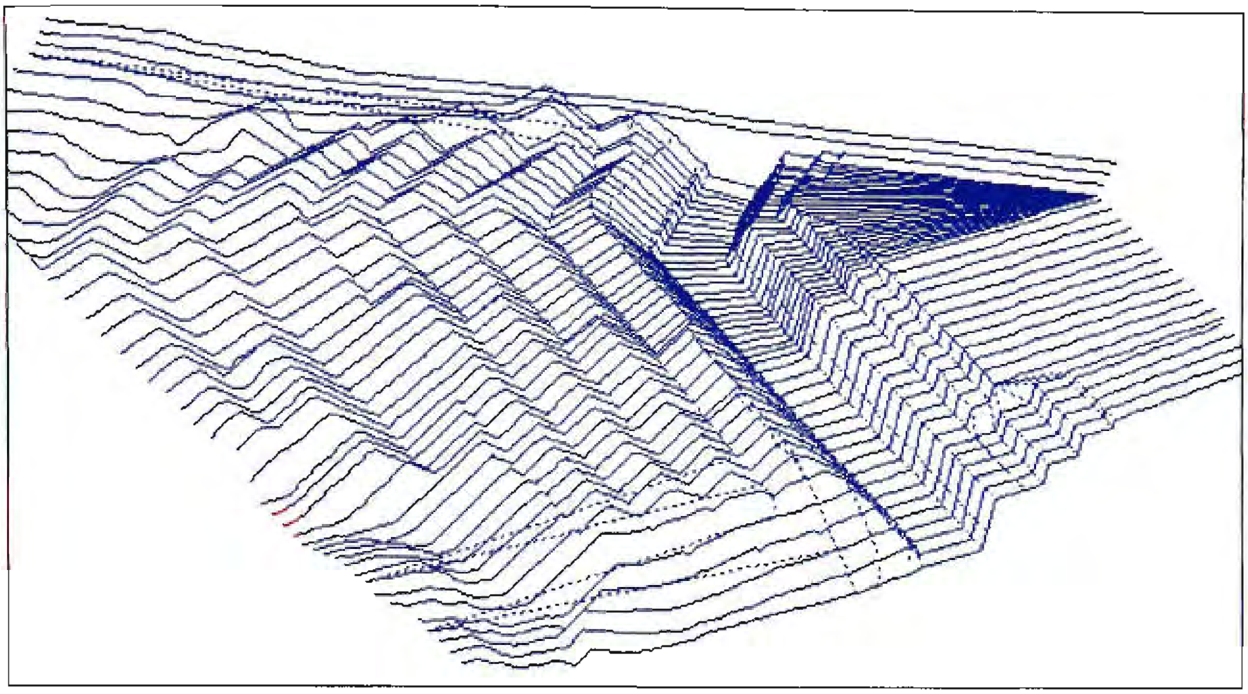


Figure 6.18- The 3D view of all the spoil strings generated in the simulated sections.

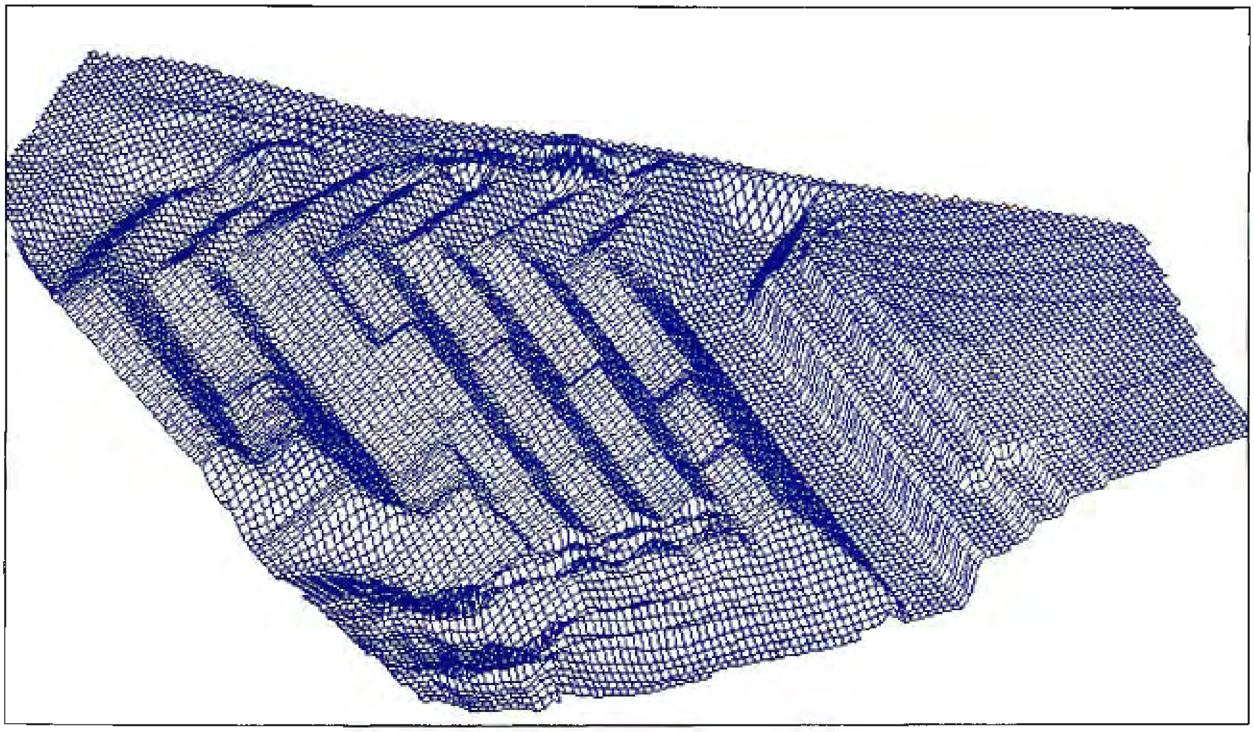


Figure 6.19- Output gridded surface of the simulated area, created from spoil strings.

The dragline simulator provides an optional output of the final spoil strings after the simulation of a pit. This process is set to be optional to save disk space and because the purpose of most of the program runs is to find optimum solutions while the user may only wish to see 3D outputs of the final design. It is also possible to view all the cross-sections in 3D while the dragline simulation is in process. Figure 6.20 shows the simulation of a strip for a set of parallel and radial sections.

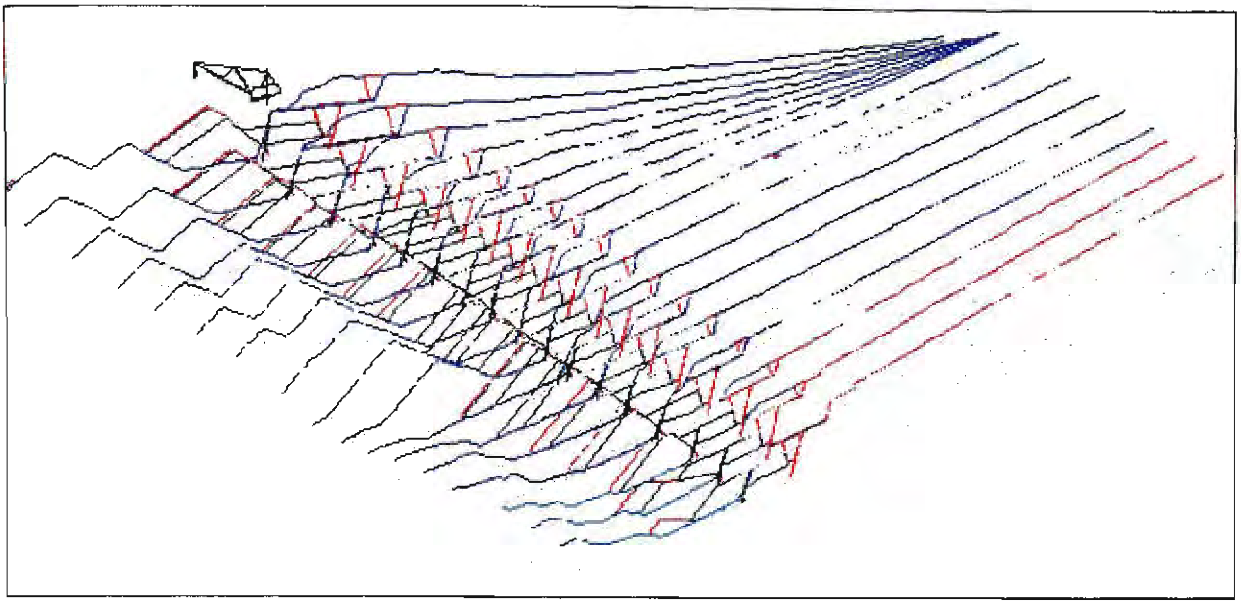


Figure 6.20- A 3D view of the dragline simulation for the entire sections.

6.4 SUMMARY

In this chapter the basic procedures used to simulate a dragline operation were discussed. These procedures serve as the core of the CADSIM system developed in this thesis and include the initial pit design, subdivision of the mining blocks, optimum dragline positions and calculation of cut and spoil profiles. The mathematics of the volume, swing angle and hoist distance calculations are also described and the related equations are provided. The general programming approach and procedures used to generate a logical sequence of the cut and spoil designs for development of a dragline mining scheme were also described in this chapter. This includes the extensive use of data gathered from the digging method survey discussed in Chapter 1.

The output files from the simulation contain valuable information which can be used for different strip mine planning purposes such as mine scheduling and the development of a reserve database. However, to allow a decision to be made based on this information, further analyses of such factors as productivity and cost of the operation are necessary.

CHAPTER SEVEN

DRAGLINE PERFORMANCE ANALYSIS

7.1 INTRODUCTION

Before a dragline productivity analysis can be performed, volumetric and swing angle information for the simulated mining blocks must first be combined with additional data from a dragline performance analysis and time study. Time study results provide the necessary results for most of these production parameters which cannot be estimated by the CADSIM dragline simulator. The time study results can also be used to determine the relationship between the elements of a dragline cycle time.

Dragline swing and hoist information and also walking and other delay times can be obtained either from performance curves provided by the equipment manufacturers or from mine site time studies. For the purpose of productivity calculations the cycle time components must be accurately estimated. Two major components, swing and return time, are governed by swing angle which is a function of stripping method and the geology of the deposit. Swing angle can be estimated from a simulation model based on the selected digging method and the geological model. Other cycle time components including, fill, dump and spot times are not governed by any factors which can be easily calculated or estimated from simulation of the dragline operation developed in this thesis. These parameters are assumed to be random variables with inherent statistical distributions and can be estimated by analysing a historical data.

Data captured by a Dragline Monitoring System (DMS) can be used for different purposes including machine performance analysis, scheduling, automated reporting and maintenance monitoring as well as evaluating the effect of geology and changes in the mode of operation. In this thesis a comprehensive time study was performed using data from a dragline monitoring system captured over a four month period. The results of the time study were then used as input in productivity calculations and also used for validation of the CADSIM model developed in this thesis.

7.2 DRAGLINE MONITORING SYSTEM

As the dragline operations extend to areas with deeper overburden and complex geological conditions during the life of the mine, varied stripping techniques are employed. In these situations it is important to have good control of the operating parameters and the machine performance. Dragline performance relies on many operating variables. A dragline monitoring system (DMS) is normally the best tool used to gather data on the dragline performance. Computer based dragline monitoring systems have been under development for about 25 years in Australia (Phillips, 1989). The basic objective of a DMS in any form involves the collecting, summarising, processing and reporting of detailed data on machine. The resulting performance analysis is useful in identifying and eliminating poor practice with the object of optimising critical mining parameters. This approach can be equally applied to the practice of blasting, stripping method and pit design. A DMS can also provide useful information on the evaluation and validation of a new stripping method.

There are five types of dragline monitoring systems which are in use or have recently been trialed in Australia. These units are the Tritronics 9000, ACIRL monitor, BHP Engineering monitor, HP Digmate and Westinghouse Lineboss (Phillips, 1989).

A DMS of any type consists of three major sections (McLean and Baldwin, 1989):

1. **On-board equipment:** this is a computer system used to log, process the raw data and generate digital outputs.
2. **Interface equipment:** this equipment provides a communication link between the dragline and the central computer in the mine office.
3. **Office computer:** this is a system to receive, compact and store data, perform additional calculations, interpret and manipulate data and present it in a large variety of tabular and graphical outputs.

A block diagram of the components of a typical monitoring system is shown in Figure 7.1. Figure 7.2 is a photograph of an on-board device which is part of a Tritronics 9000 dragline monitoring system installed on a BE 1570W dragline.

Please see print copy for image

Figure 7.1- A general block diagram of a Dragline Monitoring System (Phillips, 1989).

Figure 7.2- On-board equipment of a Tritronics 9000 dragline monitoring system.

7.2.1 Cycle Time Components

In practice a dragline cycle takes about a minute and at its normal operation a dragline makes 250,000 to 300,000 cycles each year. Any reduction in dragline cycle time can improve the overall profitability of a strip mine operation. For example, it is possible to reduce filling time and swing angles either through modifications in digging method or by improving the dragline operator's proficiency. The first step towards any improvement in a dragline operation is to have a clear idea of the different dragline actions during removal of a block of waste.

The information from a DMS is reported in the form of cycle time components and operational delays. The cycle time elements are strongly affected by the operating technique and geometry of the pit. A dragline cycle can be defined as a combination of fill, swing, dump, return and spot times, where swing and return times account for almost two thirds of the complete cycle time. The return times are not significantly shorter than the swing times as may be expected. The reason for this may be that part of the bucket positioning time is recorded as return time. The time recorded as spot time is the time between the return of the bucket to the three dimensional position of the previous cycle's bucket fill and the time when the bucket is engaged in the bank.

Compared with the other components of the cycle time such as spot, dump and fill time, the swing and return angle (or swing and return times) are more affected by the digging method employed and the geological conditions. The swing angle is primarily a function of the dragline position and the location of the cut and spoil area. These parameters are directly affected by the dragline operational mode (e.g. underhand, chop, etc.) and the digging method selected. For example, in a lowwall side dragline operation, the dragline sits on a lowwall side pad and pulls back the overburden to spoil it behind. This normally causes the swing angles to be longer compared with the normal underhand digging from the highwall side. The geological conditions such as the number and thickness of the coal seams and the thickness of overburden and interburden can also affect the swing angle.

The fill time is another important parameter in the dragline cycle time. A DMS records the fill time when a load appears on the drag ropes. There are a number of site specific factors which may affect the fill time, including the hardness of the overburden being dug and poor blasting which are believed to be the most important factors (Crosby, 1983). In a pit where part of the digging is chopping, the bucket fill time will increase. Deep digging tends to increase the fill time, as does shallow digging. Digging in the rehandle is generally easier resulting in lower fill time. Repassing also can affect the recorded fill times. A repass occurs when the bucket is not filled in one pass at the fairleads. It has been estimated that those cycles which require a repass have fill times approximately eight seconds longer than normal digging (ACARP, 1994).

Dragline monitors can provide detailed data for the key performance parameters which are essential in evaluation of the process. With today's sophisticated monitoring systems, the collection of data may no longer be a problem, however the question appears to be what data is required and how the data must be used. Mixing data from different sites with different geological conditions, machine specifications and different pit configurations does not provide valuable information and can be very misleading. This implies that the use of any monitoring data must be considered in relation to all geological and operational factors. It is more useful that the elements involved in digging a block are compared on this basis so that the specific cause of sub-optimum performance can be identified.

In 1994, Australian Dragline Performance Centre (ADPC) undertook a study to compare different dragline performance variables using raw data captured by dragline monitors. In excess of 2.6 million cycles, or the equivalent of approximately nine operating years of dragline data were processed to provide comparative performance indications for 16 draglines (ACARP, 1994). In that study all of the data from different sites and from various operating modes were analysed as one set to calculate average values of the selected parameters. The study showed that some draglines were less efficient than others, possibly due to valid reasons such as very deep overburden or rough topography. Although the study approached the problem from a global viewpoint, it emphasised that to determine the area of productivity loss a more detailed approach is required. This means that the process of dragline operation must be broken down into the individual component parts (ie. different operational modes and components such as key cut and chop cut) for analysing their effect and comparison studies.

7.3 ANALYSIS OF FIELD DATA

The data used in this thesis were captured by a Tritronics 9000 monitoring system and based on more than 100,000 cycles for two different dragline digging options. The data were then organised and processed to extract relevant statistics on different dragline activities such as fill, swing and hoist. The objectives of this part of thesis were to:

1. process and analyse actual data captured by the dragline monitor so that the critical performance parameters could be identified,
2. increase the understanding of the details of a dragline operation and the inter-relationship of the critical operational parameters,
3. provide sufficient input data for the development and calibration of the CADSIM system, particularly during the productivity estimation phase, and
4. validate the generated simulation results using the same geology and pit characteristics.

Another objective of this part of the thesis was to compare dragline performance parameters in different operating modes (ie. highwall and lowwall side). The data used for this part of the thesis was obtained from a mine that operated a three pass dragline operation. The first pass was a standard underhand technique, with a highwall key cut and a main dig component. The digging technique in the second pass was a low wall pass involving chop operations from an in-pit bench and in this pass the dragline was subject to tight spoiling and dumping to its maximum height. The requirement to dump behind the machine greatly increased the cycle time due to a longer swing angle. The third pass was essentially the same as the second pass. However, due to shorter swing angles, the cycle times are lower for the third pass. The data collected for the lowwall side consists of information from both the second and third passes.

To evaluate the interdependence of the variables which affect a dragline operation, it was important to outline the sequence of events in a dragline operation. As the first step, scheduling maps were reviewed to correlate the dragline locations with the time at which the data were recorded. This enabled the data to be separated into two sets on the basis of two distinct operational modes (ie. the highwall and lowwall side stripping). The next step was to develop routines in an EXCEL spreadsheet to convert the raw data to a manageable format.

7.3.1 Descriptive Statistics

The basic descriptive statistics (mean, standard deviation, etc.) and frequency histograms of different operational variables were generated. No comparison was made between the various components in digging a block within a specific pass (e.g. key cut and main cut in the highwall pass). This was mainly due to insufficient information in the recorded data and inconsistency in operators codes for different dragline operating modes. Table 7.1 and Figure 7.3 summarise the results of the comparison between the two operating modes in terms of average values and standard deviation. Table 7.1 also gives a comparison with the data from the ACIRL report representing average operating parameters for Australian dragline operations.

Table 7.1- Comparison of average and standard deviation of performance parameters.

Measured Parameter	Case Study (Highwall Side)		Case Study (Lowwall Side)		Data from ACIRL	
	Mean	St. Dev.[1]	Mean	St. Dev.	Mean	St. Dev.
Swing angle (deg)	73.2	31.9	120.1	45.2	92.7	9.1
Swing time (sec)	20.0	4.8	22.9	5.2	22.7	2.0
Return time (sec)	18.5	8.1	23.0	3.1	21.5	2.0
Filling depth (m)	13.4	4.7	19.0	5.6	N/A[2]	N/A
Filling time (sec)	14.6	6.7	19.6	10.2	18.3	2.0
Dumping height (m)	5.6	3.2	18.7	7.7	N/A	N/A
Dumping time (sec)	8.2	3.4	6.2	2.8	4.6	1.0
Cycle time (sec)	57.7	16.7	70.3	20.0	67.1	5.9
Fill repass (%)	3.4	0.6	4.5	1.5	5.5	1.8
Cycles per dig hour	52.2	8.3	43.7	6.3	43.6	4.2
Cycles per day	1066.0	167.1	902.2	182.1	819.0	119.4
Availability* (%)	73.5	10.4	72.8	13.8	77.7	9.0

$$* \ Availability = \frac{Operating \quad hours}{Scheduled \quad hours} \times 100$$

(1- St. Dev. = Standard Deviation)
(2- N/A = Not Available)

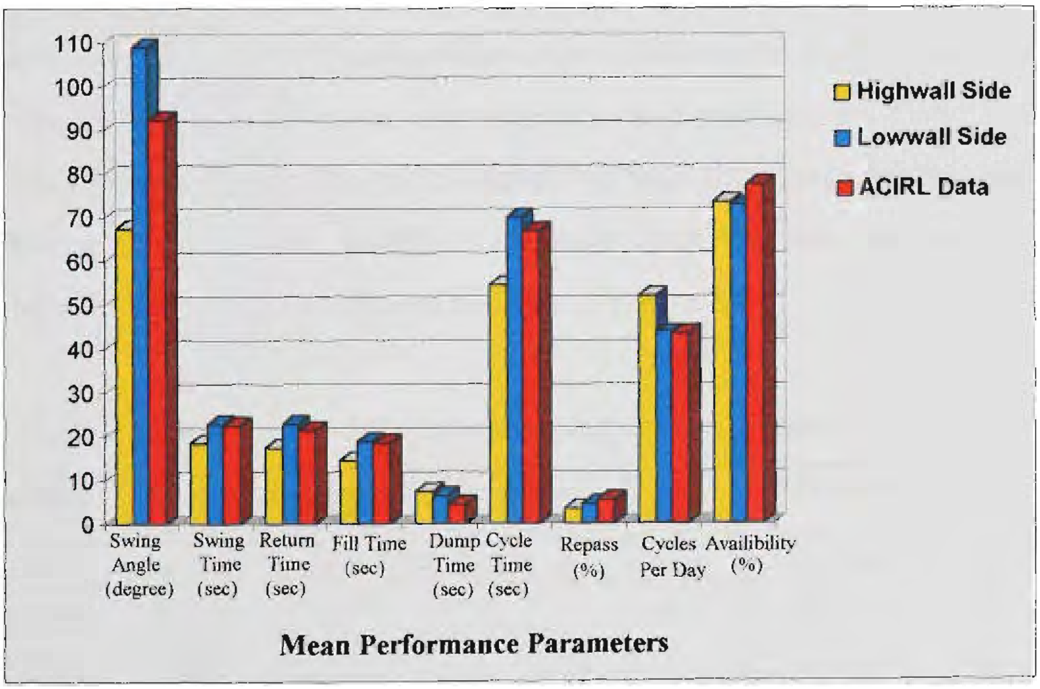


Figure 7.3- Comparison of the dragline mean performance parameters.

The reasons for the differences among the statistics of an operating variable in each data set can be explained by the changes in the digging method and the geological conditions. The average swing angle, fill time and the number of fill repasses are relatively higher in the lowwall side stripping since the dragline must fill and drag the bucket in a chopping mode.

7.3.2 Frequency Histograms and Best Fit Analysis

The Input Data Analysis module of ARENA software was used to generate frequency distributions of the performance parameters and also to perform a best fit analysis by fitting known distributions to the histograms (ARENA User's Guide, 1995). ARENA's Input Data Analysis module is a versatile tool that can be used to determine the probability distribution function that best fit a given set of input data. Once a data file has been selected, the Input Processor reads the file and determines the characteristics of the data file. After the data file has been loaded and displayed as a histogram, the next step was to fit a probability distribution function to the data using *Best Fit* option in ARENA's Input Data Analysis module. The distributions are then ranked, from best to worst, based upon the values of the respective squared errors. The quality of a curve fit is based primarily on a standard squared error criterion, which is defined as the sum of $[f_i - f(x_i)]^2$, summed over all histogram intervals. In this expression f_i refers to the relative frequency of the data for the i^{th} interval, and $f(x_i)$ refers to the relative frequency for the fitted probability distribution function (ARENA User's Guide, 1995). The detailed results of the best fit calculations from ARENA software, for both the highwall and lowwall stripping data sets, are presented in Appendix E. Tables 7.2 and 7.3 summarise the results and Figure 7.4 shows the histogram plots for the two data sets. The theoretical probability functions resulting from the best fit analysis are also superimposed over the histograms of the data in Figure 7.4.

Table 7.2- Statistics of the cycle time components for highwall side mining.

Variable	No. of Points	Min Value	Max Value	Mean Value	St. Dev*	Best Dist.**	Distribution Function	Square Error
Cycle Time	45823	10	120	57.7	16.7	Gamma	10 + GAMMA(6.38, 7.48)	0.003653
Dump Height	42560	1.0	24.9	5.57	3.2	Beta	1 + 24BETA(1.4, 5.97)	0.000782
Dump Time	45886	0.1	35.1	8.16	3.4	Beta	-0.5 + 40.5 BETA(7.39, 27.8)	0.003499
Filling Depth	45890	0.0	27.4	13.4	4.7	Beta	-0.001 + 28BETA(3.72, 4.02)	0.000807
Filling Time	45732	2.0	40.0	14.6	6.7	Erlang	1.5 + ERLA(3.27, 4)	0.000621
Hoist Distance	45934	0.0	38.5	15.8	5.4	Beta	-0.001 + 40BETA(4.75, 7.31)	0.001588
Return Time	45757	1.0	60.3	18.5	8.1	Beta	0.5 + 60BETA(4.72, 10.9)	0.002065
Swing Angle	45812	1.0	180.3	73.2	31.9	Normal	NORM(73.2, 31.9)	0.003248
Swing Time	45888	·3.0	68.2	20.0	4.8	Normal	NORM(20, 4.83)	0.005671
Tonnes / Cycle	45777	50.0	161.1	113.1	19.4	Normal	NORM(113, 19.4)	0.003361
Filling Factor	45777	0.14	1.40	0.968	0.192	Normal	NORM(0.968, 0.192)	0.003361

* *St. Dev. = Standard Deviation*
** *Dist. = Distribution*

Table 7.3- Statistics of the cycle time components for lowwall side mining.

Variable	No. of Points	Min Value	Max Value	Mean Value	St. Dev.*	Best Dist.**	Distribution Function	Square Error
Cycle Time	47738	10.2	140	70.3	20.1	Normal	NORM(70.3, 20)	0.00365
Dump Hgt. <15m	22578	1.0	14.8	6.84	3.0	Beta	1 + 14 BETA(1.77, 2.48)	0.00078
Dump Hgt. >15m	24549	15.1	57.6	35.9	7.0	Normal	NORM(35.9, 6.99)	0.00349
Dump Time	47701	1.0	25.3	6.2	2.8	Lognorm.	0.5 + LOGN(5.66, 2.4)	0.0008
Filling Depth	47213	5.1	32.6	19	5.6	Beta	5 + 28BETA(2.6, 2.58)	0.00062
Filling Time	21905	2.0	55.2	19.6	10.2	Beta	1.5 + 53.5BETA(1.91, 3.67)	0.00158
Hoist <40m	23835	0.2	40.0	22.9	8.3	Normal	NORM(22.9, 8.31)	0.00206
Hoist >40m	24021	40.0	81.3	57.7	7.2	Normal	NORM(57.7, 7.17)	0.00324
Return Time	47707	0.0	55.1	23.7	9.8	Normal	NORM(23.7, 9.75)	0.00567
Swing Angle	47470	15.0	247	120	45.0	Beta	15 + 235 BETA(2.56, 3.17)	0.00336
Swing Time	47775	3.0	40.0	22.9	5.2	Beta	2.5 + 37.5BETA(8.24, 6.98)	0.00381
Tonnes /Cycle	47666	40.0	160	107	17.9	Normal	NORM(107, 17.9)	0.00105
Filling Factor	47855	0.0	1.39	0.913	0.193	Normal	NORM(0.913, 0.193)	0.0059

** St. Dev. = Standard Deviation*
*** Dist. = Distribution*

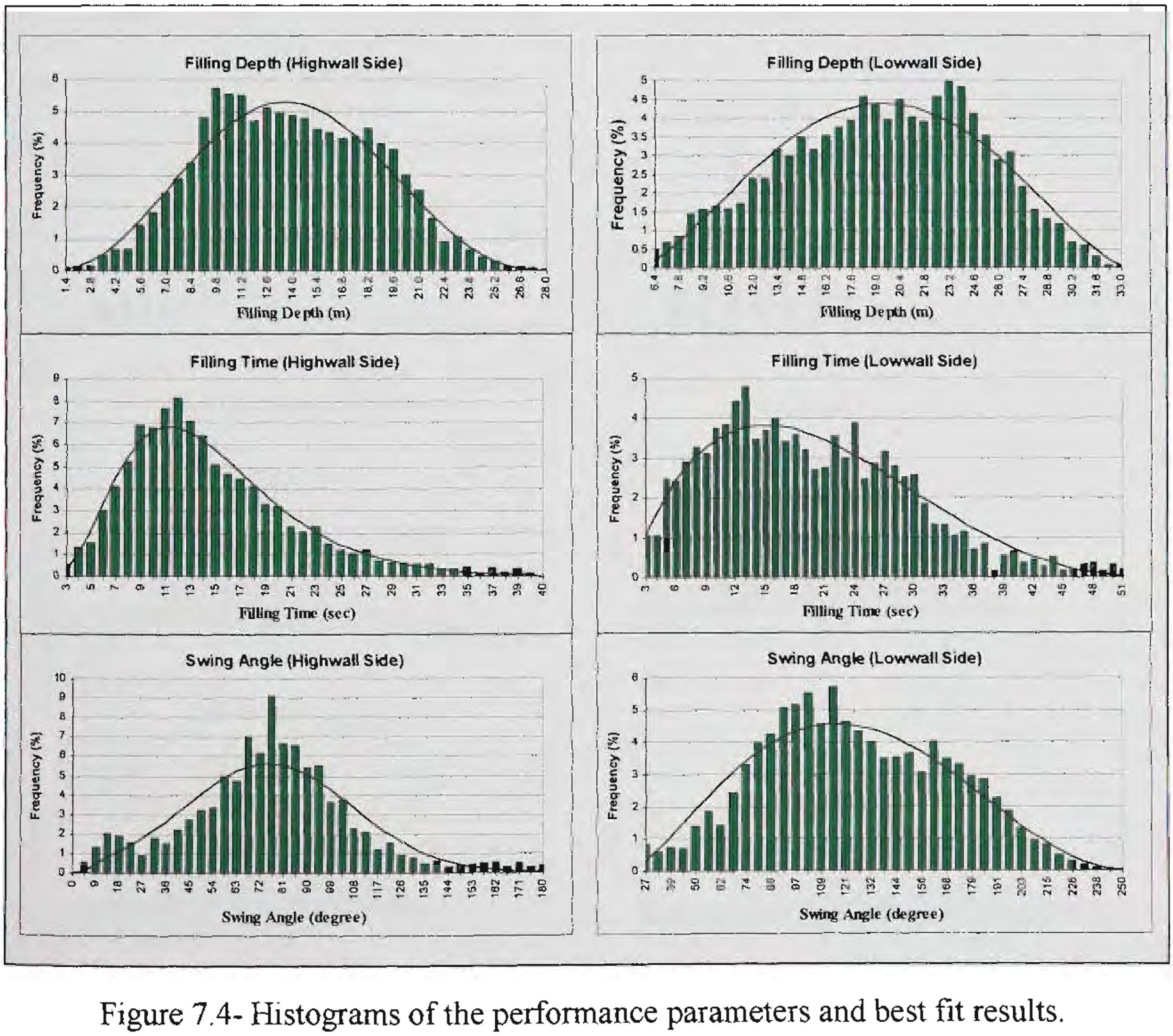


Figure 7.4- Histograms of the performance parameters and best fit results.

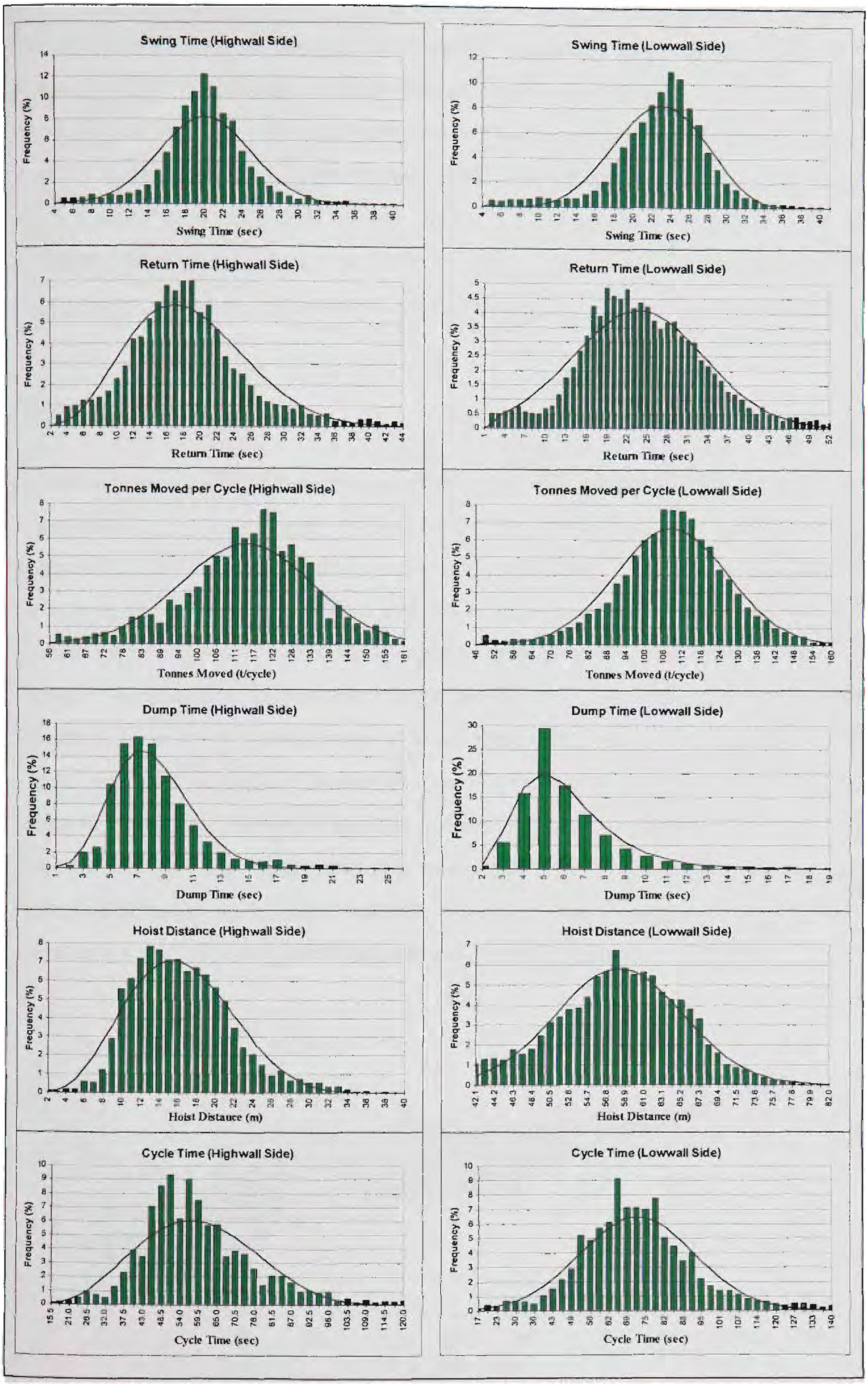


Figure 7.4- Histograms of the performance parameters and best fit results (Continued).

The results of the best fit calculations should be interpreted as guidelines rather than precise scientific calculations. This is because the relative ranking can be affected by the number of intervals within the histogram or choice of histogram end points. Thus, if two or more distribution functions show small square errors that are relatively close to each other, it is not clear that the function with the smallest square error is necessarily the best. However, the results of the best fit calculations do allow one to distinguish clearly between those functions that fit the data well and those that do not.

7.3.3 Correlation

Correlation can be defined as a measure of the relationship between variables. Usually a regression analysis is used to investigate the relationship between predictor (independent) variables and a criterion (dependent) variable. The regression analysis fits a trend line for the available data and results in an equation being derived that can be used for prediction of a dependent variable when only the independent variable is known. Two indicators *Correlation Coefficient* (R) and *Coefficient of Determination* (R^2) are used to quantify the degree of linear relationship between the variables in a simple regression analysis. Statistically, the Correlation Coefficient expresses the degree to which an independent variable is linearly related to the dependent variable, while the Coefficient of Determination is an indicator of how a dependent variable can be explained with an independent variable. Figures 7.5 and 7.6 are scatter plots of the swing time versus swing angle for highwall and lowwall stripping respectively.

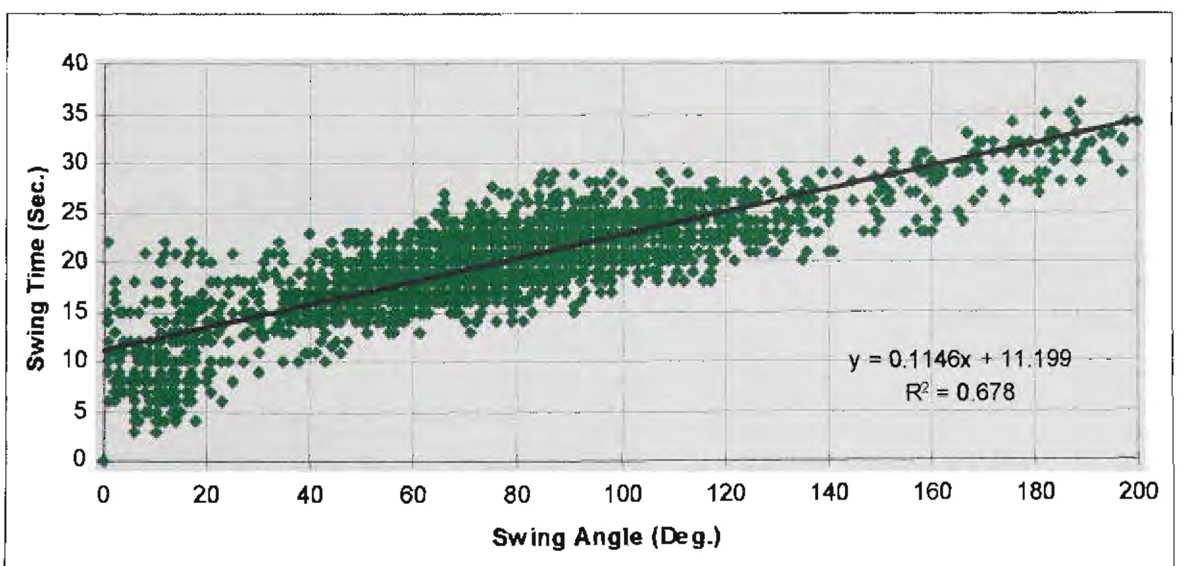


Figure 7.5- Scatter plot of swing time vs swing angle for the entire data set on highwall side.

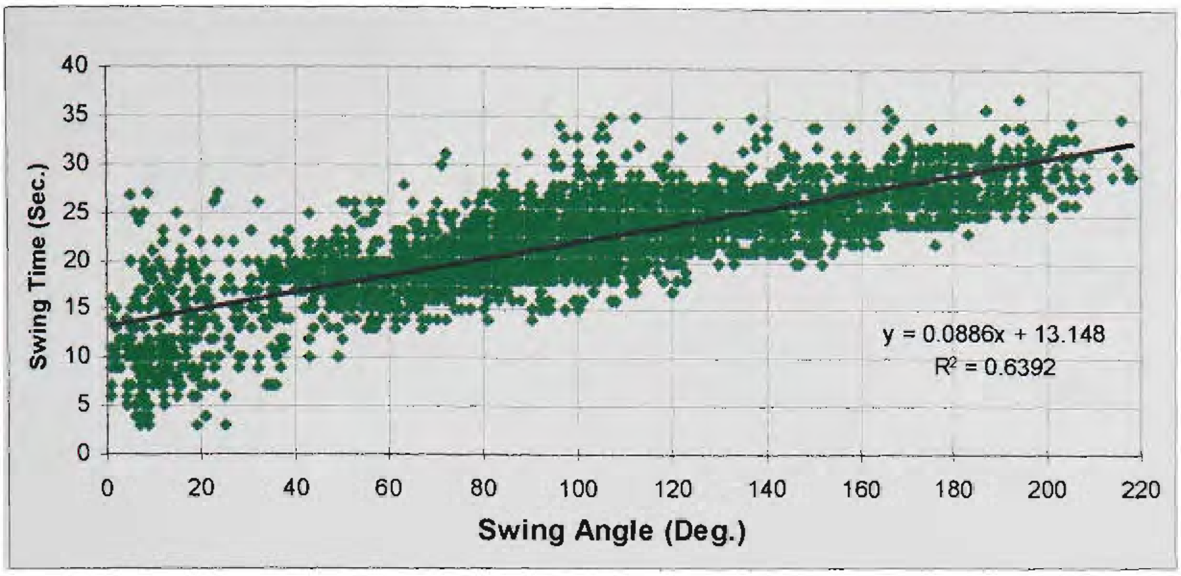


Figure 7.6- Scatter plot of swing time vs swing angle for the entire data set on lowwall side.

A preliminary regression analysis conducted on the two complete data sets (lowwall and highwall) led to the conclusion that only partial correlation existed between swing angle and swing time for whole data sets. The correlation factor for highwall stripping was $R^2 = 0.68$ ($R = 0.82$) and for lowwall stripping was $R^2 = 0.64$ ($R = 0.8$). In other words only 65 percent of swing times can be explained by a known swing angle for both stripping cases. From Figures 3 and 4 it can be seen that a poor correlation existed for swings of less than 40 degrees. The entire data set was separated into two groups (swings less and greater than 40 degrees) and the regression analysis was repeated for each group. Figures 7.7 through 7.10 are scatter plots of the two new data sets after division of the data for both stripping cases.

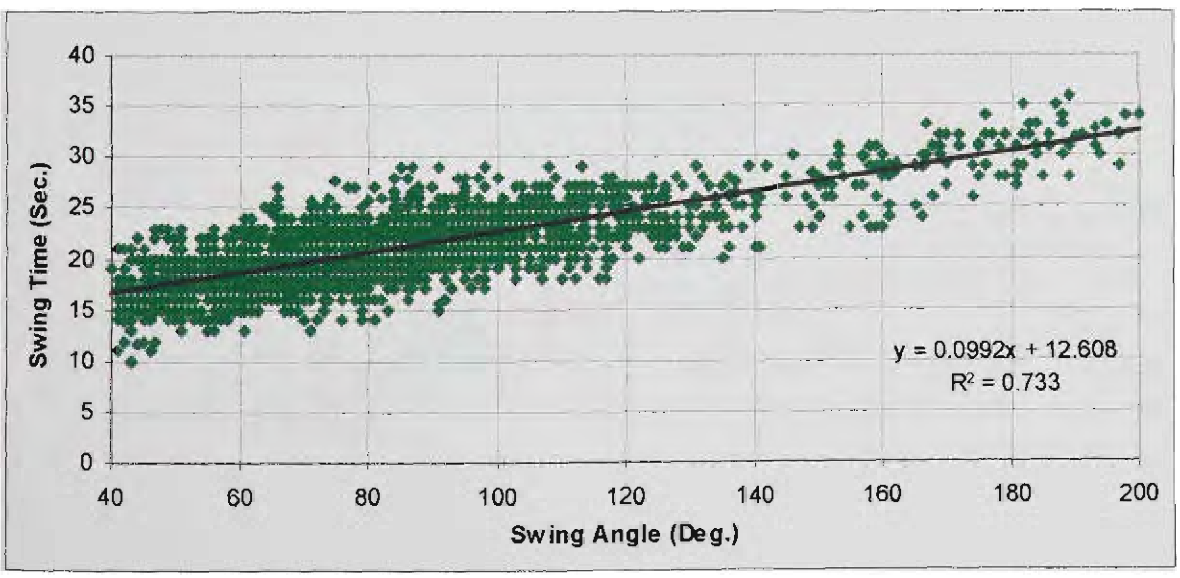


Figure 7.7- Scatter plot of swing time vs swing angle for swings > 40° on highwall side.

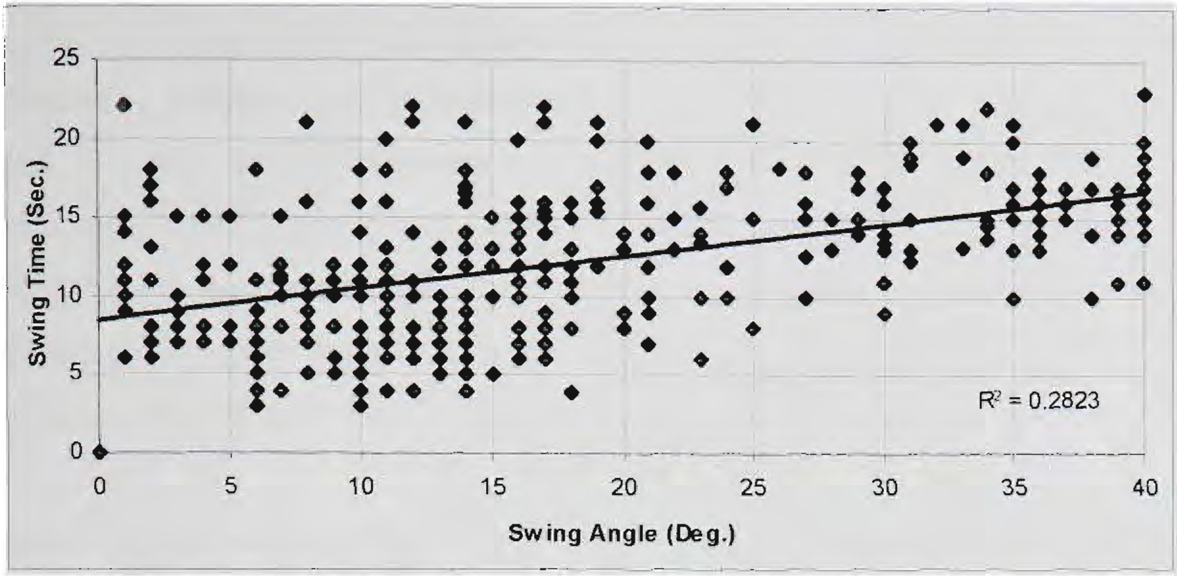


Figure 7.8- Scatter plot of swing time vs swing angle for swings < 40° on highwall side.

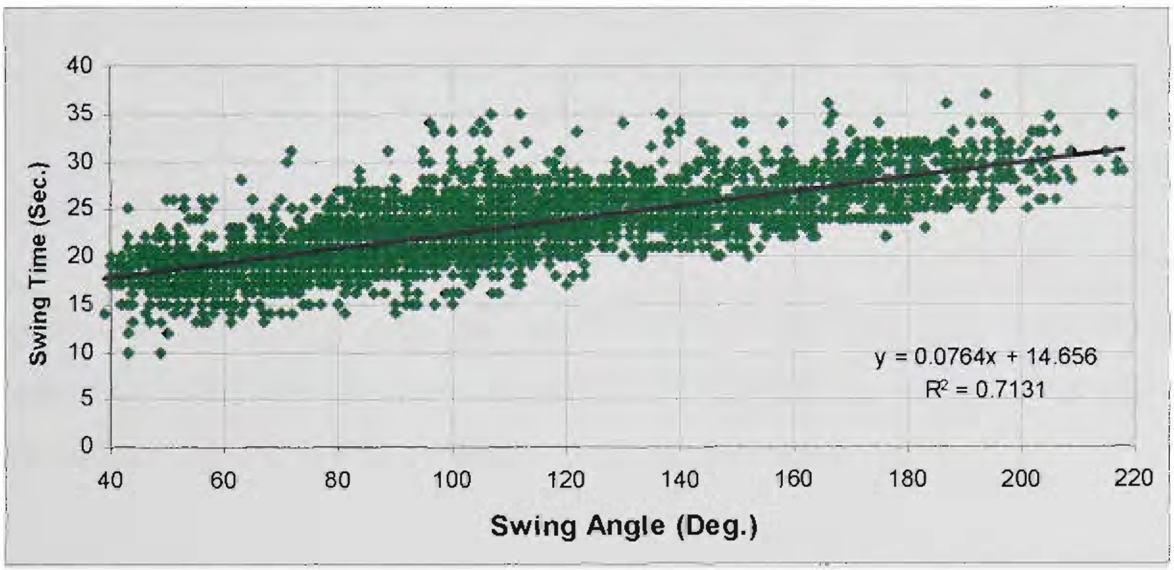


Figure 7.9- Scatter plot of swing time vs swing angle for swings > 40° on lowwall side.

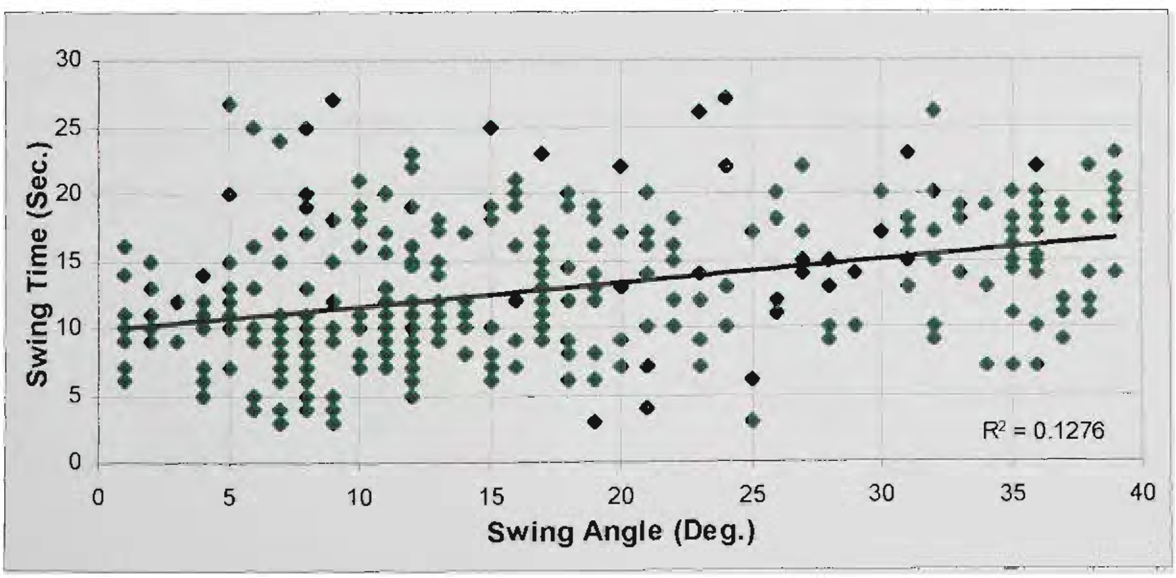


Figure 7.10- Scatter plot of swing time vs swing angle for swings < 40° on lowwall side.

It may be expected that short swings are hoist dependent and that the swing time is affected by the time required for hoist and drag payout rather than by the actual swing time. The hoist dependent swings generally occur where the dragline is operating in deep digging and high spoiling mode. The separation of two sets of swing angles improved the correlation coefficient for swing angles greater than 40 degrees for both stripping cases. The correlation coefficients increased to R^2 = 0.73 (R = 0.85) for highwall stripping and R^2 = 0.71 (R = 0.84) for lowwall stripping.

The following linear equations were developed for the two stripping cases when swing angles are greater than 40 degrees:

$$Y = 0.099X + 12.6 \qquad \textit{(for Highwall stripping)} \qquad (7.2a)$$

$$Y = 0.078X + 14.7 \qquad \textit{(for Lowwall stripping)} \qquad (7.2b)$$

where: Y = swing time in seconds, and X = swing angle in degrees.

Since most swings are within the range of 40 - 120 degrees, the calculated linear equations can be used to convert swing angles to swing times for use in productivity calculations with reasonable accuracy.

The regression analyses were also conducted to evaluate any correlation which may be apparent between geological conditions and fill and dump times. Initially it was felt that filling time and dump time would be correlated to the dig depth and dump height respectively. But examination of the results showed that there was almost no correlation between the depth of digging and fill time and also for dump height and dump time. The results for the two stripping methods are plotted in Figures 7.11 through 7.14.

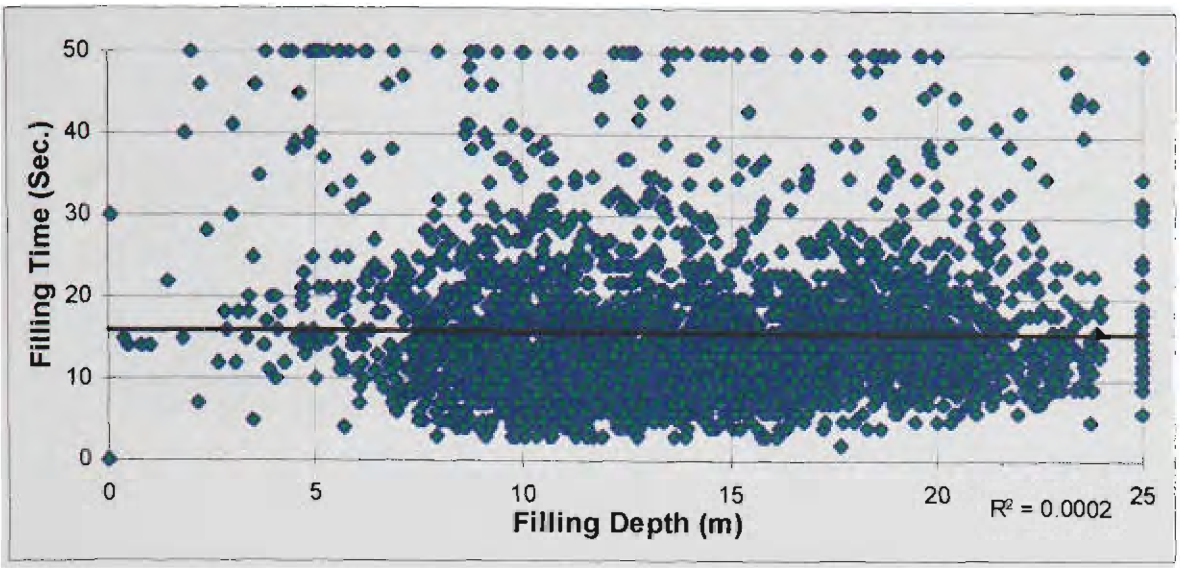


Figure 7.11- Scatter plot of filling time vs filling depth for the highwall side.

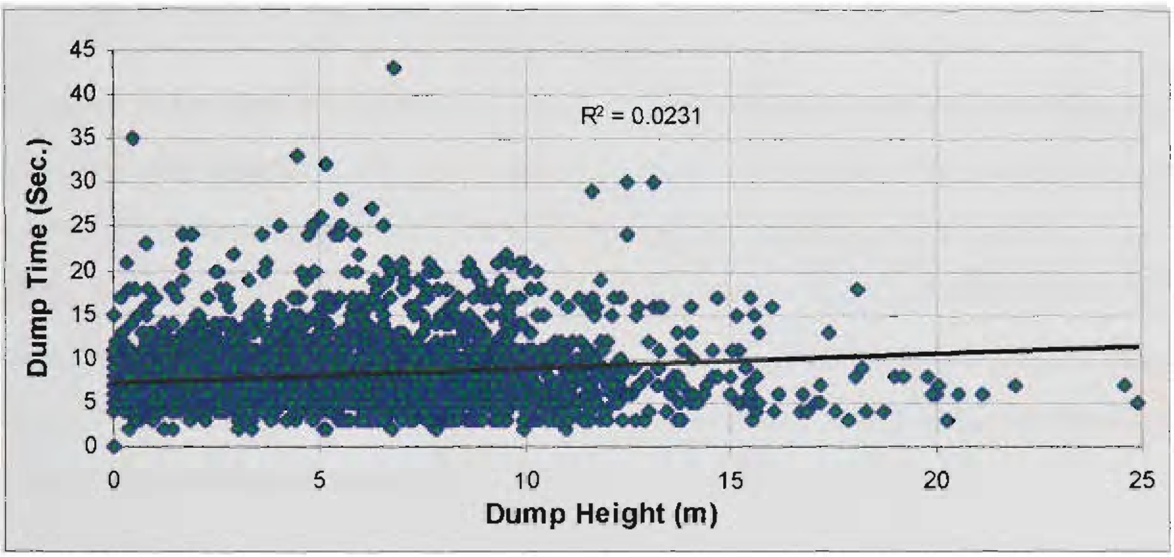


Figure 7.12- Scatter plot of dump time vs dump height for the highwall side.

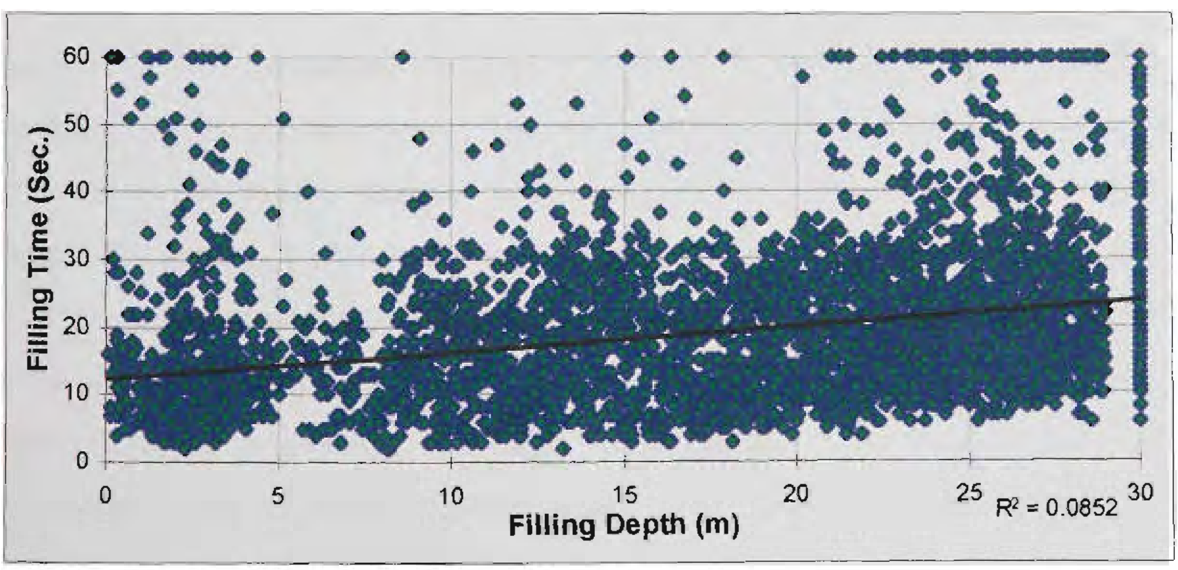


Figure 7.13- Scatter plot of filling time vs filling depth for the lowwall side.

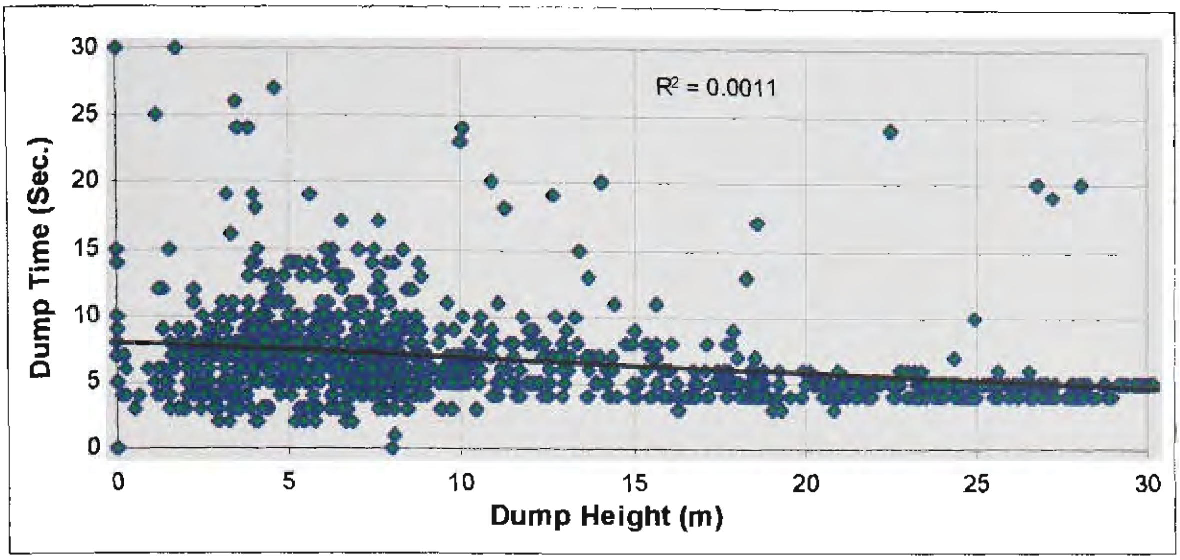


Figure 7.14- Scatter plot of dump time vs dump height for the lowwall side.

7.4 SUMMARY

The results of the time study data captured by a dragline monitoring system (DMS) are discussed in this chapter. The use of data from a DMS has many potential advantages but any evaluation using this information must include factors such as geology, digging method, blasting technique and dragline specifications. Statistical summaries of the data in terms of average values, standard deviation and frequency distributions showed that the mode of operation and the geological conditions have a significant influence on the dragline performance parameters.

A best fit analysis was also carried out using the data from the DMS to provide frequency distributions for the relevant dragline performance parameters. This information is required for stochastic analysis of the dragline's productivity to be discussed in the subsequent chapters. Regression analyses were performed to investigate any associations which may exist between the various parameters. The correlation study showed that the only demonstrable relationship which could be determined existed between swing angle and swing time. Filling time and dump time did not show any evidence of a correlation with geological and operational parameters such as digging depth or dump height, based on data which can be estimated by the dragline simulator developed in this thesis. When calculating the dragline productivity, the average value of these parameters are used either as fixed variables in a deterministic approach or as random variables with statistical distribution in a stochastic approach.

CHAPTER EIGHT

DRAGLINE PRODUCTIVITY AND COST ANALYSIS

8.1 INTRODUCTION

There are two possible methods for determining the optimum digging conditions for a dragline operation. These are analysis of mine productivity and the cost of overburden removal for a given geology and dragline. Dragline productivity can be used as a satisfactory criterion for selecting possible digging alternatives, however this applies only when there is little difference between the cost of the dragline and the associated operations such as drilling and blasting. In such cases preference is given to the option with a higher annual prime productivity. However, where the alternative stripping options involve different drilling patterns and blasting requirements, mine productivity can no longer be used as a basis for selecting an "optimum" digging method. Instead the most cost effective option is selected. Discounted cash flow (DCF) analysis has been used to calculate the break even cost for a mining operation. In this thesis a modification of this standard technique called a "discounted average cost" is applied to estimation of the major cost components associated with a dragline operation. This technique was selected because it provides more flexibility when it is intended to exclude some common cost items.

8.2 PRODUCTIVITY ANALYSIS

The cyclic nature of dragline operations means that after the movement of a single block the procedure is repeated. However, as the mining advances, the pit design and stripping method may be changed as a result of changes in geology. For example, a simple side casting method in the shallow area may change to an extended bench with a chop operation as the overburden depth increases. These changes in the pit design and the dragline mode of operation affect the productivity of the dragline due to variations in volume, swing angle and hoist distance values. In the following sections a step by step procedure is presented to show how the dragline productivity is estimated in this thesis.

8.2.1 Definition of Various Productivity Terms

The productivity of a walking dragline can be defined as the volume of prime overburden removed by the dragline per unit time. In many circumstances, particularly when throw blasting is involved, the mine productivity is completely different from the dragline productivity as a considerable proportion of overburden is moved into the final spoil by blasting or auxiliary equipment, such as dozers. The productivity terms used in this thesis are defined as follows:

Dragline prime productivity: The dragline prime is the volume removed only by the dragline and excludes rehandle and the volume removed by dozers and blasting. The dragline prime volume is then divided by the total excavation time to give the dragline prime productivity.

Dragline total productivity: The dragline total volume is the volume removed only by the dragline including rehandle, but excluding dozer and blasting. The dragline total volume is then divided by the total excavation time to give the dragline total productivity.

Mine prime productivity: The mine prime volume is the volume removed by either dragline, blasting or dozer, excluding rehandle. The mine prime volume is then divided by the total excavation time to give the mine prime productivity.

Mine total productivity calculation: The total volume includes rehandle and is the volume removed either by dragline, blasting or dozer. The total volume is then divided by the total excavation time to give the total productivity.

The above productivity terms are defined as below:

$$\textit{mine prime productivity} = \textit{prime volume of overburden/dig hours} \tag{8.1a}$$

$$\textit{mine total productivity} = \textit{mine prime productivity} \times (1 + Reh) \tag{8.1b}$$

$$\textit{dragline prime productivity} = \textit{mine prime productivity} \times (1 - Bls) \tag{8.1c}$$

$$\textit{dragline total productivity} = \textit{mine prime productivity} \times (1 - Bls + Reh) \tag{8.1d}$$

where: *Reh* = rehandle percentage, and

Bls = thrown percentage by blasting.

In these productivity calculations, all volumes are calculated as bank insitu (non-swelled) volumes. All the productivity terms are expressed in bcm/hr.

8.2.2 Prime and Total Productivity Calculation

Dragline productivity is a simple idea that is often complex to measure (Kahle, 1990). In this thesis, productivity was estimated by calculating either detailed block by block values or strip by strip values. In both approaches the dragline productivity is based upon the calculation of two primary factors:

1. The actual number of productive cycles that occur in a given time (e.g. one hour), and
2. The actual bank (prime) volume in m^3 moved in each cycle, normally called "bucket factor".

Contributing to the number of cycles are delay times and cycle time. The volume per bucket is influenced by parameters such as bucket size, swell factor, fill factor and operator skill.

The block by block method provides a more detailed type of calculation and it is more suitable for providing information for short term scheduling purposes. In this approach sections are created with an interval equal to a dragline block length (usually 30m). Relative production rates of the dragline in each operational mode such as key-cutting, chopping, rehandling and normal underhand digging are analysed by the CADSIM model. This provides basic information such as volumes, swing angles and walking requirements for productivity estimation in each of the overburden blocks being removed.

In evaluating the use of some digging methods such as multi-pass operations and for comparison purposes, non-detailed strip by strip calculations may be used. In a strip by strip calculation a block is first divided into sub-blocks and the total volume of a sub-block (eg. key cut) along a strip is used as one unit. The next step is to calculate an average value for the time required to remove that particular unit or part of the strip. When including the effect of rehandling around coal access ramps on productivity, the strip by strip approach is commonly used.

8.2.2.1 Calculation of Number of Cycles

A productive cycle refers to a cycle in which the overburden is carried in the bucket and dumped into the spoil area. The number of swings per production (dig) hour is a function of the cycle time. This may be expressed as:

$$\textit{Number of cycles per production hour} = \frac{3600}{\textit{Cycle time (sec)}} \qquad (8.2)$$

In most operations, a cycle takes approximately one minute. From Equation 8.2 it can be seen that a small reduction in cycle time (eg. a few seconds) will improve the total number of cycles and increase the productivity quite significantly. The dragline cycle time is governed by the geology, digging method, bucket geometry and operator skills. A complete cycle time can be broken into its components. The cycle components are drag to fill, swing to dump, dump, swing back, and bucket positioning.

Cycle time: Dragline cycle time as used in this thesis consists of fixed and variable time intervals. The fixed time elements are dig, positioning and dump time which are assumed to remain constant in every cycle. The operational times which vary from cycle to cycle are swing and return times. These elements represent the major components of the total cycle time (up to 70%). Both swing and return times are correlated with the swing angle. When the real data from a time study are not available (eg. for a new dragline), the swing angle versus time graphs supplied by the manufacturer may used. Equations 8.3a and 8.3b have been derived using the swing angles versus swing time graph for a P&H 9020 walking dragline.

$$T = 1.35\, X^{0.56} \qquad \text{for } X \leq 30 \qquad (8.3a)$$

$$T = 6.46 + 0.1177X \qquad \text{for } X > 30 \qquad (8.3b)$$

where: X = angle of swing in degrees, and

T = swing time in seconds.

The total cycle time which is the sum of the fixed times (filling, spot and dump times) and variable times (swing and return times) must be calculated for all components of a block such as key cut, main cut and extended bench.

8.2.2.2 Calculation of Bucket Factor

The volume moved in every cycle is affected by the three parameters, bucket capacity, swell factor, and bucket fill factor.

Swell factor: Material once excavated becomes loose and its original volume is increased. The swell factor is defined as the ratio of volumes of equal weight of material after and before excavation. The swell factor may vary from 1.1 to 1.6 for most overburdens depending on the material characteristics, fragmentation and water content (Humphrey, 1990).

Bucket fill factor: This is the percentage of the nominal bucket capacity that actually fills with material. The bucket fill factor is expressed as actual loose volume per loose volume of bucket rated. Average fill factors normally vary from 0.8 to 1.2. In practice, the bucket fill factor is slightly higher for an underhand digging than a chopping operation.

The bucket factor is defined as the equivalent prime volume of material in the dragline bucket and this is obtained by adjusting the bucket volume for swell factor and bucket fill factor.

$$Bucket\ Factor = \frac{Bucket\quad capacity(loose\quad cubic\quad metre) \times Fill\quad factor}{Swell\quad factor}$$

For a dragline equipped with a 47 cubic metre bucket and 95% bucket fill factor and 1.3 swell factor, the bucket factor is 34.35 bcm.

8.2.3 Block by Block Productivity Calculation

The productivity varies between blocks due to differences in swing angle, digging mode (underhand or chop), ease of digging (rehandle or prime), ease of spoiling and depth of cut. The prime productivity for a particular block is simply the ratio of total block volume to the total time calculated for removal of the volume. The prime productivity of the dragline can be explained by Equation 8.4.

$$Productivity\ over\ a\ block = \frac{Prime\ volume\ of\ block\ being\ excavated}{Total\ required\ time\ for\ removal\ of\ the\ block} \qquad (8.4)$$

where: $Total\ required\ time = \sum (excavation\ time + walking\ time)$

The elements of the Equation 8.4 are:

Total required time: This is the sum of all of the times taken to excavate various parts of the block (e.g. key cut, main cut, etc.) plus the walking time within the block.

Excavation time: The number of swings multiplied by the total cycle time gives the excavation time for each part of the block.

Number of swings: The number of swings required to excavate each part of the block is the volume of that part divided by the bucket factor.

Walking/Manoeuvring time: The total distance between the dragline positions when excavating the various parts of a block divided by the average walking speed gives the walking time.

The productivity of each mining block can be calculated separately. Since the depth of the dragline overburden and the length of the block (in the case of radial sections) may change for each block, the productivity over several blocks (ie. one strip) is determined as follows.

$$P_s = \frac{Total \quad volume \quad of \quad strip}{Total \quad time} = \frac{V_1 + V_2 + ...+ V_n}{T_1 + T_2 + ...+ T_n} \qquad (8.5a)$$

where: P_s = overall productivity of a strip s,

V_i = volume of the i^{th} block,

T_i = time required to remove the i^{th} block, and

n = number of blocks.

The volume of each block can be defined as:

$$V_i = D_i \times W_i \times L_i \qquad (8.5b)$$

Time spent to remove a block is defined as:

$$T_i = \frac{V_i}{P_i} \qquad (8.5c)$$

where: D_i = depth of block i,

W_i = width of block i,

L_i = the length of i^{th} block,

V_i = the i^{th} block volume, and

P_i = the productivity of the dragline for i^{th} block.

From the above, the strip productivity can be calculated as follows.

$$P_s = \frac{\sum_{i=1}^{n} D_i \times W_i \times L_i}{\sum_{i=1}^{n} T_i} = \frac{\sum_{i=1}^{n} D_i \times W_i \times L_i}{\sum_{i=1}^{n} \frac{V_i}{P_i}} = \frac{\sum_{i=1}^{n} D_i \times W_i \times L_i}{\sum_{i=1}^{n} \frac{D_i \times W_i \times L_i}{P_i}} \tag{8.6a}$$

As the strip width, W_i, is usually constant along the strip, Equation 8.6a reduces to:

$$P_s = \frac{W \sum_{i=1}^{n} D_i \times L_i}{W \sum_{i=1}^{n} \frac{D_i \times L_i}{P_i}} = \frac{\sum_{i=1}^{n} D_i \times L_i}{\sum_{i=1}^{n} \frac{D_i \times L_i}{P_i}} \tag{8.6b}$$

If no radial sections are involved the block length, L_i is constant and the Equation 8.6b also reduces to:

$$P_s = \frac{\sum_{i=1}^{n} D_i \times L_i}{\sum_{i=1}^{n} \frac{D_i \times L_i}{P_i}} = \frac{L \sum_{i=1}^{n} D_i}{L \sum_{i=1}^{n} \frac{D_i}{P_i}} = \frac{\sum_{i=1}^{n} D_i}{\sum_{i=1}^{n} \frac{D_i}{P_i}} \tag{8.6c}$$

8.2.4 Strip by Strip Productivity Calculation

In a strip by strip method of productivity calculation, the mining blocks are first divided into sub-volumes or digging components such as key cut, main cut and extended bench. It is assumed that the number of cycles per hour and the bucket factor remain constant for an individual digging component and their product defines the dragline productivity for a sub-volume of the block. The overall productivity of a strip is then a weighted average of the dragline productivity in each sub-volume based on the proportion of time spent for the removal of that sub-volume. A strip by strip productivity calculation uses the following equation:

$$P_S = \sum_{i=1}^{n} (P_i \times t_i) \tag{8.7a}$$

where: P_S = overall productivity of the strip s,

P_i = productivity of the i[th] sub-volume, and

t_i = time coefficient of the i[th] sub-volume.

Two parameters P_i and t_i are calculated as follows:

$$P_i = Number\ of\ cycles\ per\ hour \times bucket\ factor \tag{8.7b}$$

Since the "*Number of cycles per hour*" is based on the cycle time estimated for each sub-volume,

$$t_i = \frac{V_i \times C_i}{\sum_{i=1}^{n}(V_i \times C_i)} \tag{8.7c}$$

where: n = number of sub-volumes in a block (eg. overhand chop, main pass key cut, etc.) ,

V_i = the total volume of the i[th] sub-volume over a strip, and

C_i = cycle time estimated for the i[th] sub-volume.

8.2.5 Annual Productivity Calculation

Annual productivity can be estimated by considering the annual digging hours and the average prime productivity for a certain area. For existing dragline operations, dig hours can be obtained by analysing historical data, but for a new operation the dig hours can be calculated based on industrial data and on data supplied by the manufacturers. For a detailed explanation of the assessment or estimation of the annual productivity of the dragline, clear definitions of some of the ambiguous variables must first be established.

Calendar Hours: This is the actual total hours in a given period of observation or prediction, for example for a year which is 365 days. Excluded from this are the public holidays such as Christmas, Easter and Australia Day.

Scheduled Hours: This is the time during which the machine is expected to operate. Large walking draglines are typically scheduled to operate all year long and seldom less than 8000 hours per year (Humphrey, 1984), except for any period of delay directly due to mechanical or electrical problems. This includes worn part replacement and other major repair work. Usually, a large dragline needs one shift shutdown every fortnight for scheduled maintenance and four weeks every four years for major re-build of the equipment.

Available Hours: This is the part of the scheduled hours that the machine is mechanically and electrically available for work. Availability varies with the age of the equipment, the difficulty of the working conditions, the efficiency of the preventive maintenance program and so on. Large walking draglines are generally reliable pieces of equipment and a figure of 85% of the scheduled hours is normally used in calculating available hours.

Operating Hours: This is that part of the available hours during which the dragline is actually operating. A delay due to an operational problem represents available time lost and this causes the machine to be inoperative during such periods as strike time, vacation time, power source outage, adverse weather, meal breaks, and other periods when the machine or crews are not working. Locality factors must also be considered in calculating the operating hours. In New South Wales, 90% of available hours is a common figure used to calculate operating hours for a large walking dragline.

Production (Dig) Hours: A machine operating hour is that time when the motor is running, though the machine may not be doing productive work. A production (also called dig) hour is that time when the equipment is in operation in a productive capacity. Non-productive times include waiting on support equipment such as dozer work, cable handling, blasting or performing a non-productive function such as long walks. Some mining operations subtract eight minutes (13%) per operating hour to allow for actual dig hours and include any other unaccounted delays.

From the above definitions the following relationships can be derived:

$$\textit{Scheduled Hours} = \textit{Calendar Hours} - \textit{Public Holidays}$$

$$\textit{Available Hours} = \textit{Scheduled Hours} - \textit{Repair Hours}$$

$$\textit{Operating Hours} = \textit{Available Hours} - \textit{Standby Hours}$$

$$\textit{Production Hours} = \textit{Operating Hours} - \textit{Delay Hours}$$

$$\textit{Mechanical Availability}\ (\%) = \frac{\textit{Available Hours}}{\textit{Scheduled Hours}} \times 100$$

$$\textit{Operational Availability}\ (\%) = \frac{\textit{Operating Hours}}{\textit{Available Hours}} \times 100$$

$$\textit{Utilisation}\ (\%) = \frac{\textit{Production Hours}}{\textit{Operating Hours}} \times 100$$

The availability percentage (either mechanical or operational) serves as an indication of the efficiency of the maintenance program. Availability varies with the competency of the mine personnel, the efficiency of the mine plan and support equipment commitments. The utilisation percentage is used in performance predictions and is also an indication of machine management and work efficiency of the equipment. Table 8.1 shows a breakdown of the calculated calendar time for the mine used in the first case study in a normal year (ie. no major maintenance).

Table 8.1- Estimation of annual dig hours for a walking dragline.

Description		Days	Remained days	Shifts	Hours	Remained hours
CALENDAR		**365**		**1095**	**8760**	
Less						
Public holidays		2	363	6	48	8712
SCHEDULED DAYS/SHIFTS/HOURS		**363**		**1089**		**8712**
Less						
Scheduled maintenance - one 10 hrs shift/week		21.7	341.3	65	520	8192
Unscheduled maintenance (electrical, etc.)		21.7	319.7	65	520	7672
Mechanical availability	***88.1%***					
AVAILABLE DAYS/SHIFTS/HOURS		**319.7**		**959**		**7672**
Less						
Annual shutdown - strike allowance		10	309.7	30	240	7432
Wet days - power outage		5	304.7	15	120	7312
Operational availability	***95.3%***					
OPERATING DAYS/SHIFTS/HOURS		**304.7**		**914**		**7312**
Less						
Dead heading (7000m per year @ 100m/hr)					70.0	7242
Start of shift communication					152.3	7090
Operating delays (wait on dozer, cable handling, etc)					304.7	6785
Utilisation	***92.8%***					
PRODUCTION HOURS					6785	
ALLOW DIG HOURS FOR 52 MIN PER HOUR					5880	

Tables 8.2 and 8.3 are examples of the results of calculations of the detailed (block by block) and averaging (strip by strip) productivity estimations, respectively.

Table 8.2- A block by block productivity calculation of an In-Pit Bench digging method.

Sec.	Key Cut					Second Position					In Pit Bench					Overall-Total												
No	Prime Volume (bcm/m	Swing Angle deg	Time per Swing sec	Number of Swings	Total Time (hrs)	Prime Volume bcm/m	Swing Angle degrees	Time per Swing (sec)	Number of Swings	Total Time (hrs)	Prime Volume bcm/m	Swing Angle degrees	Time per Swing (sec)	Number of Swings	Total Time (hrs)	Prime Volume bcm/m	Swing Angle (degs)	Cycle Time (sec)	Walking Distance (m)	Walking Time (hrs)	Total Time (hrs)	Rehand Material (%)	Avg. Depth (m)	IPB Depth (m)	Coal Thickns (m)	Thrown Percent (%)	Total Prod. (bcm/hr)
1	1262.0	126	68.53	606.3	11.54	498.9	146	73.45	239.7	4.89	714.9	177	81.06	343.4	7.73	2917.1	144.757	73.14	126.0	1.26	25.43	7.8	52.1	39.5	6.6	22.88	2921.3
2	1264.9	128	69.02	607.7	11.65	575.7	146	73.45	276.6	5.64	628.7	175	80.57	302.0	6.76	2921.4	144.163	73.00	132.1	1.32	25.37	6.9	52.2	38.8	6.6	22.34	2919.4
3	1268.6	126	68.53	609.5	11.60	577.2	144	72.96	277.3	5.62	674.3	173	80.08	323.9	7.21	2928.8	142.698	72.64	132.6	1.33	25.75	7.1	52.3	38.9	6.6	21.05	2935.6
4	1279.4	126	68.53	614.6	11.70	494.8	145	73.20	237.7	4.83	747.9	176	80.82	359.3	8.07	2941.5	144.554	73.09	126.5	1.27	25.87	7.3	52.5	39.3	6.6	21.58	2925.3
5	1292.9	125	68.29	621.1	11.78	520.5	144	72.96	250.1	5.07	760.6	174	80.33	365.4	8.15	2959.7	143.321	72.79	128.8	1.29	26.29	7.5	52.9	39.4	6.5	20.57	2937.2
6	1315.4	127	68.78	631.9	12.07	516.5	146	73.45	248.1	5.06	749.8	176	80.82	360.2	8.09	2988.5	145.032	73.21	128.2	1.28	26.50	7.5	53.4	39.9	6.6	21.14	2922.2
7	1329.7	126	68.53	638.8	12.16	538.3	145	73.20	258.6	5.26	749.5	176	80.82	360.1	8.08	3011.9	144.225	73.01	129.2	1.29	26.79	8.3	53.6	40.6	6.7	21.44	2930.6
8	1328.3	129	69.27	638.1	12.28	596.3	146	73.45	286.5	5.84	634.0	175	80.57	304.6	6.82	3007.2	144.36	73.04	133.8	1.34	26.28	7.0	53.5	39.2	6.7	21.90	2921.0
9	1314.4	127	68.78	631.5	12.06	553.5	146	73.45	265.9	5.43	698.6	175	80.57	335.6	7.51	2981.8	144.163	73.00	130.2	1.30	26.30	7.7	53.0	39.4	6.8	21.63	2927.3
10	1296.1	130	69.52	622.7	12.02	510.1	148	73.94	245.1	5.03	686.7	176	80.82	329.9	7.41	2949.2	146.354	73.53	127.6	1.28	25.74	6.5	52.4	38.1	6.7	21.94	2905.6
11	1276.9	129	69.27	613.4	11.80	489.3	147	73.69	235.1	4.81	701.7	175	80.57	337.1	7.54	2918.6	145.648	73.36	126.3	1.26	25.42	6.3	51.9	37.6	6.7	21.74	2912.2
12	1262.0	129	69.27	606.3	11.67	451.6	147	73.69	217.0	4.44	718.3	176	80.82	345.1	7.75	2896.3	146.225	73.50	123.6	1.24	25.09	6.2	51.5	37.2	6.7	22.21	2907.8
13	1252.3	129	69.27	601.6	11.58	468.3	147	73.69	225.0	4.61	713.3	174	80.33	342.7	7.65	2884.3	145.651	73.36	124.8	1.25	25.08	6.1	51.3	36.9	6.7	21.71	2911.9
14	1251.0	130	69.52	601.0	11.61	499.5	147	73.69	240.0	4.91	663.7	175	80.57	318.9	7.14	2885.7	145.888	73.42	126.9	1.27	24.92	6.0	51.1	37.1	6.6	22.33	2906.0
15	1233.6	130	69.52	592.6	11.44	467.0	147	73.69	224.4	4.59	696.4	175	80.57	334.6	7.49	2872.2	146.386	73.54	124.3	1.24	24.77	5.9	50.7	36.9	6.6	22.44	2903.4
16	1193.3	129	69.27	573.3	11.03	376.5	147	73.69	180.9	3.70	779.8	175	80.57	374.6	8.38	2825.9	147.151	73.73	117.5	1.18	24.29	5.7	49.9	36.4	6.6	22.59	2901.6
17	1129.5	128	69.02	542.6	10.40	311.6	148	73.94	149.7	3.07	797.4	175	80.57	383.1	8.57	2736.6	147.526	73.82	112.0	1.12	23.17	4.6	48.6	34.7	6.6	22.84	2898.1
18	1057.5	126	68.53	508.0	9.67	241.9	146	73.45	116.2	2.37	892.1	174	80.33	428.6	9.56	2640.8	147.747	73.88	106.1	1.06	22.67	4.4	47.0	33.9	6.7	21.42	2900.6
19	1006.9	124	68.04	483.7	9.14	204.6	144	72.96	98.3	1.99	909.3	172	79.83	436.8	9.69	2566.9	146.51	73.57	102.8	1.03	21.85	4.3	45.9	33.4	6.6	21.65	2911.8
20	977.3	123	67.80	469.5	8.84	191.3	143	72.71	91.9	1.86	922.0	171	79.59	442.9	9.79	2521.1	145.999	73.45	101.6	1.02	21.51	4.1	45.2	32.9	6.6	21.21	2916.2
21	962.7	123	67.80	462.5	8.71	179.9	142	72.46	86.4	1.74	933.0	170	79.34	448.2	9.88	2493.9	145.774	73.39	100.7	1.01	21.34	4.0	44.7	32.5	6.7	20.81	2918.6
22	965.3	123	67.80	463.7	8.73	175.0	143	72.71	84.1	1.70	931.3	170	79.34	447.4	9.86	2491.0	145.819	73.40	100.4	1.00	21.30	4.0	44.7	32.3	6.7	20.83	2918.3
23	962.5	123	67.80	462.4	8.71	169.6	143	72.71	81.5	1.65	931.9	170	79.34	447.7	9.87	2480.3	145.864	73.41	100.1	1.00	21.22	4.0	44.6	32.1	6.7	20.74	2917.8
24	964.2	123	67.80	463.2	8.72	164.7	142	72.46	79.1	1.59	967.1	169	79.10	464.6	10.21	2479.3	145.718	73.38	99.9	1.00	21.52	3.9	44.5	32.0	6.8	19.38	2921.5
25	964.1	124	68.04	463.2	8.75	141.8	143	72.71	68.1	1.38	971.2	169	79.10	466.6	10.25	2460.4	146.338	73.53	98.7	0.99	21.37	3.6	43.7	31.2	6.7	19.21	2916.1
26	979.6	126	68.53	470.6	8.96	118.1	145	73.20	56.7	1.15	889.1	169	79.10	427.1	9.38	2445.7	146.372	73.54	97.4	0.97	20.47	3.5	43.6	31.0	6.7	22.30	2911.6
27	970.9	125	68.29	466.4	8.85	106.7	144	72.96	51.3	1.04	903.5	168	78.85	434.1	9.51	2419.5	145.634	73.36	96.6	0.97	20.36	3.5	43.1	30.5	6.7	21.57	2919.2
28	965.9	125	68.29	464.0	8.80	119.4	145	73.20	57.4	1.17	937.1	170	79.34	450.2	9.92	2409.2	147.032	73.70	97.5	0.98	20.87	3.4	42.9	30.3	6.7	19.46	2907.7
29	968.2	126	68.53	465.1	8.85	119.5	145	73.20	57.4	1.17	916.5	170	79.34	440.3	9.70	2414.1	147.254	73.75	97.5	0.98	20.70	3.4	42.9	30.3	6.7	20.38	2904.5
30	974.7	126	68.53	468.3	8.91	122.4	146	73.45	58.8	1.20	895.8	170	79.34	430.4	9.48	2425.1	147.006	73.69	97.7	0.98	20.58	3.4	43.1	30.4	6.7	21.25	2905.7
31	984.2	128	69.02	472.8	9.07	156.2	147	73.69	75.0	1.54	875.4	172	79.83	420.6	9.33	2439.5	148.58	74.08	100.3	1.00	20.93	3.4	43.3	30.9	6.7	20.80	2889.2
32	996.2	128	69.02	478.6	9.18	157.1	148	73.94	75.5	1.55	857.0	173	80.08	411.7	9.16	2454.9	148.747	74.12	100.6	1.01	20.89	3.4	43.5	30.9	6.7	21.55	2886.9
33	1000.7	129	69.27	480.8	9.25	155.9	148	73.94	74.9	1.54	838.7	173	80.08	402.9	8.96	2461.8	148.979	74.18	100.6	1.01	20.76	3.4	43.5	30.9	6.7	22.38	2883.7
34	1001.1	129	69.27	480.9	9.25	175.7	148	73.94	84.4	1.73	818.2	172	79.83	393.1	8.72	2459.0	148.309	74.01	102.1	1.02	20.73	3.4	43.5	30.8	6.7	22.28	2887.7
35	997.6	129	69.27	479.3	9.22	173.4	148	73.94	83.3	1.71	817.8	172	79.83	392.9	8.71	2452.5	148.338	74.02	101.9	1.02	20.66	3.4	43.3	30.7	6.7	22.30	2887.3
36	995.9	129	69.27	478.4	9.21	173.1	148	73.94	83.2	1.71	816.4	172	79.83	392.2	8.70	2448.2	148.338	74.02	101.9	1.02	20.63	3.4	43.3	30.7	6.7	22.30	2887.0
37	994.5	129	69.27	477.8	9.19	173.1	148	73.94	83.2	1.71	816.0	172	79.83	392.0	8.69	2445.6	148.347	74.02	101.9	1.02	20.61	3.4	43.2	30.7	6.8	22.29	2886.8
38	1008.3	129	69.27	484.4	9.32	204.6	147	73.69	98.3	2.01	799.6	172	79.83	384.1	8.52	2468.2	147.915	73.92	104.4	1.04	20.90	3.5	43.6	31.1	6.8	21.95	2889.4
39	1000.5	129	69.27	480.7	9.25	181.1	148	73.94	87.0	1.79	800.8	173	80.08	384.7	8.56	2457.3	148.51	74.06	102.5	1.03	20.62	3.5	43.4	31.0	6.9	22.80	2884.4
40	973.5	128	69.02	467.7	8.97	144.8	147	73.69	69.6	1.42	861.4	172	79.83	413.8	9.18	2413.3	148.535	74.07	99.5	1.00	20.56	3.3	42.7	30.4	6.9	21.31	2888.2

Table 8.3: A strip by strip productivity clculation of a Split Bench digging method.

Operation	Pass I		Overhand Chop		Main Underhand Dig					Dump to	Pass1	Total
	Key Cut	Main Cut	1st Cut	2nd Cut	Key Cut	1st Cut	2nd Cut	3rd Cut	4th Cut	Next Sect.	Rehandle	
CYCLE PARAMETERS												
1st Swing Angle (deg)	6.0	6.0	112.9	132.9	13.6	12.3	25.9	9.8	22.8	116.0	90.0	
1st Hoist Dist. (m)	7.0	9.0	0.0	0.0	21.6	12.2	12.2	11.7	30.4	50.7	0.0	
Swing Angle to Dump (deg)	84.0	96.0	0.0	0.0	90.0	41.7	51.2	73.7	63.3	0.0	0.0	
Hoist Dist. to Dump (m)	0.0	0.0	0.0	0.0	0.0	0.0	24.2	20.6	42.9	0.0	0.0	
Volume of prime (bcm)	5182	25803	157781	75091	869669	493329	634966	0	0	0	0	2261821
Volume of Rehandle (bcm)	0	0	0	0	0	0	0	488189	254801	120163	228	863381
Total	5182	25803	157781	75091	869669	493329	634966	488189	254801	120163	228	3125202
	Pass1 Rehandle		**0.01%**	Bridge Rehandle		**34.0%**	Ramp Rehandle		**39.3%**	Total Rehandle		**5.3%**
CYCLE COMPONENTS (seconds)												
1st Swing Time Loaded	6.3	6.3	23.7	26.2	7.5	7.3	9.5	6.9	9.0	24.1	20.9	
1st Hoist Time Loaded	4.5	5.3	0.0	0.0	9.8	6.4	6.4	6.2	13.0	20.3	0.0	
2nd Swing Time Loaded	20.1	21.6	0.0	0.0	20.9	12.0	16.0	18.8	17.5	0.0	0.0	
2ndHoist Time Loaded	0.0	0.0	2.0	2.0	0.0	0.0	8.7	7.4	15.5	0.0	2.0	
Dump Time	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	
Hoist Pay Time	1.7	2.1	0.0	0.0	5.1	2.9	8.6	7.6	17.3	11.9	0.0	
Swing Time Empty	19.8	21.3	22.6	25.1	21.5	15.4	18.2	19.0	19.3	23.0	19.8	
Drag Spot Time	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	3.0	
Drag to Fill Time	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	15.0	
Theoretical Cycle Time	67.3	70.2	69.3	74.3	73.1	55.7	64.7	65.8	70.8	68.1	63.7	
Operator Adjustment Factor	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
Adjusted Cycle Time	67.3	70.2	69.3	74.3	73.1	55.7	64.7	65.8	70.8	68.1	63.7	
HOURLY PRODUCTION												
Cycles/Hour	53.5	51.3	51.9	48.5	49.2	64.6	55.6	54.7	50.8	52.9	56.5	
Bucket Capacity (bcm/cycle)	47.0	47.0	47.0	47.0	47.0	47.0	47.0	47.0	47.0	47.0	47.0	
Material Swell Factor	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	1.25	
Bucket Fill Factor	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	0.90	
BCM/Cycle	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	33.8	
BCM/Dig Hour	**1811**	**1735**	**1757**	**1640**	**1665**	**2187**	**1882**	**1852**	**1720**	**1789**	**1913**	
% Rehandle	0.0	0.0	0.0	0.0	0.0	0.0	0.0	21.6	11.3	5.3	0.0	39.3
% of Total Yearly Volume	0.2	0.8	5.0	2.4	27.8	15.8	20.3	15.6	8.2	3.8	0.0	100.0
% of Total Yearly Time	0.2	0.9	5.2	2.7	30.4	13.1	19.6	15.3	8.6	3.9	0.0	100.0
Weighted Average Rate (bcm/op.hr)		**181973**										

8.2.6 Sensitivity Analysis

Once the productivity calculations have been completed, it is desirable to perform further sensitivity analysis in order to determine the optimum design variables. A common method of sensitivity analysis is to vary each component variable at a baseline ± 5% and observe how much change has been introduced to the target function. To optimise an operational parameter, the selected variable is first changed within a practical range and the dragline simulation is repeated for each new situation. The productivity is recalculated for each case to assess the effect of the changes made on the total operation.

Dragline productivity is a complex function of many variables. In general, geological and geotechnical parameters cannot be controlled in a mine site. The mine must be planned in such way that it is fairly insensitive to these parameters whenever possible. In practice the only control available is the selection of the dragline and its deployment. In most cases there is not much flexibility in changing dragline specifications. Therefore, factors such as pit configuration and mode of operation are the only parameters that can be controlled (Ramani and Bandopadhyay, 1985).

In a simple sensitivity analysis, only the effect of one input value is examined at a time and it is assumed that all other variables remain constant. However, such an approach does not attempt to quantify the effect of the inherent randomness of the parameters and it also ignores any combined effect of parameters. Sensitivity analysis does not in itself assess the *risk* of achievable production targets subject to changes in dragline operating parameters. To measure risk, the probability of change occurring also has to be considered (Runge, 1994). Usually stochastic (also called probabilistic) methods are used in quantifying uncertainty within a model to provide a logical and systematic analysis of uncertainty.

8.3 STOCHASTIC (RISK) ANALYSIS

To conduct a stochastic analysis, probability distributions must be determined for some or all of the uncertain values in the model. A sampling technique such as the Monte Carlo technique is then applied to determine the possible range of results for the target measures. Since such a technique is explicitly addressed through the use of distributions, the outcomes are also described in terms of the distribution of a set of possible values rather than a constant value. Techniques for risk analysis attempt to assess the overall design risk assuming certain risk characteristics of the critical values. After repeated simulations, a statistical distribution of probable results is obtained, from which the probability can be deduced of the actual results being greater or less than some cut-off criteria.

In this part of the thesis @RISK© software was used to perform the stochastic analysis. @RISK© is a product of Palisade Corporation and works with spreadsheet software as an add-in option. A major advantage of @RISK© is that it allows the user to work in a familiar and standard spreadsheet modelling environment such as Microsoft Excel and Lotus 1-2-3. A generic flowchart of the procedure is presented in Figure 8.1. In general, to perform a risk analysis for a productivity calculation using @RISK© the following four steps are required:

1. Create a model to establish the relationship between the variables and calculate the productivity in a spreadsheet format.
2. Identify certain and uncertain input values. For the uncertain variables, specifying their possible values with probability distributions. The uncertain results and the target values which are to be analysed must be identified in this stage as well.
3. Simulate the model many times, each time using different randomly selected sets of values from the defined probability distributions by applying a sampling approach such as the Monte Carlo technique.
4. Determine the range and probabilities of all possible outcomes for the results using stored statistics from the simulation runs.

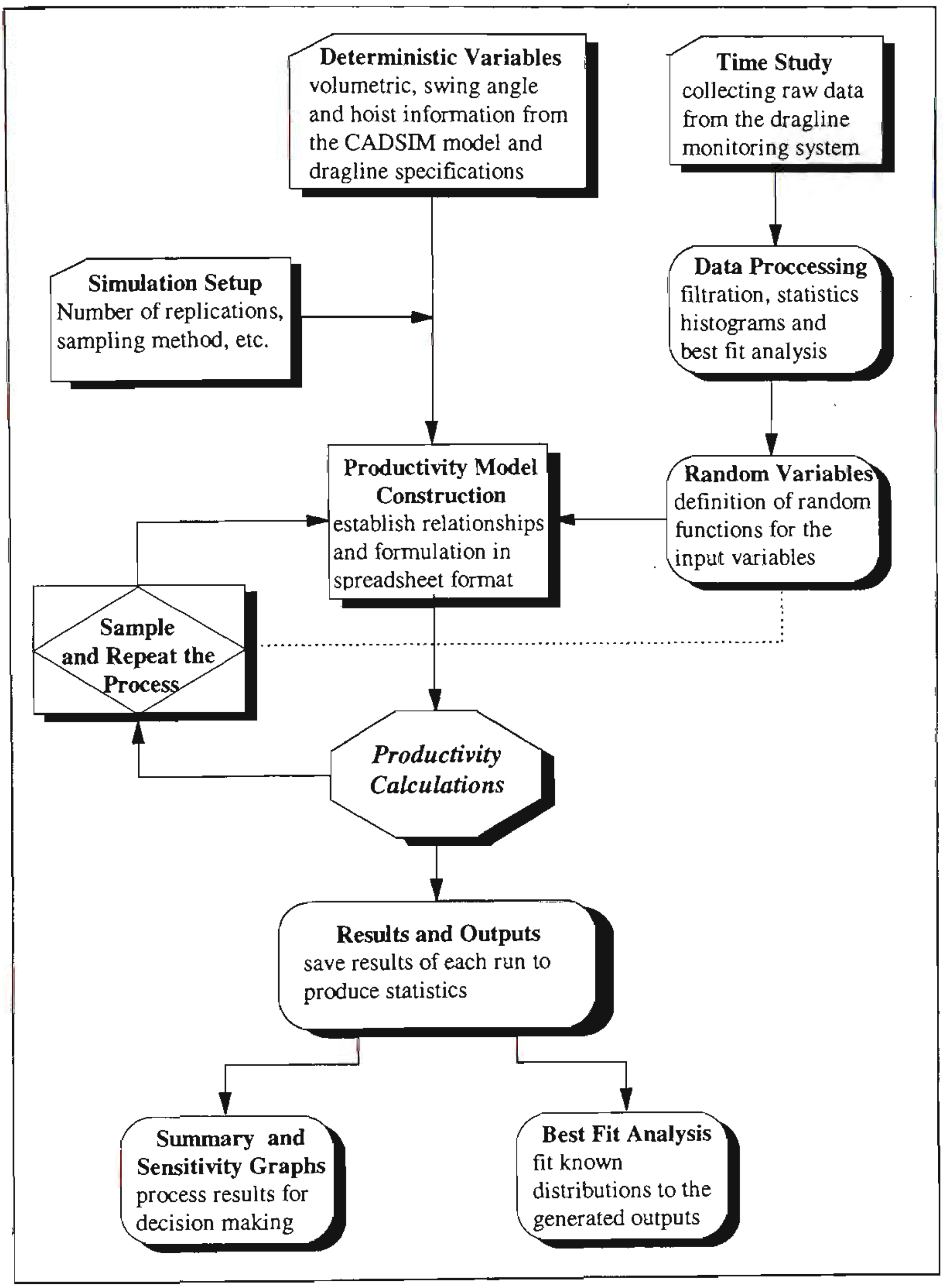


Figure 8.1- A generic flowchart for the stochastic analysis of productivity.

In a dragline operation, the input parameters are governed by geological and geotechnical characteristics of the deposit, the machine specifications, pit configuration and mode of operation. Only a few parameters such as some cycle time components (ie. filling time, dump time) can be treated as randomly distributed variable functions. A list of random variables used and their specifications in the stochastic productivity calculations are presented in Table 8.4.

Table 8.4 - Random variables used for the stochastic productivity calculation.

Input Random Variables	Distribution	Min.	Max.	Mean	Std. Dev.
Dump Time (Lowwall Side) (sec.)	LOGNORMAL	1	25	6.2	2.78
Dump Time (Highwall Side) (sec.)	BETA	1	35	8.2	3.43
Filling Time (Lowwall Side) (sec.)	BETA	2	55	19.6	10.2
Filling Time (Highwall Side) (sec.)	ERLANG	2	40	14.6	6.72
Filling Factor (Lowwall Side)	NORMAL	0.1	1.39	0.913	0.193
Filling Factor (Highwall Side)	NORMAL	0.1	1.40	0.968	0.192

To select values of the input data that correctly reflect the random variations, the values must be sampled from a distribution that reflects the appropriate range of possible values and their relative frequencies. @RISK© randomly assigns values to uncertain variables by using either a Latin Hypercube or a Monte Carlo sampling technique (@RISK 3.0 User's Guide Manual, 1994).

The objective of conducting the simulation is to assess the likely distribution of the project's target resulting from uncertainty in some or all input variables. To make this assessment, it is important that a reasonably large number of replications be performed. The number of iterations required to generate reasonable outputs varies depending on the variable being simulated and their distribution functions. For example, more complex models with highly skewed distributions will require more iterations than simpler models. Fewer than 100 replications is usually insufficient. To have a reasonable distribution and stable results, at least 250 replications should be considered (Seila and Banks, 1990). It is also important to run enough iterations so that the statistics generated on the outputs are reliable. As more iterations are run, the change in the statistics become less and less until they converge. This implies that the output distributions created during the simulation become more "stable".

A coefficient of convergence can be defined and used to monitor the stability of the simulated results. The statistics produced in each new number of simulations are then compared with the same statistics calculated from the previous set of simulations. The amount of change in the statistics due to the additional iterations is then calculated. To determine the convergence coefficient for the mean and the standard deviation the following relationship can be applied:

$$\textit{Convergence Coefficient} = 1 - \frac{|\text{New Value - Old Value}|}{\text{New Value}}$$

The relationship between the Convergence Coefficient and the number of replications for the monitored statistics of the prime productivity calculation is shown in Figure 8.2. As the graph suggests, the simulation reaches a stable condition after 300 replications for the mean value, but in the case of the standard deviation at least 1000 replications are necessary for convergence.

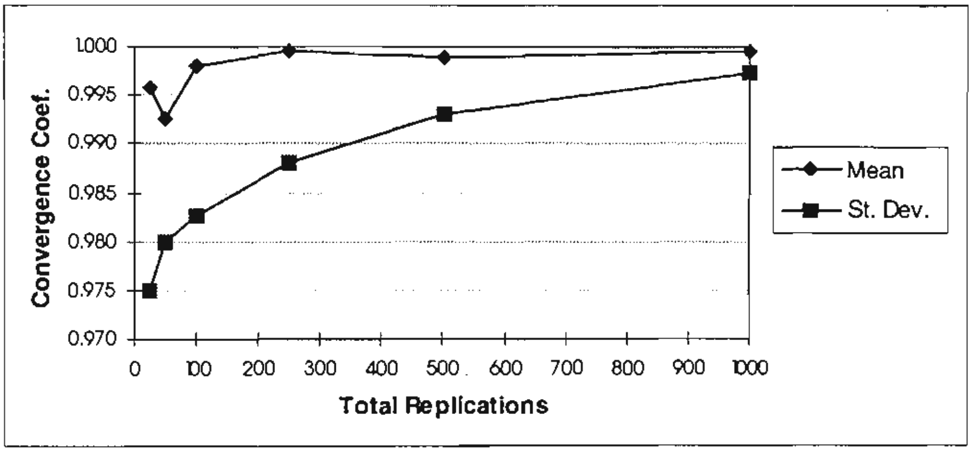


Figure 8.2- Effect of number of simulation replications on the convergence coefficient.

8.3.2 Case Study Stochastic Analysis

A stochastic model was constructed and combined with the productivity model developed for a case study to calculate both the prime and total productivity terms. The distribution of cycle time and dragline performance variables were available from time study data described in Chapter 7. Following 2000 iterations for the optimum pit configurations, distributions of possible values of the total and prime productivity were obtained (Figures 8.3 and 8.4). The direct calculation indicated a productivity level of 9.08 (Mbcm/y) for annual prime and 9.72 (Mbcm/y) for annual total productivity. Stochastic modelling resulted in mean productivity values of 9.22 (Mbcm/y) and 9.85 (Mbcm/y) for prime and total productivity respectively.

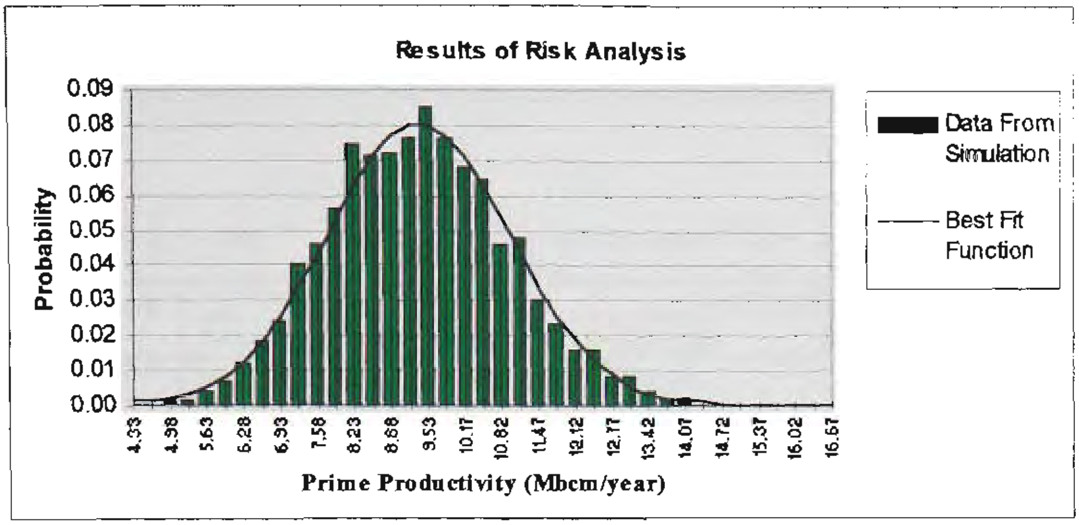


Figure 8.3- Probability histograms of simulation results for annual prime productivity.

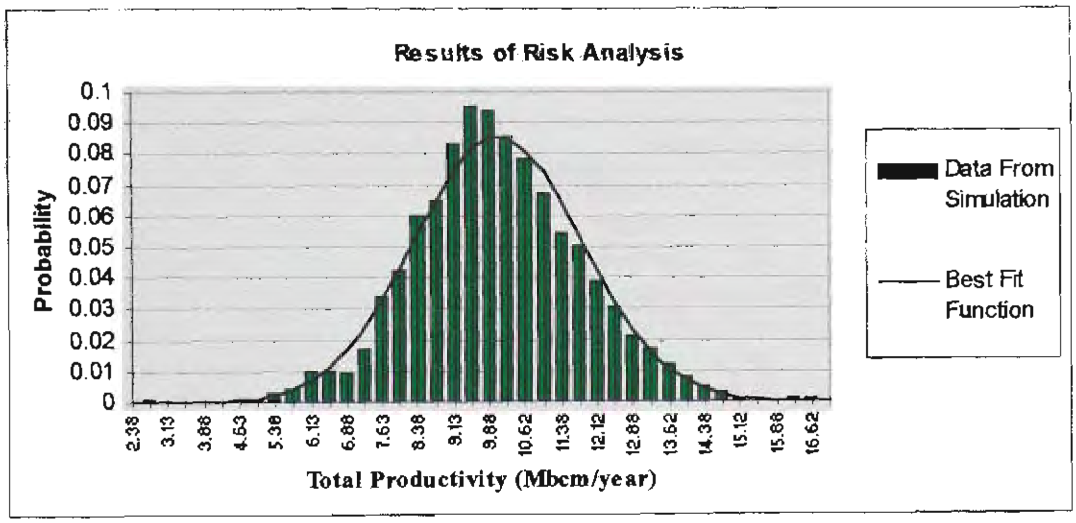


Figure 8.4- Probability histograms of simulation results for annual total productivity.

Best fit analyses were also performed on the results of the risk analysis for both prime and total productivity, using the capabilities of the ARENA software. Tables 8.5 and 8.6 also describe the results of the best fit ranking and statistics for a normal distribution as the "best" function fitted to the data for both cases.

As with the histogram graphs, cumulative graphs can be produced to gain a better understanding of the results. The ascending and descending cumulative graphs are used to estimate the probability that the actual value will be greater or less than a particular value. All of these estimates are measures of risk and will provide more information than a single value calculated from the most likely values of the random variables (Seila and Banks 1990). Figure 8.5 and 8.6 are ascending and descending cumulative graphs for the results of the risk analysis for both annual prime and annual total productivity. For example, from Figure 8.5 it can be seen that there is a 45% risk that the annual prime productivity drops to value less than 9.5 (Mbcm/y). Similarly, in Figure 8.6 it is shown that there is only 25% chance that the annual prime exceeds 11 (Mbcm/y).

Table 8.5- Best fit analysis results on stochastic simulation outputs.

Rank	Prime Productivity		Total Productivity	
	Function	**Sq Error**	**Function**	**Sq Error**
1	Normal	0.000989	Normal	0.000658
2	Gamma	0.00103	Gamma	0.000803
3	Erlang	0.00104	Erlang	0.00092
4	Beta	0.00114	Beta	0.000932
5	Weibull	0.00119	Weibull	0.00169
6	Lognormal	0.00159	Lognormal	0.0017
7	Triangular	0.00638	Triangular	0.0152
8	Uniform	0.0245	Uniform	0.0377
9	Exponential	0.0374	Exponential	0.05

Table 8.6- Statistics from best fit analysis on the simulation outputs.

Prime Productivity	Total Productivity
Statistics	**Statistics**
Maximum : 14.42	Maximum : 16.3
Minimum : 3.85	Minimum : 2.55
Sample Mean = 9.22	Sample Mean = 9.85
Sample Std Dev = 1.57	Sample Std Dev = 1.74
Variance = 2.47	Variance = 3.04
Skewness = 0.13	Skewness = 0.12
Kurtosis = 2.90	Kurtosis = 3.40
Mode = 9.16	Mode = 9.77
Distribution Function	**Distribution Function**
Normal- NORM(9.22, 1.57)	Normal- NORM(9.85, 1.74)
Sq Error = 0.000989	Sq Error = 0.000658

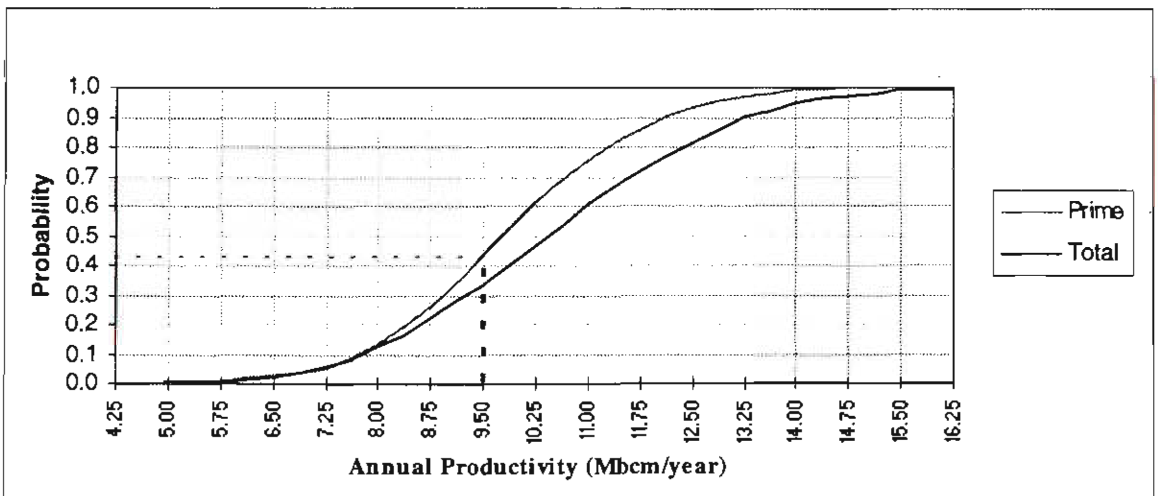


Figure 8.5- Ascending cumulative graph for annual productivity.

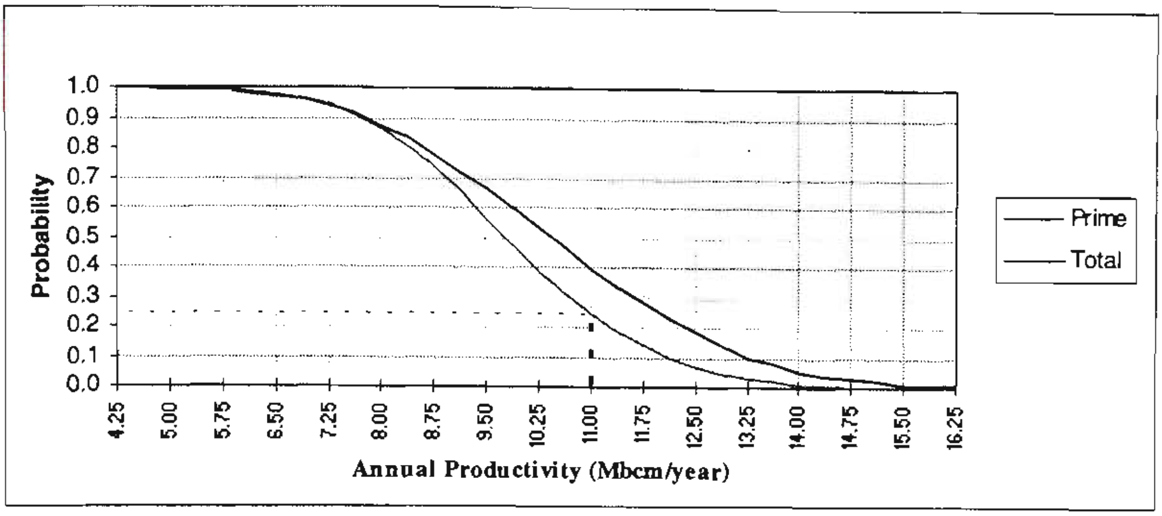


Figure 8.6- Descending cumulative graph for annual productivity.

8.3.2.1 Pit Optimisation using the Stochastic Approach

Stochastic productivity calculations were used to optimise dragline pit geometry for a given stripping method and dragline specification. Practical strip widths were first simulated by the CADSIM model. The results from the CADSIM model were summarised and imported into the stochastic model in a spreadsheet. Table 8.7 summarises the statistics produced after 2000 iterations for various strip widths.

Table 8.7- Statistics of the result of simulation for different strip widths.

Strip Width	**40m**		**50m**		**60m**		**70m**		**80m**	
Productivity (Mbcm/y)	**Prime**	**Total**	**Prime**	**Total**	**Prime**	**Total**	**Prime**	**Total**	**Prime**	**Total**
Minimum	4.16	2.57	3.85	2.55	3.80	2.50	3.74	2.45	3.64	2.40
Maximum	14.16	16.72	13.97	16.31	13.79	16.12	13.55	15.85	12.96	15.11
Mean	9.38	10.03	9.22	9.85	9.13	9.75	8.99	9.61	8.64	9.23
Std Deviation	1.61	1.80	1.57	1.74	1.55	1.72	1.52	1.69	1.45	1.60
Variance	2.60	3.25	2.47	3.04	2.41	2.97	2.32	2.87	2.09	2.57
Skewness	0.14	0.15	0.13	0.12	0.13	0.11	0.12	0.11	0.11	0.09
Kurtosis	2.87	3.35	2.90	3.40	2.90	3.40	2.89	3.40	2.89	3.42
Mode	8.91	10.00	9.16	9.77	9.05	9.65	8.89	9.48	8.53	9.07

Figures 8.5 and 8.6 are the summary graphs of prime and total productivity calculations produced from the simulation. The graphs show that narrower strips are more productive in this case study. After discussing these results with the mine manager a strip width of 50m was selected as the optimum width to meet the practical requirements for coal handling.

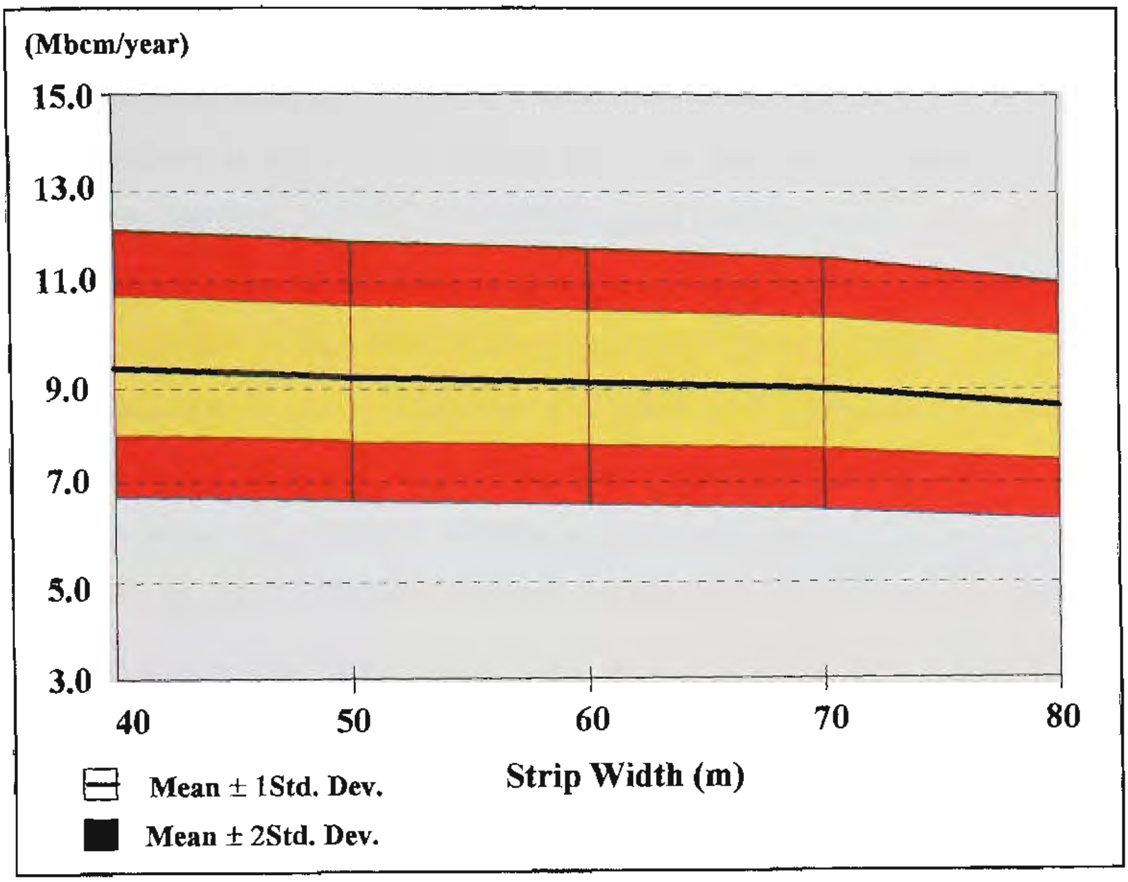


Figure 8.7- A summary graph of the effect of strip width on prime productivity.

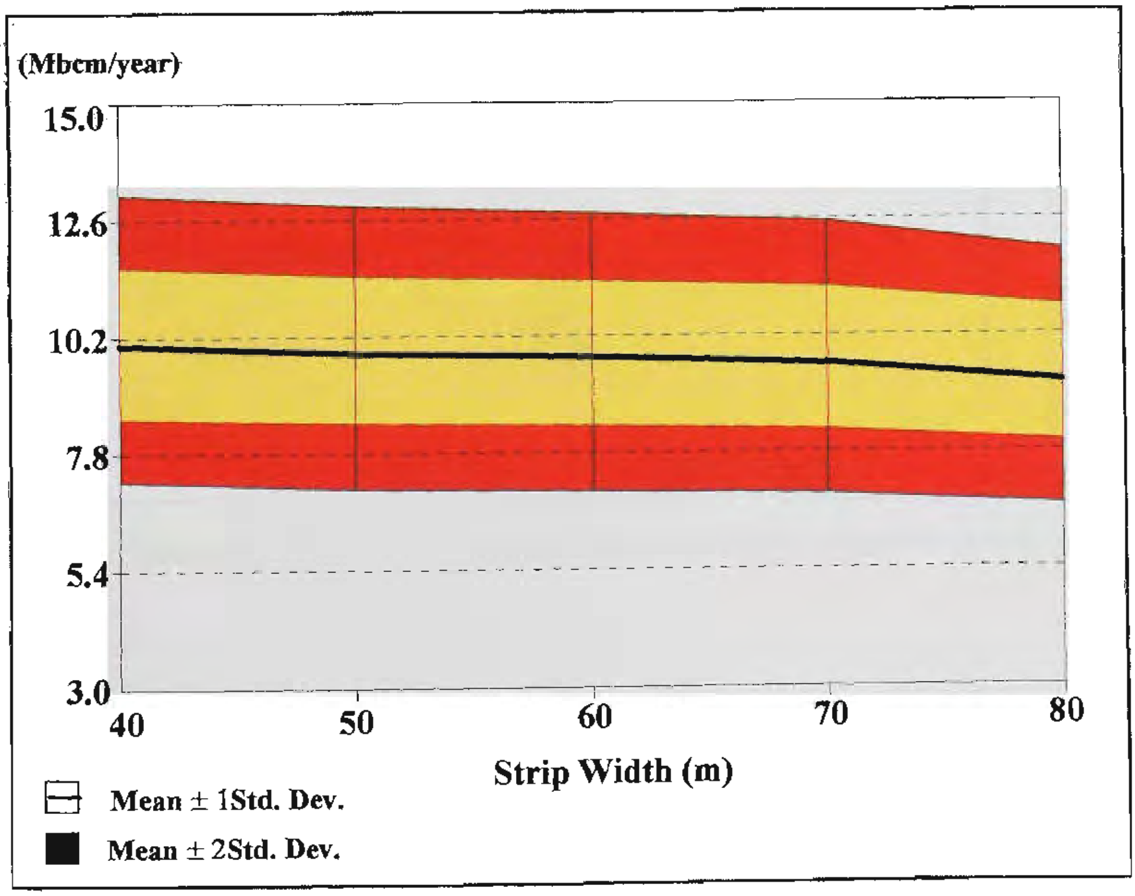


Figure 8.8- A summary graph of the effect of strip width on total productivity.

@RISK© software was used to perform a sensitivity analysis by a multi-variate stepwise regression analysis. Figure 8.9 shows a *Tornado* graph of variables that the prime productivity is sensitive to (Table 8.4). As the results suggest, filling factor parameters for both highwall and lowwall stripping are the most critical parameters to prime productivity. In Figure 8.9 a negative coefficient value for a parameter means that productivity can be increased with a reduction in that parameter. The case study mine presented here is a multi seam operation with the dragline removing the last two interburdens from the lowwall side. This means the dragline spends more time on the lowwall side removing a greater volume of the waste than the highwall side. This explains why prime productivity is more sensitive to changes in the lowwall side parameters.

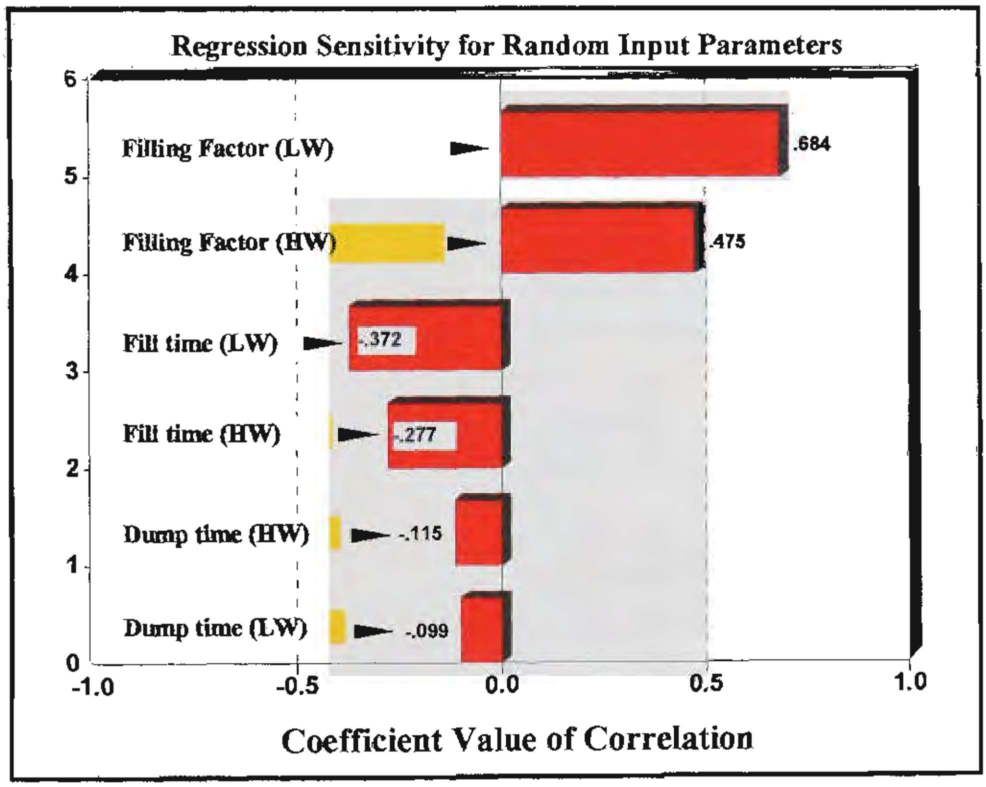


Figure 8.9- Sensitivity analysis of prime productivity against uncertain input variables.

8.4 COST ANALYSIS

Figure 8.10 shows a summary of the costing procedure which was designed to complete the dragline digging method selection process. The two distinct phases are:

1- The first phase is to estimate the capital and operating costs based on the productivity calculations and operational requirements.

2- The second phase is to conduct the financial analysis using a modified cash flow technique called the Discounted Average Cost method. The end results of this technique are average costs for the digging option.

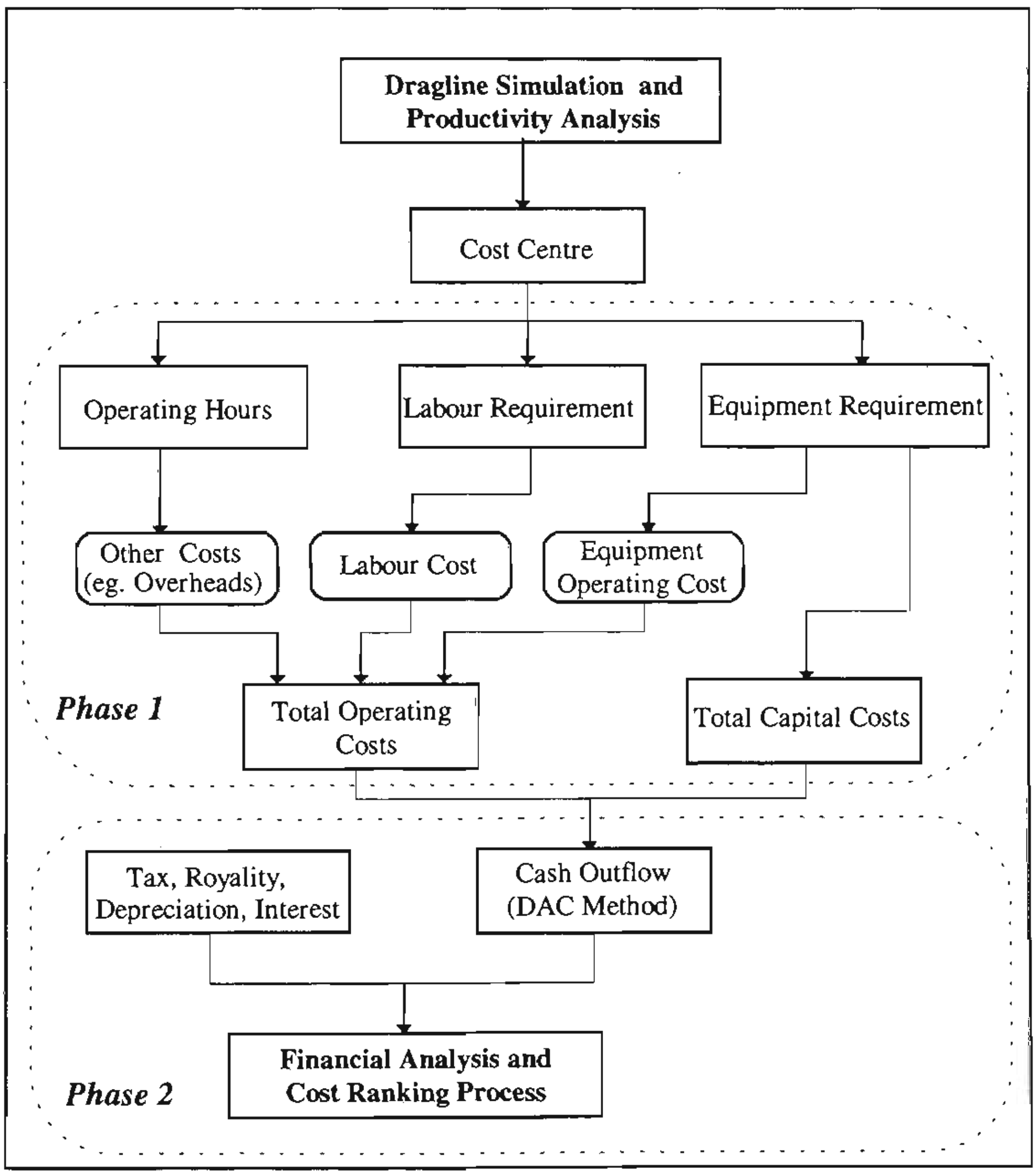


Figure 8.10- Costing flow chart (Modified after Noakes and Lanz, 1993).

Both phases are repeated for each major cost component to arrive at the Discounted Average Cost of the components. The component costs could be added together to provide the total cost and one cash flow then used for the whole operation. However, for comparison purposes it is preferable to separately analyse each component. This will identify the contribution of each part to the total costs so that the source of higher costs can be readily identified. The major cost components associated with dragline stripping and waste removal considered in this thesis are:

1. drill and blasting operation,
2. dragline operation, and
3. dozing operation.

The first step in developing a cost model is to develop a cost data base information for mining activities and mining equipment. There is no single, simple and reliable source of cost information for the mining industry. Typical sources of data are:

- Historical data of an ongoing operation,
- Historical data of similar mines using the same methods of operations,
- Manufacturers, consultants, banks and government agencies,
- Contractor quotations,
- Rules and formulae available in the literature.

8.4.1 Capital Costs

The total costs associated with the purchase and installation of the equipment is calculated as capital cost. For example the total capital cost of a walking dragline is a combination of the following costs (USBM, 1987):

1. purchased equipment cost (77%),
2. construction labour cost (20%),
3. construction supply cost (1%), and
4. transportation cost (2%)

The auxiliary equipment capital costs associated with a walking dragline are typically 3% of the dragline capital cost.

Depreciation and equipment life are other capital related items which must be calculated for cash flow purposes. The principal purpose of depreciation is to allocate the capital cost of an asset to the period during which the asset makes a contribution toward earning revenue (EPRI, 1981). Depreciation is a tax allowance that is assigned over a number of years for capital expenditure. In any year this allowance is subtracted from the pre-tax profit thereby reducing the tax payable. A straight line depreciation method is used in this thesis.

8.4.2 Operating Costs

Operating costs are best estimated from field studies and mine records. A breakdown of the typical operating costs of a medium size dragline is shown in Figure 8.11.

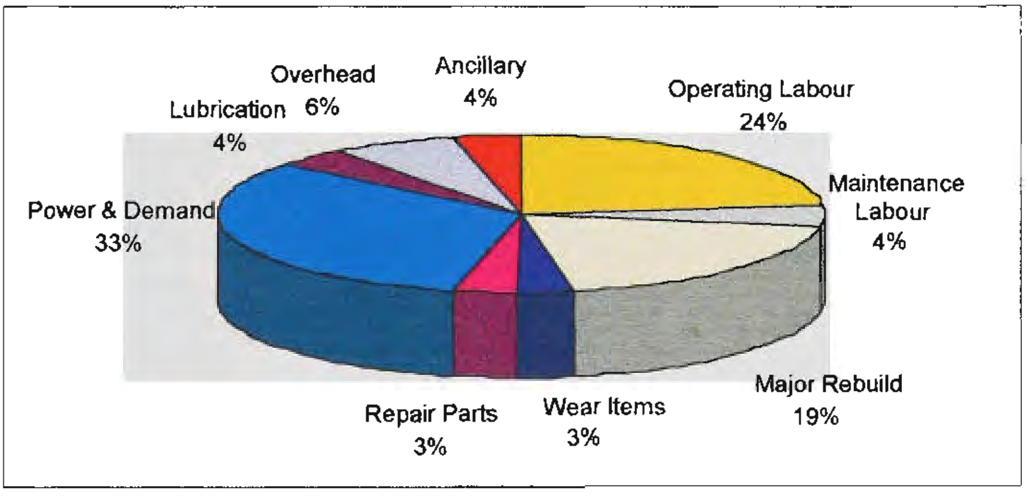


Figure 8.11- Breakdown of operating costs for a 43 m^3 bucket Marion 8050 dragline.

8.4.2.1 Labour Cost

Labour costs typically represent over 40% of the controllable operating costs in Australian open cut coal mines (Noakes and Lanz, 1993). The labour costs are calculated on a weekly and an annual basis. The total labour costs must be broken down into direct (operating) and indirect (maintenance) costs. To determine the total cost of each group it is necessary to include following items:

Shift roster: This information is used to determine the annual working and operating time for each labour group. It defines the pattern and schedule of work, period and amount of payments such as sick leave, annual leave, workers' compensation, etc. From the nominated shift roster for an item of equipment, the number of annual hours worked can be calculated.

Group/Level of employee: This item defines the weekly rate of payment depending the skill, experience and nature of the operation for both operating and maintenance labour. Total weekly and annual labour costs are estimated here based on the award type which is set by New South Wales Mineral Council Award Services (NSW Mineral Council Award Services, 1996). Table 8.8 is an example of NSW weekly and annual labour costs for an experienced operator of a dragline of less than 46 m^3.

Table 8.8 - Typical weekly and annual cost of a dragline operator (After Westcott et al, 1991).

Description	Cost per Week	Cost per Year
Base Wage	A$604.80	A$31328.64
Above Award Pay Increment	0.00	0.00
Adjusted Base Wage	604.80	31328.64
Maximum Hours per Week @ Normal Rate	35.00	1813.00
Maximum Hours per Week @ 1.5 Times	10.00	518.00
Maximum Hours per Week @ 2.0 Times	0.00	0.00
Equivalent Hours	15.00	777.00
Total Overtime Cost	259.20	13426.56
Total Adjusted Cost	864.00	44755.20
Average Shift Premium	86.40	4475.52
Weekly Bonus	180.00	9324.00
Other Shift Allowance	47.40	2455.32
Average Gross Wage	1177.80	61010.04
Sick Leave	58.74	1814.40
Public Holidays	39.16	1209.60
Annual Leave (Loading)	97.90	3024.00
Long Service	31.33	967.68
Compassionate	31.33	967.68
Total Gross Wage	1436.26	74398.34
Workers Compensation	39.50	1220.20
Payroll Tax	120.43	3719.92
Pension/Superannuation	144.52	4463.90
TOTAL LABOUR COST	**A$ 1740.72**	**A$ 90169.10**

Manpower number: This covers the total number of labourers required to run a given machine. It should also allow for shift roster coverage, absenteeism and multiple operations on one machine. Normally, a specific item of equipment requires a certain minimum number of workers when it is operating. For example, a dragline requires an operator and an oiler. Usually large equipment such as a dragline or shovel is manned even during maintenance (Westcott and Hall, 1993). The following formula can be used to determine the number of labour required.

$$Manpower\ Number = \frac{Manned\quad Yearly\quad Hours}{No.\quad of\quad Hours\quad Worked\quad per\quad Year} \times Absenteeism\quad Factor$$

For a large walking dragline with a four panel roster (4×7 continuous shift roster), the total number of hours worked per annum at a normal rate is 1985 hours. Assuming two operators remain with the machine on service days and that the leave and absenteeism runs at 13%, the total number of men required is calculated as follows (see Table 8.1):

$$Manpower\ Required = 2 \times \left(\frac{8712}{1985} \times 1.13 \right) = 10$$

In this example with two operators and a four panel roster only eight people are available and the extra hours must be obtained through overtime payment, which is at a different rate. Therefore, total labour cost per annum for a dragline operation can be calculated as follows:

$$Total\ Yearly\ Cost = (Yearly\ Wage)\ (No.\ @\ Ordinary\ Rate + No.\ @\ 1.5\ Rate)$$

$$= \$90{,}170 \times [8 + (2 \times 1.5)] = \$991{,}200$$

$$Hourly\ Op.\ Labour\ Cost = Yearly\ Cost\ /\ Op.\ Hours = \$991{,}200\ /\ 8712\ hr = 114\ \$/hr$$

Maintenance ratio: To calculate the maintenance labour requirements often a maintenance ratio is applied for each equipment. This is a ratio of repair man-hours required per equipment production hours. For an example for a dragline with the maintenance ratio of 2 and production hours 6700 hrs/year, the total maintenance man-hours required are equal to 13400 hrs/year.

Locality: The total number of men required for a given fleet of equipment can vary considerably from region to region. For example, in New South Wales a four panel dragline roster is used while a five panel roster is used in Queensland.

8.4.2.2 Supply Costs (Consumable)

Supply costs include electrical, fuel and lubricant charges. Operating costs associated with electrically powered equipment include a charge for energy consumption as well as a maximum demand charge. The maximum demand charge reflects the installed capacity of the power generating facility. Demand is usually estimated at 10% to 15% higher than the average power, however the demand charge is highly sensitive to the number of electrically powered items of equipment and the schedule of operation.

Fuel costs are obviously based on the cost of fuel, as well as the consumption rate and working conditions. Lubrication costs are usually be calculated as 20% to 40% of the fuel costs, depending on the proportion of hydraulic components of the equipment. For equipment such as draglines with no fuel consumption, the lubrication cost is calculated based on a consumption rate expressed as litres per hour which can be obtained from the manufacturers data or operational records. The fuel consumption rate is then multiplied by its appropriate unit cost to provide an hourly lubrication cost.

8.4.2.3 Repair and Wear Items

The cost of repair and replacement of worn parts, also called maintenance supplies is not easy to calculate. A simplified and commonly used method for this calculation is to calculate the hourly cost as a percentage of the equipment capital cost divided by the number of operating hours per year. Typical values for the repair parts factor range from 3% to 10 % of the capital cost of the equipment.

Wear items, also called operating supplies, include such items as bucket teeth, ropes, cutting edges and so on. A common method of calculating the hourly cost of the wear items is to divide the cost of each individual item by its estimated life and then sum up all the costs. This method requires a good understanding of all wearing items, their costs as well as their average operating life. Another similar method used for repair estimation can be to estimate the cost of wear items. With this approach a yearly wear part factor can be applied to capital costs and the result divided by the number of operating hours per year. A yearly wear part capital factor can vary between 0.1% to 0.4%, depending on ground conditions, rock hardness and abrasiveness (Noakes and Lanz, 1993).

8.4.2.4 Major Overhauls

The overhaul items cover the major equipment items exchanged or rebuilt during the life of the equipment. If adequate information is available, this can be estimated as the cost of building up individual components such as body, dragline tub and frame divided by the frequency of exchange. Alternatively, another approach is to assume that a proportion of the initial capital cost will be required for equipment rebuild after a specific period. Typically, for large equipment, this will be 15% of the initial capital cost with a frequency of every 12,000 hours. Table 8.9 sets out a series of costing factors for a normal job condition. (Runge, 1992).

Table 8.9- Typical factors for various equipment.

	Walking Dragline	**Dozer**	**Waste Drill**	**Coal Drill**	**Grader**
Typical Life (op. hr)	100,000	20,000	75,000	35,000	18,000
Repair Factor (Typical Life)	0.035	0.25	0.15	0.25	0.25
Major Overhaul (% Capital)	3%	15%	10%	12.5%	15%
Frequently of Major re-builds (hr)	20,000	10,000	15,000	10,000	10,000
Maintenance Ratio (man-hr/op.hr)	1.7 - 3.0	0.5 - 0.8	1.1 - 1.6	1.1 - 1.5	0.3 - 0.5

8.4.2.5 Other Indirect Costs

In addition to the major items of mining equipment at a site, there are a large number of smaller items which should be considered in a normal costing procedure. Individually, these costs are low compared with the major operating costs such as labour or supply costs, but cumulatively they often contribute significantly to total operating costs.

Typically, most of these costs do not have a direct relationship to the pit operations, however these costs must be incurred by major equipment and operating components. Items in this category may include:

- Ancillary equipment (such as water/fuel trucks, light vehicles and equipment, etc.,
- Administration (labour and consumable),
- Engineering design,
- Rehabilitation and environmental,
- Safety and training,
- Development and construction, and
- Miscellaneous.

When performing a preliminary feasibility study for comparison purposes, many estimators do not include indirect costs due to the complexity of measurement and allocation of these costs. As in the case of direct costs, indirect costs are site specific. For a quick estimate, an additional 15% to 20% of the total operating costs can be added to account for the total minor costs. Also some operations may treat some of the above costs such as administration and rehabilitation costs as a separate cost centre.

8.4.3 Major Equipment and Blasting Cost Calculation

Using the formulae and factors described in the previous sections, a spreadsheet was prepared to calculate the operational costs of the major equipment. The calculated costs were then entered into a cash flow table for calculation of Discounted Average Costs. Table 8.10 is an example of the calculation of operating costs.

Table 8.10- Equipment operating cost calculation.

Cost Component	Dragline	Drill	Dozer
General			
Equipment capital cost ($)	60,000,000	1,250,000	1,200,000
Operating hours (hr)	6000	3000	4000
Operational Costs			
Power			Not Applicable
Power unit cost ($/kW-hr)	0.059	0.059	
Usage (kW/hr)	10000.0	250.0	
Total power post ($/hr)	590.0	14.75	
Fuel	Not Applicable	Not Applicable	
Fuel unit cost ($/lt)			0.38
Average consumption (l/hr)			50.0
Total fuel cost ($/hr)			19.0
Lubrication			
Lubricant unit cost ($/l)	3.0	3.0	
Average consumption (l/hr)	15.0	12.0	
Percentage of fuel cost			60.0%
Total lube cost ($/hr)	45.0	36.0	20.37
Repair Parts			
Capital cost repair factor	0.03	0.10	0.15
Total repair cost ($/hr)	300.0	41.6	45.0
Wear Parts			
Capital cost wear factor	0.015	0.17	0.15
Total repair cost ($/hr)	250.0	70.8	45.0
Major Overhaul			
Major rebuild cost per year ($/year)	500,000	150,000	100,000
Total rebuild cost($/hr)	83.33	50.00	16.67
Labour			
Maintenance			
Manpower required (ratio)	2	1	0.4
Hourly wage($/hr)	22.0	22.0	22.0
Total cost ($/hr)	44.0	22.0	8.8
Operating			
Manpower required	8	6	4
Absentee factor	1.2	1.2	1.2
Total cost ($/hr)	160.0	132.0	44.0
Grand total costs ($/hr)	**1472.33**	**367.15**	**198.84**

Table 8.11 is an example of the drilling requirements and blasting cost calculations. Without a thorough study of the physical parameters of material and field tests only broad estimates can be made. In this thesis basic data such as the required powder

factor, explosive type, drilling patterns and capital costs were provided by the case study mines.

Table 8.10- Calculation of blasting cost.

Component	*Value*
General	
Dragline productivity (bcm/hr)	4055
Dragline operating hours (hr)	6000
Drilling operating hours (hr)	3000
Drilling pattern	
Hole depth (m)	45
Hole diameter (mm)	288
Spacing (m)	11
Burden (m)	7.5
Penetration rate (m/hr)	30
Required drilling (m/bcm)	0.012
Required drilling (bcm/m)	82.50
Required drilling (m/hr)	49.15
Number of drills	3
Annual productivity (m)	270,000
Annual productivity (bcm)	22,275,000
Blasting costs	
Explosive cost ($/kg))	0.542
Accessories (% of explosive costs)	10%
Total cost ($/kg)	0.596
Powder factor (kg/bcm)	0.685
Total cost ($/bcm)	0.408
Total cost ($/hr)	**2450.4**

8.4.4 Discounted Average Cost Method

When planning a dragline operation and analysing the various alternatives available, unit cost is normally the criterion used for decision making. The unit cost is generally computed for comparison purposes on the basis of either cost per unit volume of waste (bcm) or per tonne of coal. There are a number of investment criteria used to rank alternative options based on the economics of the project. Three most widely used criteria are Net Present Value (NPV), Internal Rate of Return (IRR), and Payback Period method. All these methods can be calculated using a discounted cash flow (DCF) method (Schenck, 1985; Sorentino, 1994).

The DCF technique considers capital and operating costs and measures the time value of money. However, many examples do not lend themselves to such an analysis. For example, if a company is comparing alternative draglines for purchase and those draglines are only used to remove waste there are no direct revenues, since all of the cash flow is *outflow,* and the conventional DCF analysis will not work. For this kind of problem a variation on the conventional DCF technique, termed Discounted Average Cost Method (DAC) is adopted (Runge, 1992). The discounted average cost of production is the *price* which yields a cash flow giving an NPV of zero when discounted at the required interest rate.

Table 8.12 is a spreadsheet table prepared to calculate discounted average cost of a dozer operation. In this example the equipment life is five years and the last line must sum to zero for out-flow and in-flow net present values. The unit rate (ie. revenue per bcm) is calculated iteratively in the spreadsheet to ensure the net present value is zero at the end of equipment life.

The DAC method is suitable for decision making and takes into account equipment replacement strategy, depreciation, tax, and the discount rate. This technique does not require any production price for the cash flow analysis. The method focuses on the tasks at hand (ie. stripping, coal mining) in isolation from the effect of different revenue streams which may bias the results. The final objective in using this technique is to determine what price would apply using a specific method so that it can be compared with other methods available.

Table 8.12- Cash Flow analysis of the Dozer operation using a Discounted Average Cost method.

	0	1	2	3	4	5	Total
Dozer Productivity, bcm/hour		100	99	98	97	96	
Schedule Annual Op. Hours		3000	3000	3000	3000	3800	
Total Material Dozer, bcm		33000000	32670000	32343300	32019867	40152913	**170186080**
Capital Cost							
Cost of Dozer	**2000000**						
Trade in Value							
Value for Depreciation		2000000	1600000	1200000	800000	400000	
Claimable Depreciation		400000	400000	400000	400000	400000	**2000000**
Operating Cost							
Operating Costs/Op.Hours		140	142	145	148	151	
Total Operating Cost		418680	427054	435595	444307	574044	**2299679**
Financial Calculation:							
Contract Price @ $0.035/bcm		1791892	1773973	1756233	1738671	2180294	
Less Operating Cost		418680	427054	435595	444307	574044	
Nett Operating Surplus		1373212	1346920	1320639	1294365	1606249	
less Depreciation Allowances		400000	400000	400000	400000	400000	
Profit for Taxation		973212	946920	920639	894365	1206249	
Less Tax Payable @ 39%		379553	369299	359049	348802	470437	
Plus Depreciation Allowances		400000	400000	400000	400000	400000	
Nett Cash Flow	**-2000000**	601857	589737	577584	565397	659085	
Discount Factor @ 15 % ROI	**1.0000**	0.8696	0.7561	0.6575	0.5718	0.4972	
Net Present Value	**-2000000**	523354	445926	379771	323268	327682	**0**

Equivalent Operating Cost	**0.0135**
Equivalent Capital Cost	**0.0213**
Discounted Average Cost	**0.0348**

Using a discounted average cost analysis, the overall cost of each component or cost centre (ie. dragline operation, drill and blast and dozing costs) can be estimated. The end result of the analysis will then indicate how much of the coal price must support every bank cubic metre of the waste removed. For example if the discounted average cost for dragline operation component with an interest rate of 15% is $0.77, the overall cost to maintain the NPV equal to zero at the 15% return required by the company is $0.765 per bcm. The discounted average cost indicates the minimum value that the contractor requires to receive from the mine for every bank cubic metre drilled and blasted. The calculation of discounted average cost is the ultimate solution generated by the cash flow analysis and was used as a basis for the decision making in selecting optimum dragline digging options. The cash flow analysis and the calculation of the discounted average cost for a dragline and drilling operation are illustrated in Tables 8.13 and 8.14 respectively.

Table 8.13- Cash Flow analysis of the Dragline operation using a Discounted Average Cost Method.

	0	1	2	3	4	5	6	17	18	20	21	Total
Waste Thickness, metre		45	45	45	45	45	45	45	45	45	45	
Dragline Productivity, bcm/hour		2767	3113	3459	3459	3459	3459	3459	3459	3459	3459	
Schedule Annual Operating Hours		5700	5700	5700	5700	4872	5700	4872	5700	5700	5700	
Total Material Movement, bcm		15773040	17744670	19716300	19716300	16852248	19716300	16852248	19716300	19716300	19716300	376954902
Capital Cost Components												
Cost of Dragline	60000000											60000000
Salvage Value											0	
Value for Depreciation		60000000	57000000	54000000	51000000	48000000	45000000	12000000	9000000	6000000	3000000	
Claimable Depreciation		3000000	3000000	3000000	3000000	3000000	3000000	3000000	3000000	3000000	3000000	60000000
Cost of Bucket						3000000						
Trade in Value						0					0	
Value for Depreciation		3000000	2400000	1800000	1200000	600000	3000000	2400000	1800000	1200000	600000	
Claimable Depreciation		600000	600000	600000	600000	600000	600000	600000	600000	600000	600000	12000000
Operating Cost Components												
Operating Costs/Op.Hours		1157	1180	1204	1228	1252	1277	1588	1620	1652	1685	
Total Operating Cost		6593950	6725829	6860346	6997552	6100687	7280254	7737147	9233122	9417784	9606140	155526813
Financial Calculation:												
Contract Price @ $1.2863/bcm		20288078	22824087	25360097	25360097	21676209	25360097	21676209	25360097	25360097	25360097	
Less Operating Cost		6593950	6725829	6860346	6997552	6100687	7280254	7737147	9233122	9417784	9606140	
Nett Operating Surplus		13694128	16098258	18499752	18362545	15575522	18079844	13939063	16126975	15942313	15753957	
less Depreciation Allowances		3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	
Profit for Taxation		10094128	12498258	14899752	14762545	11975522	14479844	10339063	12526975	12342313	12153957	
Less Tax Payable @ 39%		3936710	4874321	5810903	5757392	4670454	5647139	4032235	4885520	4813502	4740043	
Profit After Tax		6157418	7623938	9088848	9005152	7305068	8832705	6306828	7641455	7528811	7413914	
Plus Depreciation Allowances		3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	3600000	
Nett Cash Flow	-72000000	9757418	11223938	12688848	12605152	10905068	12432705	9906828	11241455	11128811	11013914	
Discount Factor @ 15 % ROI	1.0000	0.8696	0.7561	0.6575	0.5718	0.4972	0.4323	0.0929	0.0808	0.0703	0.0611	
Net Present Value	-72000000	8484711	8486909	8343124	7207037	5421746	5375001	920601	908367	781969	672953	0

Equivalent Operating Cost	**0.4126**	
Equivalent Capital Cost	**0.8737**	
Discounted Average Cost	**1.2863**	
	Dragline	**Bucket**
Life(year)	20	5
Capital Cost	60,000,000	3,000,000
Tax Rate	0.39	
Interest Rate	0.15	
Dragline Prime Productivity(bcm/hr)	3459	
Dragline Operating Cost	1156.83	
Operating Hours (Normal Year)	5700	
Operating Hours (Maintenance Year)	4872	

Table 8.14- Cash Flow analysis of the Drill & Blasting operation using a Discounted Average Cost Method.

	0	1	2	3	4	5	6	7	8	Total
Waste Thickness, metre		45	45	45	45	45	45	45	45	
Drill Productivity m/hour		90	90	90	90	90	90	90	90	
Schedule Annual Operating Hours		3000	3000	3000	3000	4872	3000	3000	3000	
Total Material drilled (Mbcm)		29700000	29700000	29700000	29700000	48232800	29700000	29700000	29700000	256132800
Capital Cost Components										
Cost of Drill	**6000000**									
Trade in Value										
Value for Depreciation		6000000	5250000	4500000	3750000	3000000	2250000	1500000	750000	
Claimable Depreciation		750000	750000	750000	750000	750000	750000	750000	750000	**6000000**
Operating Cost Components										
Operating Costs/Op.Hours		2731.577	2786.20854	2841.93	2898.77	2956.75	3015.88	3076.20	3137.72	
Total Operating Cost		8194731	8358628	8525798	8696314	14405270	9047645	9228598	9413170	**75870153**
Financial Calculation:										
Contract Price @ $0.35/bcm		10282018	10282018	10282018	10282018	16697997	10282018	10282018	10282018	
Less Operating Cost		8194731	8358626	8525798	8696314	14405270	9047645	9228598	9413170	
Nett Operating Surplus		2087287	1923392	1756220	1585704	2292727	1234373	1053420	868848	
less Depreciation Allowances		750000	750000	750000	750000	750000	750000	750000	750000	
Profit for Taxation		1337287	1173392	1006220	835704	1542727	484373	303420	118848	
Less Tax Payable @ 39%		521542	457623	392428	325924	601663	188905	118334	46351	
Profit After Tax		815745	715769	613794	509779	941063	295467	185086	72497	
Plus Depreciation Allowances		750000	750000	750000	750000	750000	750000	750000	750000	
Nett Cash Flow	-6000000	1565745	1465769	1363794	1259779	1691063	1045467	935088	822497	
Discount Factor @ 15 % ROI	1	0.870	0.756	0.658	0.572	0.497	0.432	0.376	0.327	
Net Present Value	-6000000	1361517	1108332	896717	720283	840757	451984	351533	268876	**0**

Equivalent Operating Cost	**0.296**
Equivalent Capital Cost	**0.050**
Discounted Average Cost	**0.346**
	Drill
Machine Life (year)	8
Capital Cost	2000000
No of Machine Required	3
Total Capital Cost	6000000
Depreciation	750000
Tax Rate	0.39
Interest Rate	0.15
Drilling Productivity (m/hr)	90
Drill & Blast Operating Cost	2731.58
Operating Hours (Normal Year)	3000

8.5 SUMMARY

In this chapter the process of analysing the results from the simulation model has been described in detail. The analysing techniques described in this chapter were productivity (both deterministic and stochastic) and cost analysis.

From a combination of basic formulae, data from the CADSIM dragline simulator part and time study data, productivity can be estimated. Productivity calculations are usually made on the basis of inadequate data. Many input parameters are actually random variables, but their point estimates are used. A good decision may result using most likely values, but a better decision is possible using a stochastic simulation. Stochastic risk modelling provides more information than does a direct calculation. Also, such modelling allows a more realistic assessment of the potential results to be expected for different variables.

Using Monte Carlo simulation, the procedure employed here was to sample the values of the random variables from their respective distributions and recompute the target function using the sampled value. By using an adequate number of replications of this procedure an estimation of distribution of the outcomes becomes possible. The computed values of the outcomes from the simulation can be used to plot the frequency distribution and to estimate the mean and the standard deviation values. The stochastic productivity estimation showed that annual dragline productivity was sensitive to the cycle time components and might vary within a significant range due to the variability of the random input parameters.

Following completion of the productivity analysis for a coal property a financial evaluation of the simulated digging methods should also be done. In order to establish a cost analysis for a given project a number of parameters must be defined. Productivity analysis of the dragline operation has provided some of the basic information required for a cost analysis study. The process of cost estimation is generally conducted for comparison purposes on the basis of either cost per hour, per bcm, or per tonne of coal exposed. In this process the cost associated with each scenario needs to be compared with that of other alternatives to obtain the "best" solution to a mine planning problem.

A Discounted Average Cost (DAC) method was used as part of the financial analysis stage of the thesis. This method is in fact a version of the conventional Discounted Cash Flow (DCF). The DAC method is more suitable for decision making processes as it does not include the revenues from the coal being won which tend to bias the decision. The objective is to determine what costs would apply using a given method, so that the method can be compared with other alternative methods.

CHAPTER NINE

VALIDATION OF THE CADSIM MODEL

9.1 INTRODUCTION

A simulation model must be validated before it can be used for analysing various planning and design proposals. The validity of a simulation model relies on the ability of the model to produce results comparable with data from real operations. In this chapter the CADSIM model validation process is presented using a productivity analysis for a multi seam dragline operation.

9.2 MODEL VALIDATION

The process of the validation for CADSIM outputs was performed for both the dragline simulator and mine productivity calculations. The validation of the dragline simulator consisted of testing the programming logic and its ability to mimic dragline operations. Based on the thesis objectives the following three procedures were used for the model validation:

1. The logic of the volumetric calculations of the CADSIM model was tested using a simple block of waste and comparing the results with the results obtained from manual calculations and hand drawings.

2. The CADSIM model was used to simulate a standard extended bench method for a hypothetical section. The results were compared with the outputs from a commercial computer package DAAPA3.
3. The abilities of the CADSIM model to estimate swing angles, hoist distances and volumetric calculations were tested using a real multi seam operation. The results from the simulation were compared with actual data from a dragline monitoring system.

9.2.1 Manual Technique

Manual validation involved the development of various manual calculations and 2D range diagrams for both hypothetical and real operations. The first simulation runs were comparatively simple in concept and design. Both trigonometric and planimetric calculations were used to verify CADSIM outputs for volumetric calculations. This enabled the dragline simulation model's logics and programming aspects to be checked. Manually generated plans and 3D drawings were developed using established basic formulae for calculations of swing angle and hoist distance for various dragline positions while removing a block of overburden. The outputs from the CADSIM model for swing angle, walking patterns and hoist distance information were then compared with the manual calculations. Comparison of the results showed that the model could accurately perform the required calculations for a dragline operation.

9.2.2 Comparison with DAAPA3

A commercial computer package DAAPA3 was employed to develop hypothetical dragline operations. DAAPA3 is a product of Runge Mining Pty Ltd which uses a trigonometric approach to calculate volumes and to estimate productivity of the dragline operations for a mining block based on 2D range diagrams. The results from DAAPA3 were compared with outputs from the CADSIM model using the same set of parameters. The dragline specifications and strip parameters used for the simple test case are presented in Tables 9.1 and 9.2.

Table 9.1- Dragline specifications used in the testing case.

Terminology	Dimension
Operating radius (m)	87.5
Bucket capacity (m^3)	45.0
Maximum dump height (m)	45.0
Maximum dig depth (m)	50.0
Tub clearance radius (m)	10.0
Shoe clearance radius (m)	13.0
Tail clearance radius (m)	25.0

Table 9.2- Pit geometry and productivity parameters.

Parameter	Value
Waste thickness (m)	35.0
Coal seam thickness (m)	7.0
Angle of repose (deg)	37.0
Swell factor	1.3
Undercut lowwall angle (deg)	45.0
Spoil berm width (m)	0.0
Waste highwall angle (deg)	75.0
Coal seam angle (deg)	75.0
Key cut lowwall angle (deg)	65.0
Key cut width (m)	7.0
Coal dip (deg)	2.0
Coal edge trench width (m)	0.0
Dig block length (m)	30.0
Bucket filling time (sec)	15.0
Bucket spot time (sec)	3.0
Dump time (sec)	3.0
Walking speed (m/hr)	100.0
Utilisation (%)	86.6

Figures 9.1 and 9.2 are graphic outputs from the CADSIM model and DAAPA3 for the comparative case study while Table 9.3 shows a summary of the results obtained. The comparison shows good agreement (less than 5%) between the outputs from DAAPA3 and the results from the CADSIM model. It was also found from the comparison that using a CAD based approach for volumetric and swing angle calculations makes the model much more flexible in handling various situations which may be met during the simulation of a complex dragline operation.

Table 9.3- Comparison of the results from DAAPA3 and the CADSIM model.

Component	Parameter	DAAPA3	CADSIM	Variation*
Key Cut				
	Volume (bcm/m)	734.5	724.3	1.4
	Av. swing angle (deg)	77.0	78.5	-1.9
	Swing time (sec)	17.8	18.7	-5.1
	Cycle time (sec)	56.7	58.4	-3.0
	No. of cycles	489.7	482.0	1.6
	Walking time (min)	2406.0	2318.0	3.7
	Total time (hr)	9.6	9.7	-0.8
	Productivity (bcm/hr)	2305.0	2243.0	2.7
Main Block				
	Volume (bcm/m)	1468.6	1430.1	2.6
	Av. swing angle (deg)	92.0	85.5	7.1
	Swing time (sec)	20.8	19.2	7.7
	Cycle time (sec)	62.7	59.5	5.1
	No. of cycles	979.1	953.2	2.6
	Walking time (sec)	2502.0	2420.0	3.3
	Total time (hr)	20.2	19.2	5.0
	Productivity (bcm/hr)	2177.7	2234.5	-2.6
Bridge				
	Volume (bcm/m)	529.5	541.1	-2.2
	Av. swing angle (deg)	80.4	76.0	5.5
	Swing time (sec)	18.4	18.3	0.5
	Cycle time (sec)	57.8	57.6	0.3
	No. of cycles	353.0	360.7	-2.2
	Walking time (sec)	354.0	360.0	-1.7
	Total time (hr)	6.6	6.9	-3.8
	Productivity (bcm/hr)	2393.4	2404.7	-0.5
Totals				
	Volume (bcm/m)	2732.6	2695.5	1.4
	Av. swing angle (deg)	85.7	81.7	4.7
	Swing time (sec)	19.5	18.9	3.3
	Cycle time (sec)	60.1	58.8	2.2
	No. of cycles	1821.8	1795.9	1.4
	Walking time (sec)	5262.0	5098.0	3.1
	Total time (hr)	36.4	35.7	1.8
	Rehandle (%)	24.0%	25.1%	-4.5
	Prime Productivity (bcm/hr)	1814.4	1808.9	0.3
	Total Productivity (bcm/hr)	2250.4	2263.2	-0.6

$$* \ Variation = \frac{DAAPA3\ Outputs\ -\ CADSIM\ results}{DAAPA3\ Outputs} \times 100$$

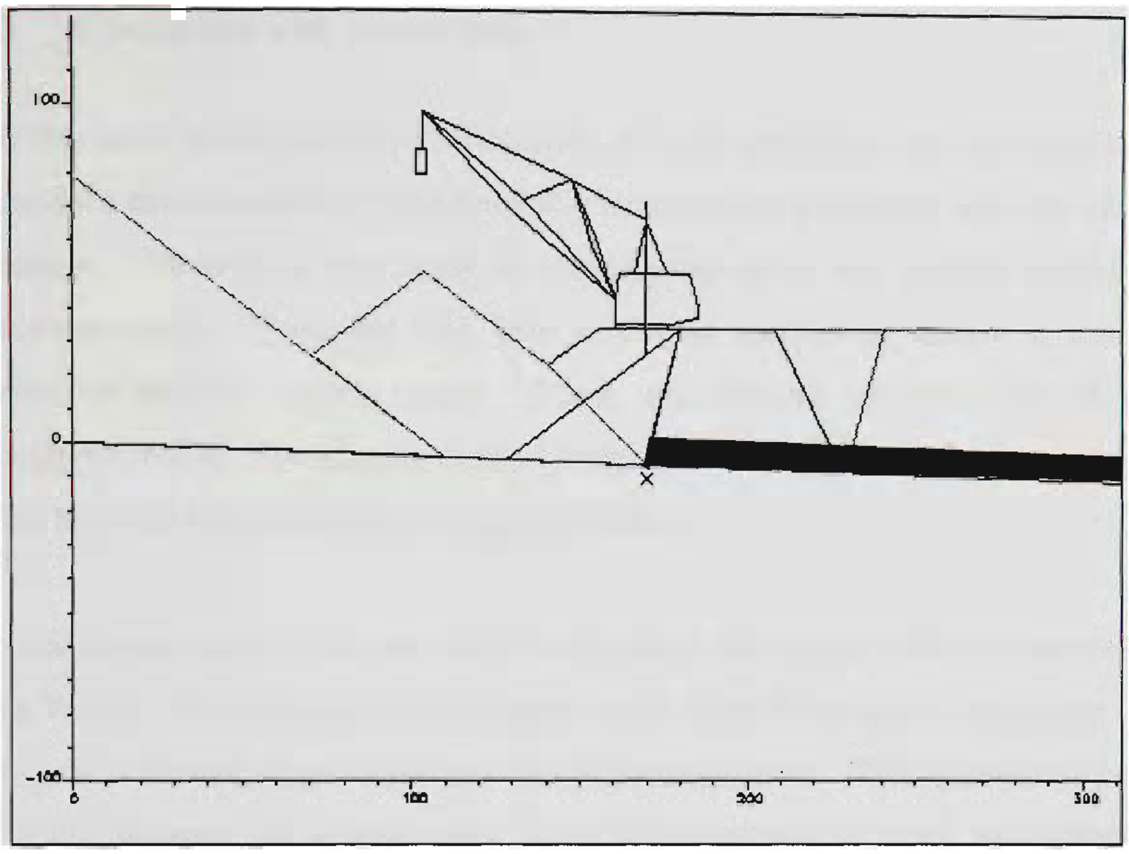


Figure 9.1- Sample outputs from the CADSIM model for the simple case study.

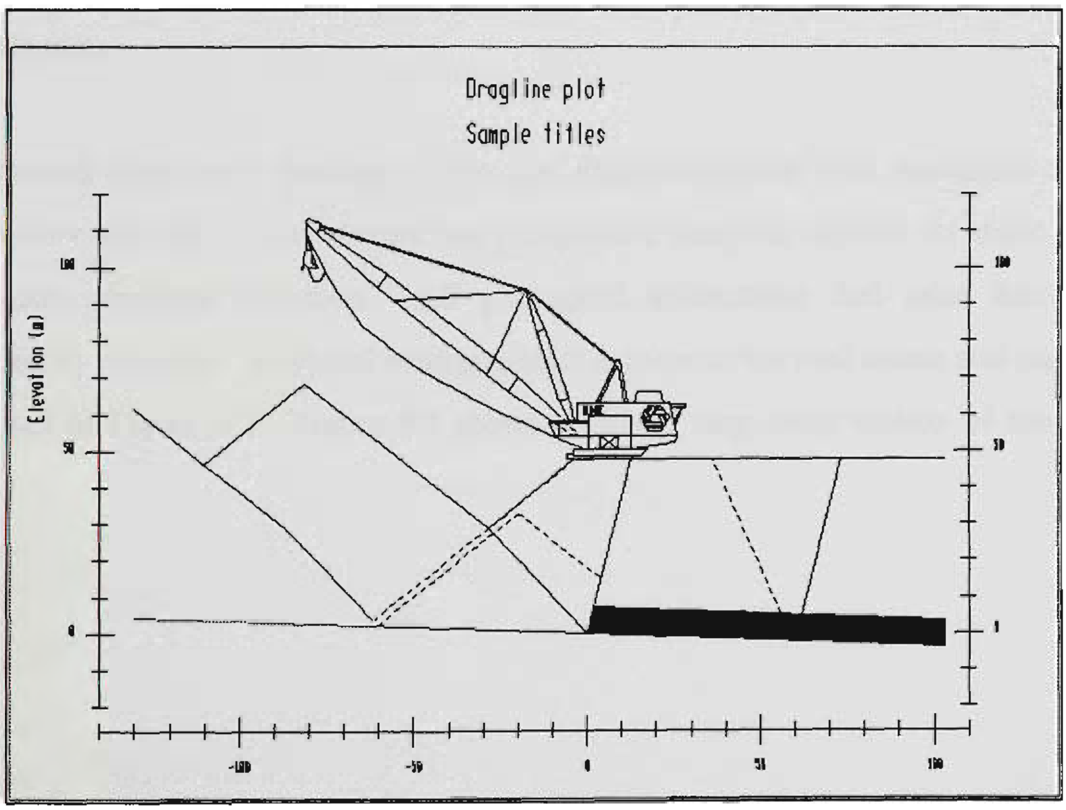


Figure 9.2- Sample outputs from DAAPA3 for the simple case study.

9.2.3 Comparison with Actual Data

With the use of simple and hypothetical data, it is not possible to test and demonstrate the model's entire capabilities and flexibility in reproducing complex and real dragline operations. Therefore a case study of a multi pass mine was used to validate the simulation results. Using the data from a dragline monitoring system it was also possible to validate various aspects of both the dragline simulator and its mine productivity results. The digging method analysed was a conventional single highwall, double low wall side pass dragline stripping method.

The coal deposit used in the case study is situated in the Hunter Valley in eastern New South Wales. The topography in the region varies from flat to gently undulating. The study area is located on the north-east side of the mine lease. Mining has been carried out at this property for several years, using both conventional truck and shovel and dragline methods of stripping. Currently the mine produces approximately 2.7 million tonnes of both metallurgical and thermal coal per annum for the export market. The mine is a complex multi seam operation with the total coal production from several separate pits.

The variable inter-seam geology of the coal deposit together with numerous splitting and coalescence of the coal seams has produced a complex deposit for multi-pit and multi-seam dragline operation. All geological information and mine limits were provided by the mine. A typical stratigraphical column of the coal seams and partings is presented in Figure 9.3. Figure 9.4 shows a typical long cross section of the current strip.

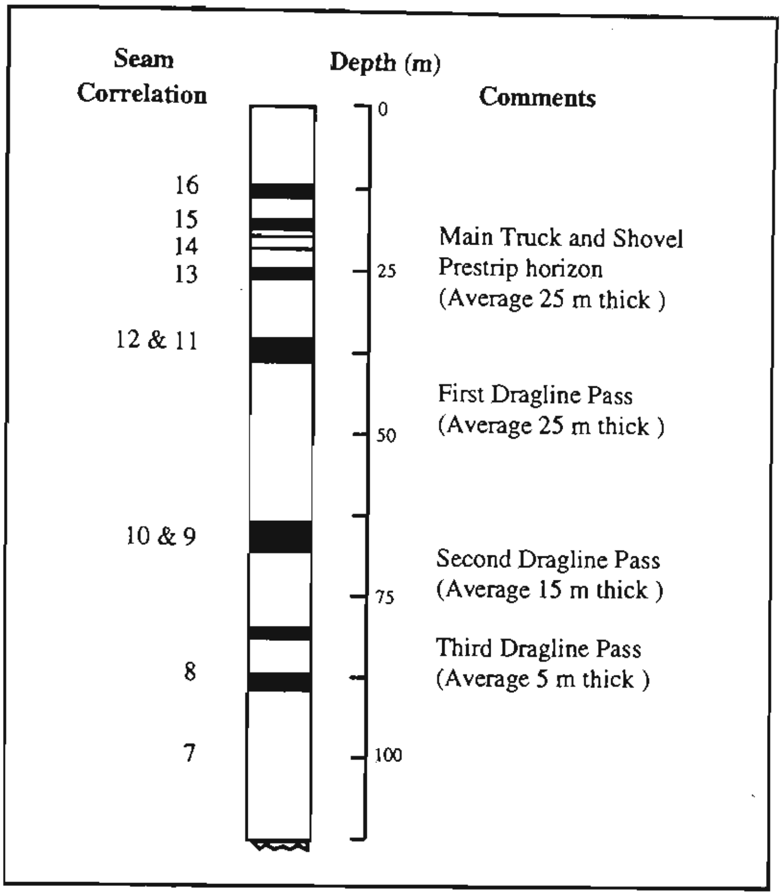
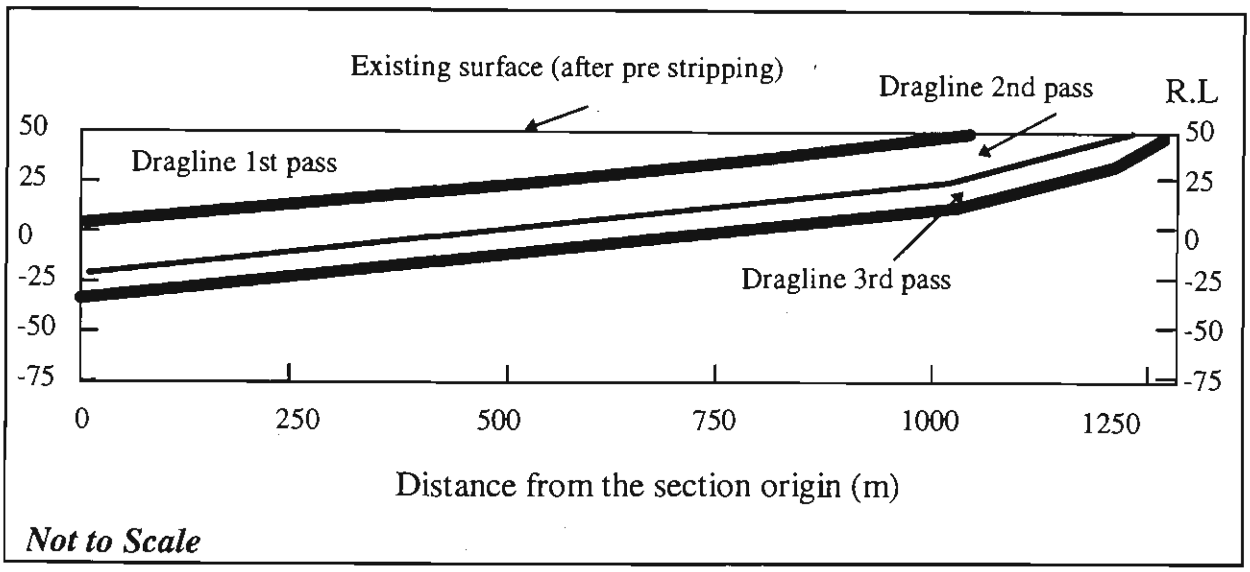


Figure 9.3- A typical stratigraphic sequence of the case study pit.



9.4- Schematic long cross section of the dragline passes.

9.2.3.1 Dragline Digging Method

The mine operates a standard BE 1370W walking dragline equipped with a 48 cubic metre bucket. Table 9.4 summarises the critical parameters of the dragline.

Table 9.4- The dimensional parameters of the BE 1370-W walking dragline

Terminology	Dimension (m)
Operating radius	85.0
Boom foot radius	12.2
Clearance height	3.8
Dumping clearance	21.0
Dumping height	48.0
Boom point height	69.0
Digging depth	57.9
Tub diameter	21.33
Boom angle	41*

** In degree*

The geological data required for the simulation were provided by the mine in form of a gridded seam model. The grids from the geological model were used to develop the initial strings in the cross sections. A width of 30 metres (equal to the length of each mining block) was selected between the sections. This results in 45 parallel cross sections covering a 1300m strip. Table 9.5 shows the strip and material parameters used for the simulation.

Table 9.5- Strip and material parameters used for simulation.

Parameter	Value
Highwall angle (deg)	70.0
Key cut angle (spoil side) (deg)	63.0
Spoil repose angle (deg)	37.0
Strip width (m)	55.0
Spoil swell factor	1.3
Coal trench width (m)	5.0
Prestrip offset (m)	25.0
Prestrip highwall angle (deg)	63.0

The current dragline operation at the study mine involves three dragline passes. Figure 9.5 shows a general view of a typical single highwall, double low wall dragline operation, generated from the CADSIM dragline simulator.

First Pass: This is a standard underhand technique with highwall key and main cut components. The overburden thickness ranges between 11 to 37m and the coal thickness varies from 1.9 to 2.2m. The spoil is directly dumped into the previous strip void so there is no rehandle for bridging. However, there is a ten percent rehandle mostly due to the coal haulage ramps.

Second Pass: The dragline technique in this pass is a lowwall pass involving chop operations from an in-pit bench. In this pass the dragline operation is tight spoiling and dumping to its maximum height. The requirement to dump behind the machine greatly increases the cycle time due to the longer swing angle. The interburden varies between 7 to 17m in thickness, and the coal seam thickness ranges from 1.0 to 1.5m.

Third Pass: The third pass is essentially the same as the second pass. However, due to the shorter swing angles employed, the cycle times are reduced compared with those of the second pass. In this pass the interburden varies in thickness from 2 to 5m, and overlies a 0.5 to 1.5m thick coal seam. The coal seam dip angles over the area vary from 4 to 6 degrees.

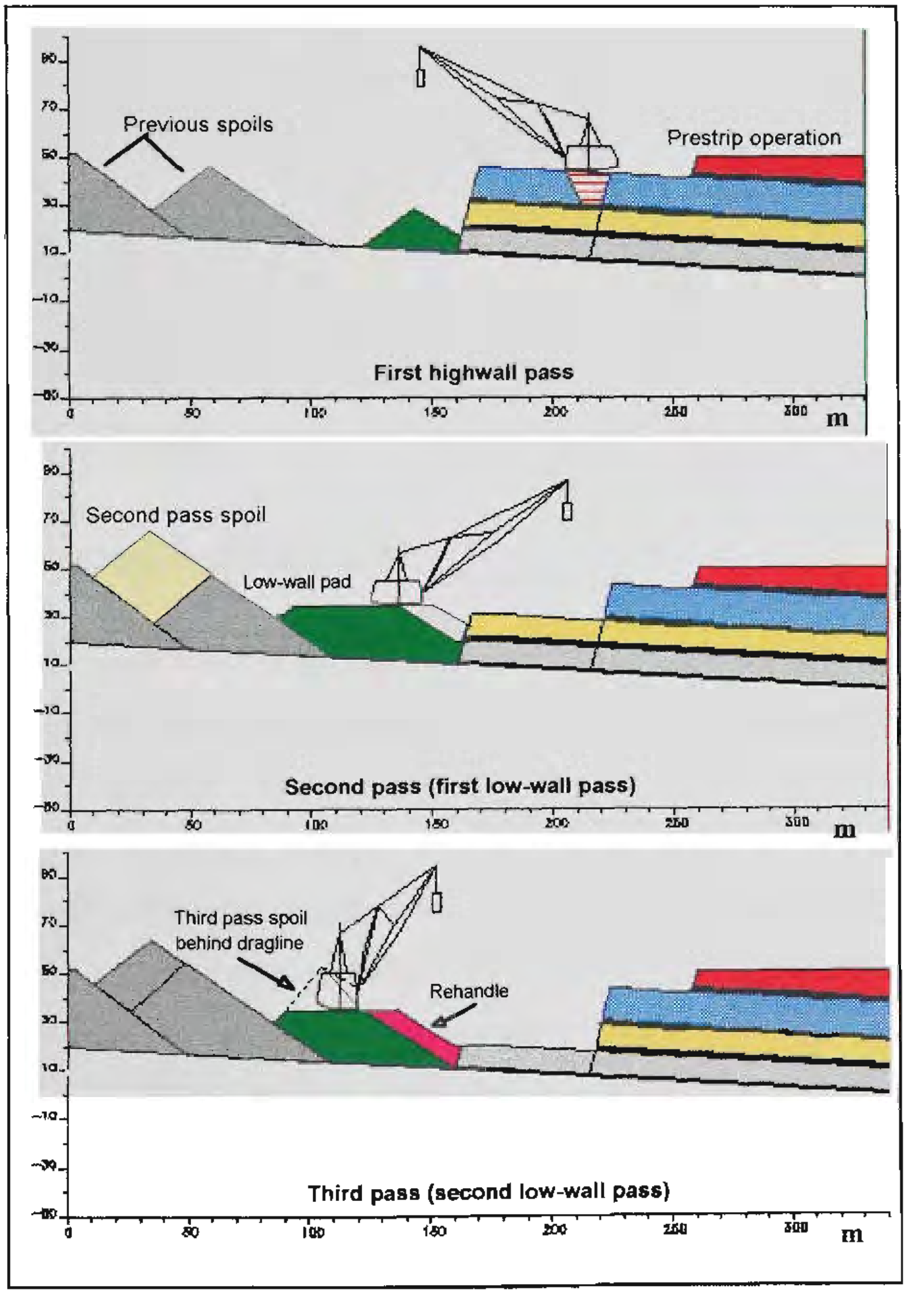


Figure 9.5- Three seam operation, single highwall and double low wall method (Output from the CADSIM model).

Figure 9.6 shows a view of the current operations at the mine with the dragline removing the second interburden from the lowwall side.

Figure 9.6- Dragline removes the second interburden from the lowwall side.

9.2.3.2 Comparison of the Results

The CADSIM model was run for the same pit configuration being used at the mine at the time when the data were captured. The pit was approximately 1.5km long, extending south to north. Since the dragline performance is effected by its mode of operation, it is necessary first to separate the actual mine data for both the highwall side and lowwall side. The mine results from the lowwall side consists of information for both the second and third pass. As the CADSIM model uses a deterministic approach, only the mean value of the parameters recorded by the monitoring data were used for comparison.

The dragline monitoring system records dig rate as tonnes per cycle. The tonnage was converted to volume using an average rock density of 2.2. The volume per cycle was then multiplied by the number of cycles per hour to estimate the actual dig rate recorded by the dragline monitor. A comparison of CADSIM results and data from the monitoring system for both the highwall and the lowwall passes is given in Table 9.6. The comparison results are illustrated in Figures 9.7 and 9.8 for the highwall and the lowwall stripping, respectively.

Table 9.6- Comparison of the monitoring data and the dragline simulation results.

Performance Parameter	Highwall Side			Lowwall Side		
	Monitoring System	CADSIM Model	Variation*	Monitoring System	CADSIM Model	Variation*
Fill time (sec)	14.6	18.0	25%	19.6	18.0	-8%
Swing time (sec)	18.5	15.1	-18%	22.9	23.9	4%
Swing angle (degrees)	73.2	55.6	-24%	120.1	126.1	5%
Dump time (sec)	8.2	6.0	-27%	6.2	6.0	-9%
Return time (sec)	18.5	15.9	-14%	23.0	24.8	8%
Cycle time (sec)	57.7	54.9	5%	70.3	73.6	5%
Hoist distance (m)	15.8	17.0	8%	35.8	36.5	2%
Cycle/dig hr	52	57	9%	43	41	-6%
Cycle/Scheduled hr	44	43	-2%	38	37	-3%
Cycle/day	1066	1020	-4%	901	880	-2%
Dig hours per day	17.6	17.3	-2%	17.5	17.3	-1%
Dig rate (bcm/hr)	1995	1890	-5%	1515	1434	-5%
Efficiency (%)**	73.3	72.6	-1%	72.8	72.6	1%

$$* \ Variation = \frac{CADSIM\ Results\ -\ Data\ from\ Monitor}{Data\ from\ Monitor} x100$$

$$** \ \mathit{Efficiency} = \frac{Dig\quad hours}{Scheduled\quad hours} \times 100$$

Comparison of the data from the monitoring system and the CADSIM model outputs shows that the CADSIM model is able to distinguish the mode of operation and predict most of the operational parameters within an acceptable range. The variation in the estimation of the critical values such as dig rate and number of cycles per day is around 5%. The highest discrepancy (14 to 27% variation) was in the estimation of cycle time components at the highwall side. A review of the logic used in the dragline operation simulation part of the model revealed that the discrepancy could be a result of incorrect dragline positions. Various segments of the dragline simulator CADSIM and productivity calculation parts of the model were modified using the monitoring data and practical input from the mine personnel. Having the model modified and calibrated, productivity was recalculated and different pit designs were evaluated to optimise the pit configurations. The result of the pit optimisation for this case study is presented in Appendix E.

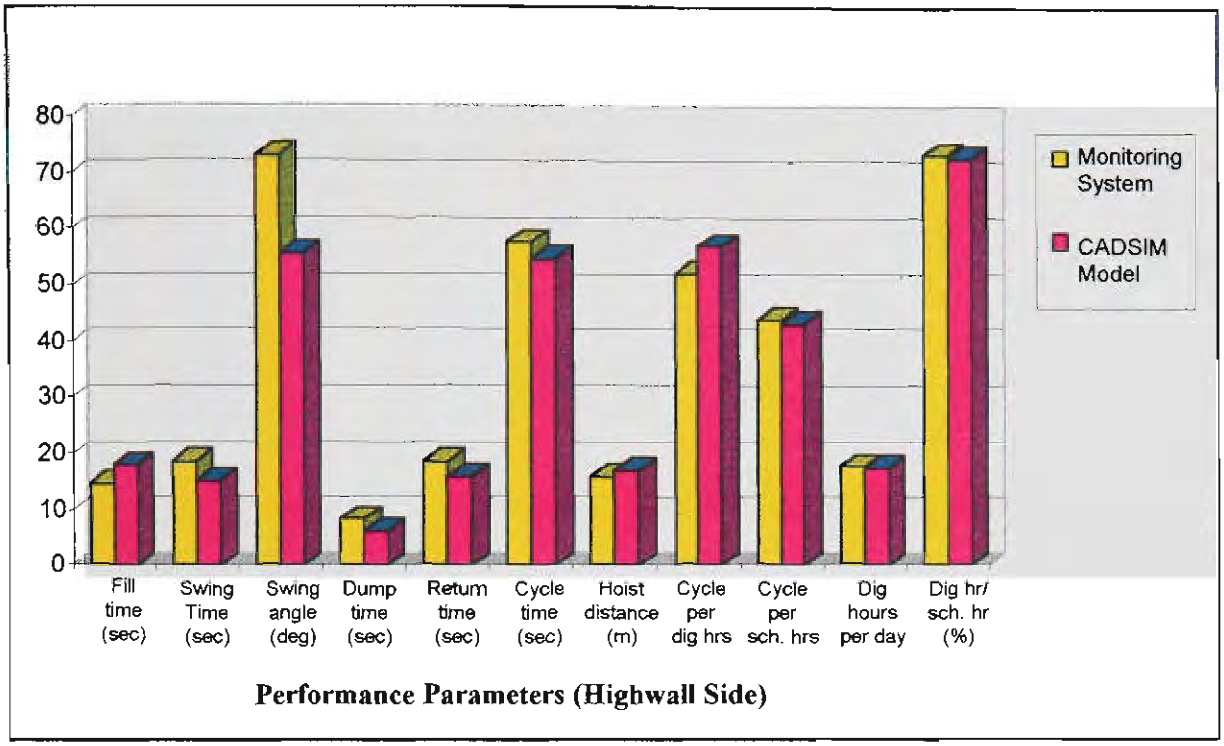


Figure 9.7- Comparison of the results from the dragline monitoring system and the CADSIM model for the highwall side stripping.

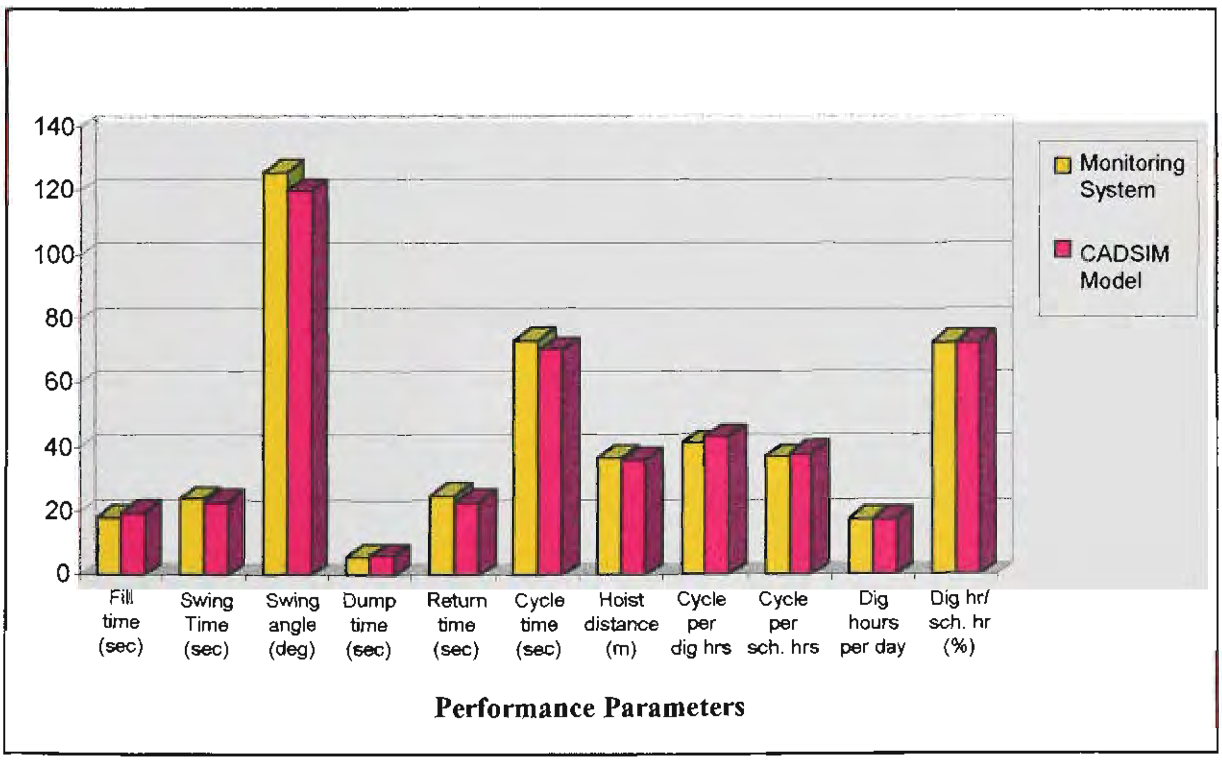


Figure 9.8- Comparison of the results from the dragline monitoring system and the CADSIM model for the lowwall side stripping.

9.3 SUMMARY

A comparison between the actual mine data from a dragline monitoring system and CADSIM results from both the dragline simulator and productivity parts of the model showed that the CADSIM model developed for this thesis could be used to indicate a suitable mode of operation and predict the most important operational parameters accurately.

CHAPTER TEN

APPLICATIONS OF THE CADSIM MODEL

10.1 INTRODUCTION

The CADSIM model was used to evaluate the operational options for two existing large strip mines one in Hunter Valley of New South Wales and the other in the Bowen Basin, Queensland. The first case study shows that the system developed in this thesis can be used for selecting an optimum digging method for a new dragline operation. The second case study demonstrates the application of the CADSIM model to optimise pit geometry and strip layout while taking into consideration the effect of pit orientation and the position of the coal access ramps.

10.2 CASE STUDY 1

The object of this case study was to select a suitable digging method for a new dragline with a given geology. The three digging methods considered to be applicable to the mine were:

1. A standard highwall key cut method utilising an extended bench. The extended bench must have sufficient length to allow the lowwall edge of the coal to be cleaned by a trench of one dragline bucket width.
2. A lowwall dragline method utilising a highwall chop and a pull back operation.
3. A dragline method utilising an extended key cut on the first pass from the highwall, and a lowwall pull back operation on the second (and third) pass from an in-pit bench.

10.2.1 Geology of the Deposit

The coal seams to be extracted within the lease include the Whybrow, Wambo, Glen Munro, Woodland Hill, Whynot and Blackfield. The seams dip less than 5 degrees in the central and western parts of the lease. Figure 10.1 shows a typical cross section of the area and a typical stratigraphical column of the coal seams and partings is given in Figure 10.2. The original geological data in the form of a gridded seam model was provided by the mine. The geological model was created from a massive exploration program and contains a set of 2D gridded surfaces representing roof and floor of the coal seams as well as the topographical surface.

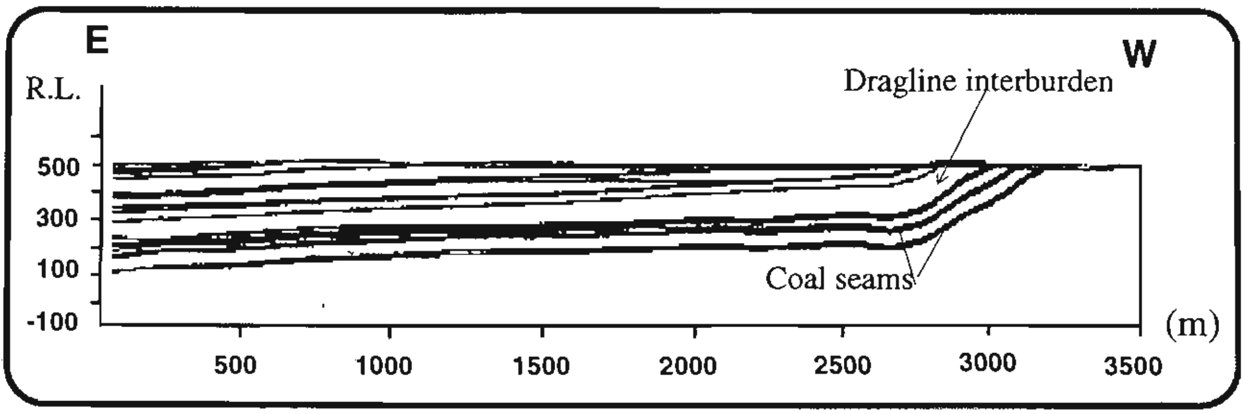


Figure 10.1- A typical long cross-section of the deposit.

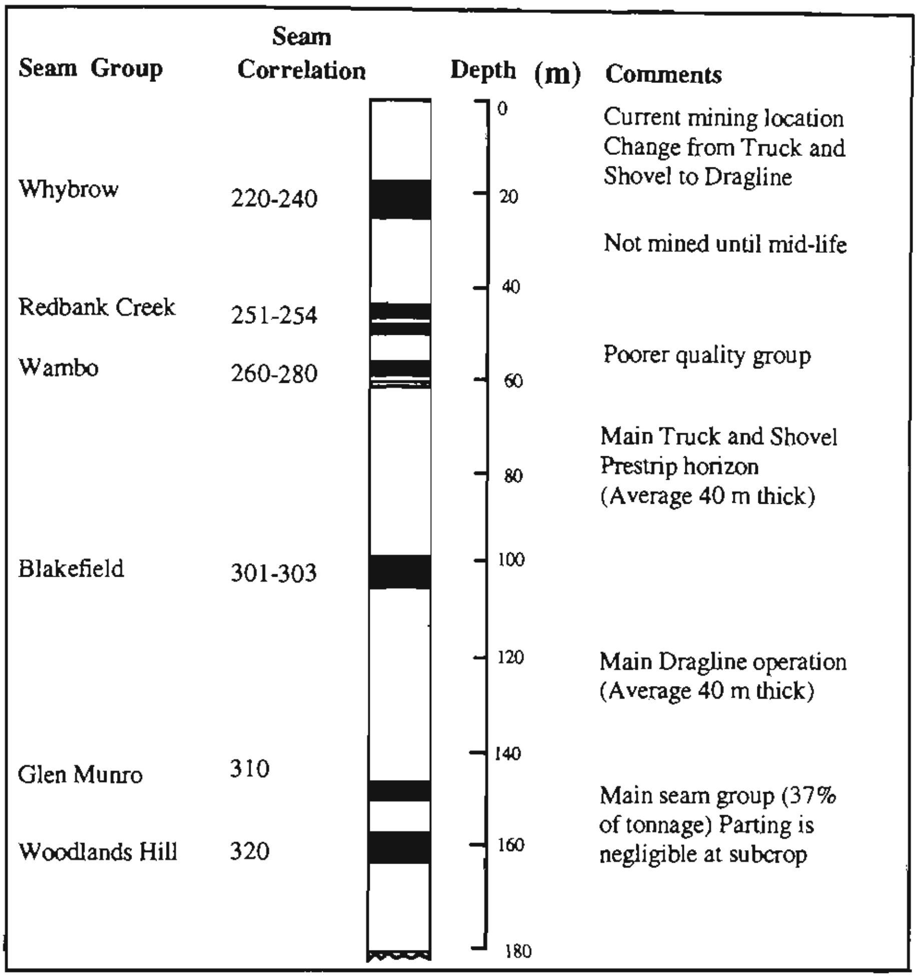


Figure 10.2- A typical stratigraphic sequence of the first case study.

Having created the section mounts, the geological sections were then generated by accessing the grids from the geological model. The dragline simulation was carried out on 30 strips divided into two distinct areas of north and south. With an average length of 1200m for each strip and a 30m interval between sections, some 40 parallel sections perpendicular to the strike of the strips were created to cover the mining area in each part. The volumetric calculations were based on a 10×10 metre grid for the surface of the coal seam and the topography. Table 10.1 presents a list of grids and definitions of the layers used for the dragline simulation.

The DSLX's option "Generate Geology" was used to intersect the section mounts and grids. The strings resulting from this intersection were written into the ASCII files to be used in the dragline simulator CADSIM.

Table 10.1- The selected grids and layer definition for simulation.

Grid Surface		Layer Defined
Code	**Description**	
TOPS	Original Topography	
		Truck and loader operation
260SR	Wambo Seam Roof	
		Wambo seam group
260SF	Wambo Seam Floor	
		Main truck and shovel prestrip horizon
301SR	Upper Blackfield Roof	
		Blackfield seam group
303SF	Lower Blackfield Floor	
		Main dragline operating horizon
312SR*	Split of Glen M. Roof	
		A split of Glen Munro seam group*
312SF*	Split of Glen M. Floor	
		Dragline operation partings
310SR	Glen Munro Roof	
		Glen Munro seam group
310SF	Glen Munro Floor	
		Dragline operation partings
320SR	Woolands Hill Roof	
		Woodlands Hill seam group
320SF	Woolands Hill Floor	

* This coal seam is to be extracted only in the Southern area.

10.2.2 Surface Mining Layout

All of the mining methods to be considered in this case study have the following common features:

- A central ramp access from the surface to the floor of the coal deposit will be used as access to the pit for coal extraction by truck and loader. Because of the depth to the floor of the seam, ramp volumes will be very large (greater than 180,000 bcm), even at the shallow points of access.
- Mining will advance from the central ramp to the end of pit in both northern and southern areas. The average length of the strips in each area (from central ramp to the end of pit) ranges from 1200 to 1300m. The length of 1300 is long enough to avoid excessive dragline walking time delays and spoil room loss near the ramps.
- As the parting between the Woodlands Hill and Glen Munro coal seams is negligible in the first 15 strips, especially at the northern area, the study

considered both coal seams as a single seam. Consequently, the proposed methods include only a single seam operation for the dragline.

Figure 10.3 is a schematic view of the dragline strips layout. The mining area is divided by a central lowwall access ramp to create northern and southern areas. Each area consists of a 110 metre wide box-cut. Mining will commence from the seam outcrops in the eastern side and proceed in the western direction. The selected layout allows a relatively shallow but long box-cut. The strip lengths at the pit vary rapidly in the first three strips of Area 1 (the southern part) due to the coal structures in this area of the deposit. The nature of the mine boundary impacts on the length of the strips in Area 2 (the northern area) and as mining advances the strip lengths increase slightly.

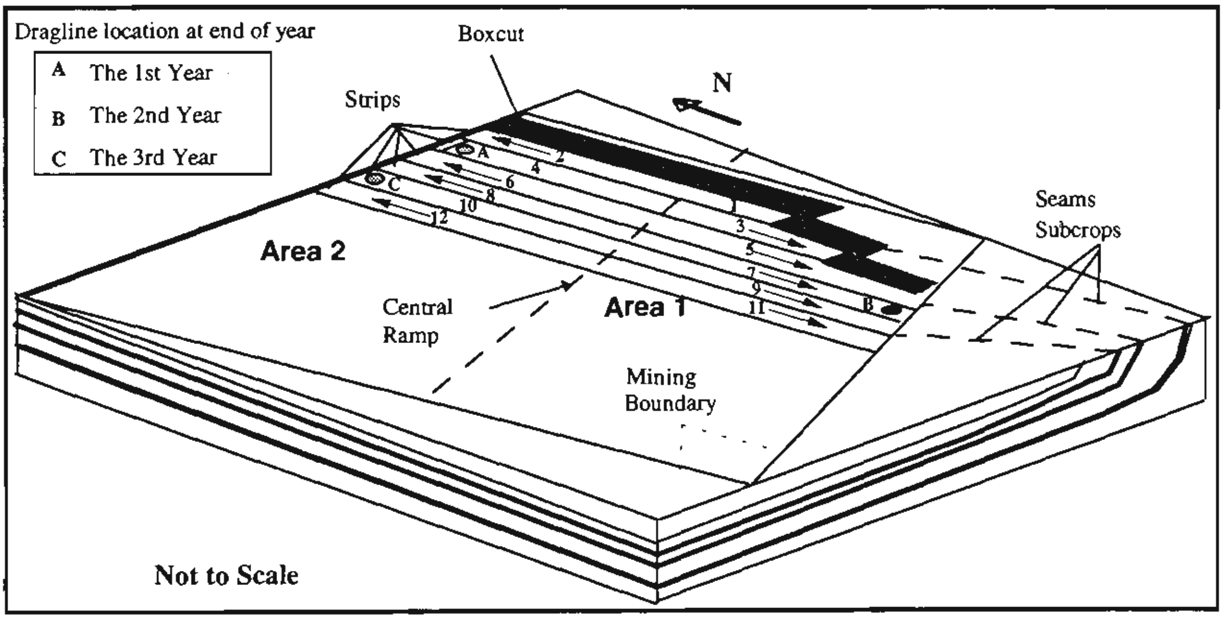


Figure 10.3- A schematic block diagram of strips in the mine pit

10.2.3 Dragline Digging Methods

The mine will operate a standard P&H walking dragline model 9020-S equipped with a 90.2 cubic metre bucket. Figure 10.4 and Table 10.2 summarise the critical parameters of the P&H walking dragline.

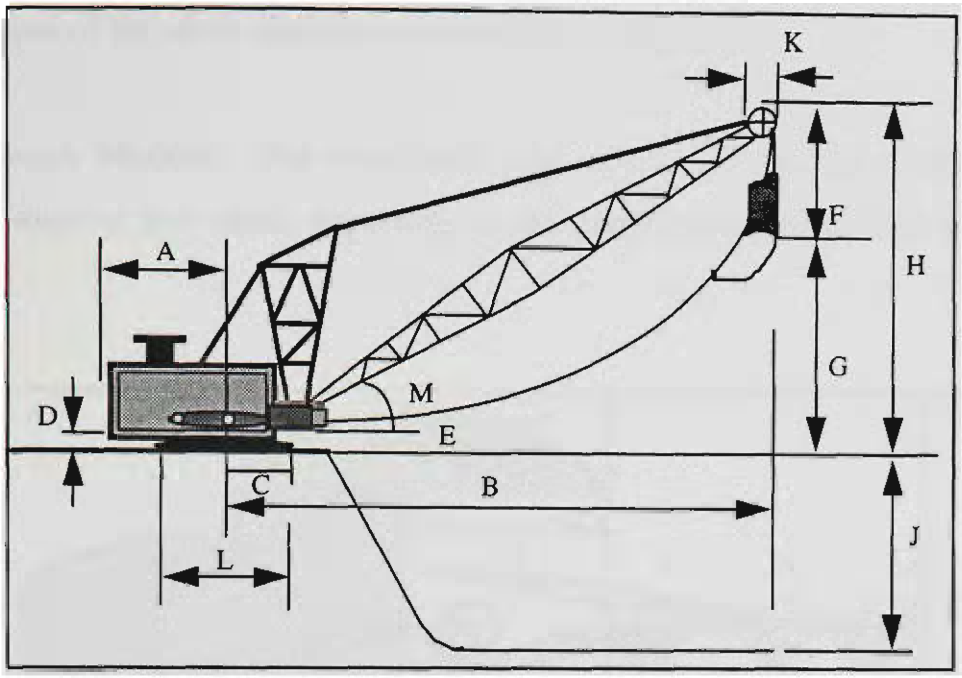


Figure 10.4- Dimensional diagram of a walking dragline.

Table 10.2-Dimension terminology of a P&H 9020-S walking dragline.

Terminology	Dimension	Letter Code (refer to Figure 10.4)
Clearance radius (m)	24.4	A
Operating radius (m)	87.5	B
Boom foot radius (m)	12.2	C
Clearance height (m)	3.8	D
Boom foot height (m)	4.9	E
Dumping clearance (m)	21.0	F
Dumping height (m)	48.0	G
Boom point height (m)	69.0	H
Digging depth (m)	57.9	J
Point sheave pitch diameter (m)	3.4	K
Tub diameter (m)	21.3	L
Boom angle (deg)	41	M

The three digging methods considered in this case study are:

1. Standard Extended Bench,
2. Lowwall In-Pit Bench, and
3. Extended Key Cut.

These methods are the common digging methods used to uncover a single coal seam in Australian strip mines.

The descriptions of the above digging procedures are given below.

Extended Bench Method: The volumetric calculations for the first method were divided into three or four cases, depending on the dragline position and pit geometry (Figure 10.5).

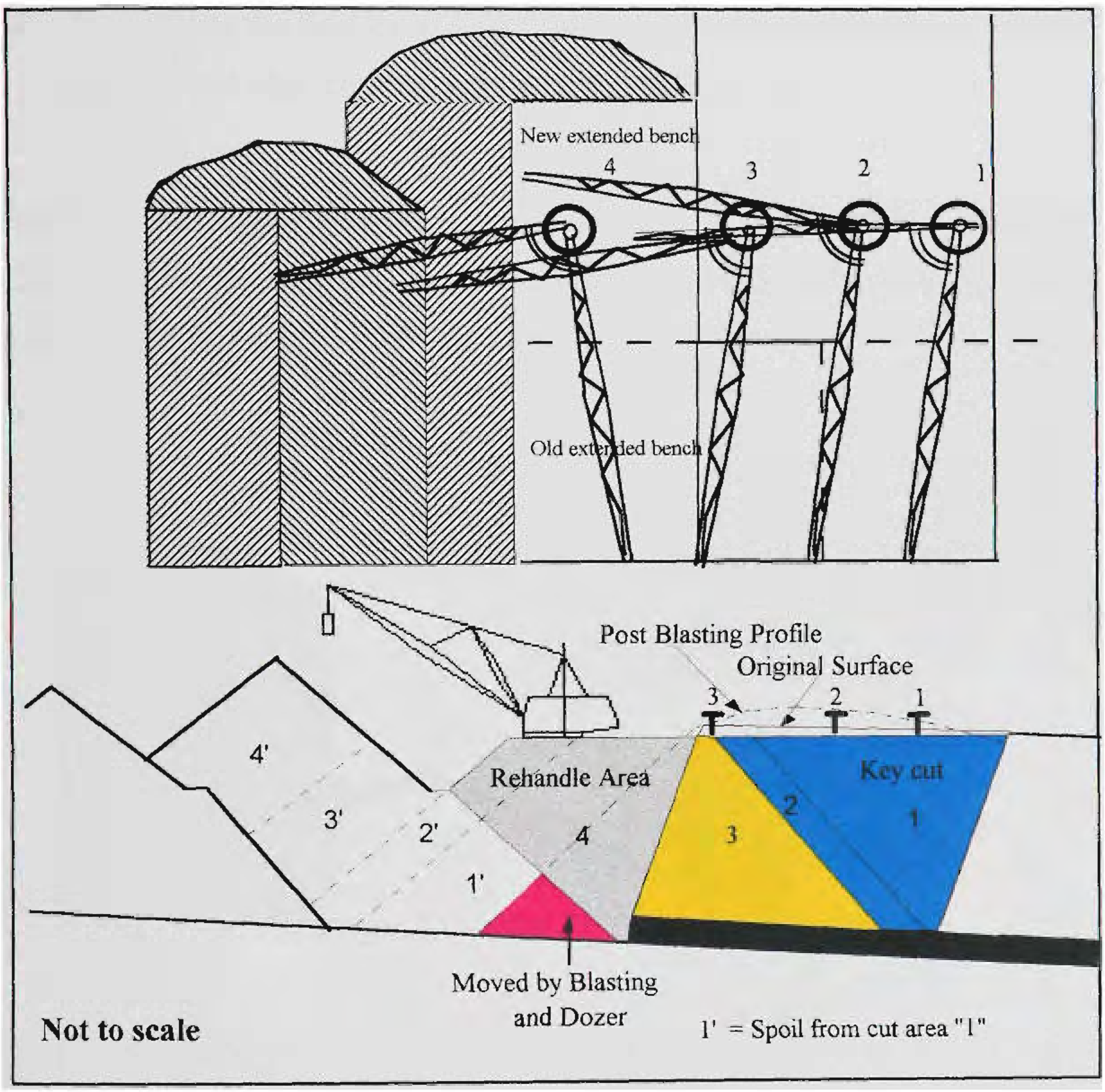


Figure 10.5- A schematic view of the dragline positions during the excavation of a block in an Extended Bench method.

To complete the excavation of a block of the overburden, the dragline uses the following four positions:

1. To excavate a key cut of one and a half bucket widths on the top of the coal. The spoil from the key cut is used to form the extended bench for the next block.

2. To continue to extend the bench using material from widening of the key cut until the bridge is completed. This position may be excluded if material from the key cut is sufficient to construct the extended bench.
3. To excavate the remaining material from the main cut and cast it to the spoil area. This position is dictated by the operating radius of the dragline and the available spoil room.
4. To complete the final excavation of the previous extended bench and clean the lowwall coal edge, creating a 5m wide trench.

Lowwall In-Pit Bench Method: In this method the volume of the block to be excavated is divided into different sub-volumes depending on the dragline positions. The division of the pit cross-section is shown in Figure 10.6.

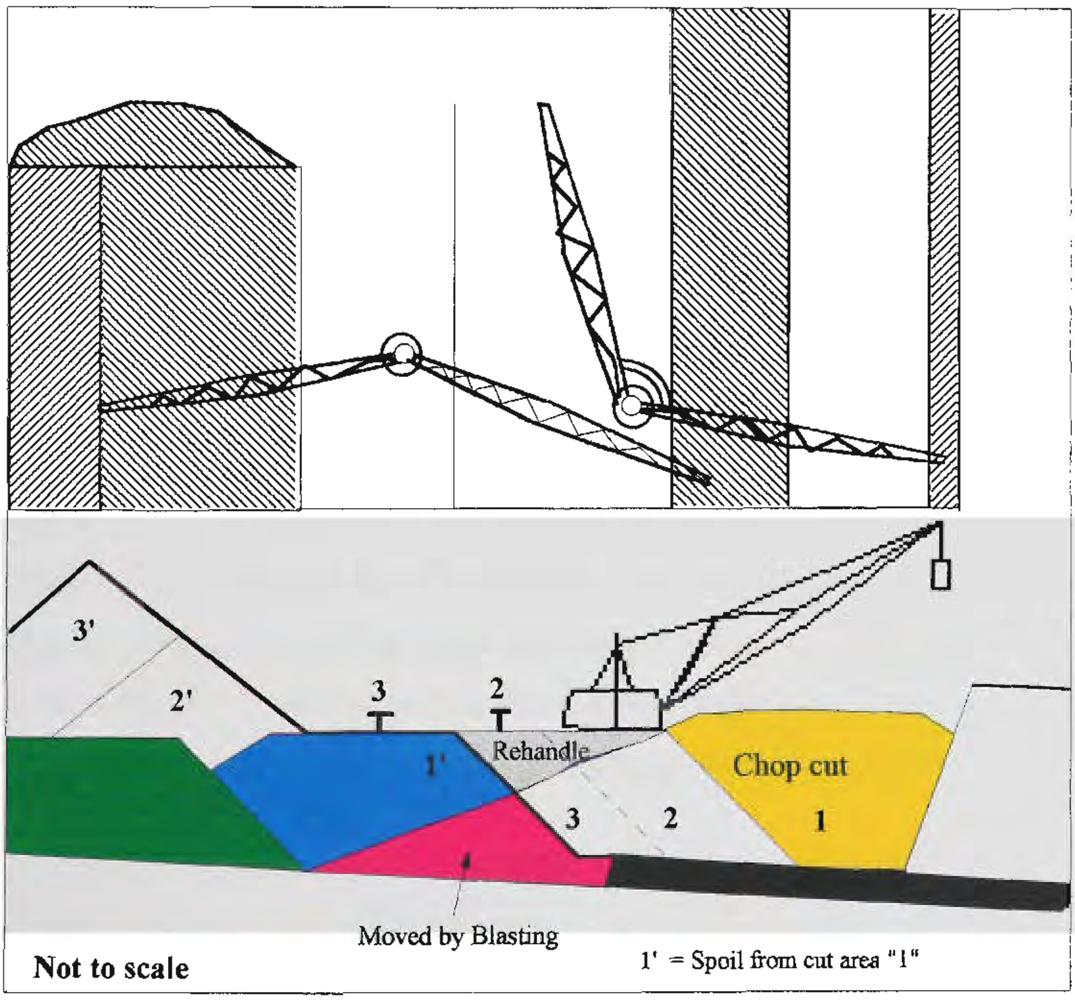


Figure 10.6- A schematic view of the dragline positions during the excavation of a block using the In-Pit Bench method.

The three dragline positions and the criteria to change the dragline's locations are described below.

1. The first sub-volume is to produce a key cut of one and a half bucket widths on the top of the coal seam with the dragline chopping the overburden from a lowwall side in pit bench. The bulk of the chop cut is used to form an in-pit bench on the lowwall side of the pit. The location of the dragline is dictated by the dragline reach and post blasting profile. From this position the dragline should reach the new highwall toe point.
2. If the spoil from the chop (key) cut is not sufficient to create the desired in-pit bench or the dragline cannot reach the inside corner of the key cut in the final position, a second position will be required to extend the key cut. This second position becomes necessary when the overburden depth is shallow compared to the coal and the parting thicknesses. This arises when the spoil from key cut in the shallow area is not enough to make the in-pit bench. Using the second position also reduces the swing angles and cycle times. The spoil from the second position is dumped into the final void when the in-pit bench is completed.
3. This is a pull back operation and the dragline removes the remainder of the material and dumps it into the void behind the return pass.

Extended Key Cut Method: The division of the pit cross-section for volumetric calculation in the Extended Key Cut method is shown in Figure 10.7. The procedures used for the dragline simulation of this method are almost identical to those used for the Lowwall In-Pit Bench method except that the first dragline position to remove the key cut (extended key cut) is at the highwall side instead of the lowwall side.

The CADSIM three modules EXTBENCH, INPIT and EXTKEY simulate the sequences of the dragline operation for the three digging methods. All the three digging methods are associated with some sort of throw blasting profiles which significantly effect the results of the simulation. After definition of sections and the establishment of strip geometry, a post blasting trajectory was simulated depending on pit geometry and swell factor for the three digging methods. The blasting profiles for various geological conditions and pit configurations were predicted by SABREX blasting software. A CADSIM macro (BLAST) was coded to read the blasting profiles as strings and fit the strings to the geometry of the simulated pit. A simulation of the digging process was then run for the sections from the centre ramp outwards to the northern pit boundary. The procedure was repeated from the centre ramp towards the southern pit boundary.

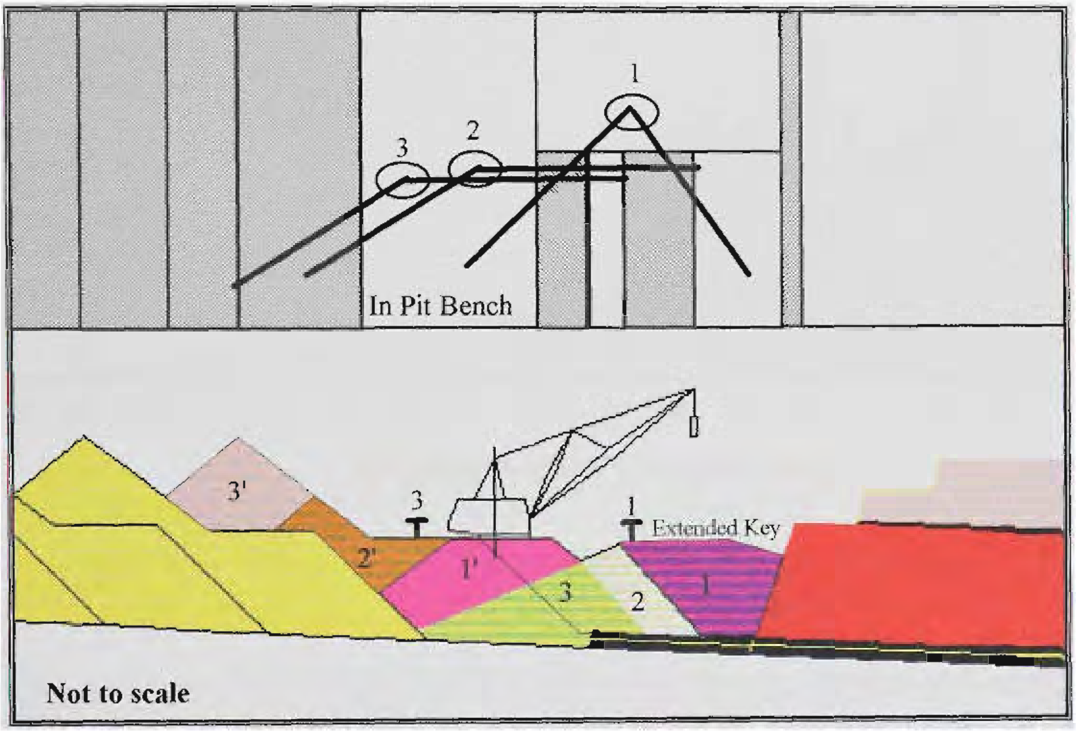


Figure 10.7- A schematic view of the dragline positions during the excavation of a block using the Extended Key Cut method.

10.2.4 Simulation Results

The CADSIM model is totally flexible in providing outputs in tabular and graphical format and information that can be used to evaluate mine productivity and other planning parameters were considered for pit optimisation study. The effects of both the overburden depth and strip width on rehandle percentage and productivity were investigated within the practical limits for each of the three digging methods.

10.2.4.1 Impact of the Overburden Depth

The thickness of the interburden allocated to the dragline (the interburden between Blackfield and Glen Munro coal seams) varies from 28 to 56 metres with an average of

47 metres over the mining area (Figure 10.8). These changes in depth of the dragline working level affect the dragline productivity due to the changes in rehandle and thrown percentage as well as cycle time. However, because the maximum digging depth of the available dragline is 58m the entire overburden can be removed as one pass.

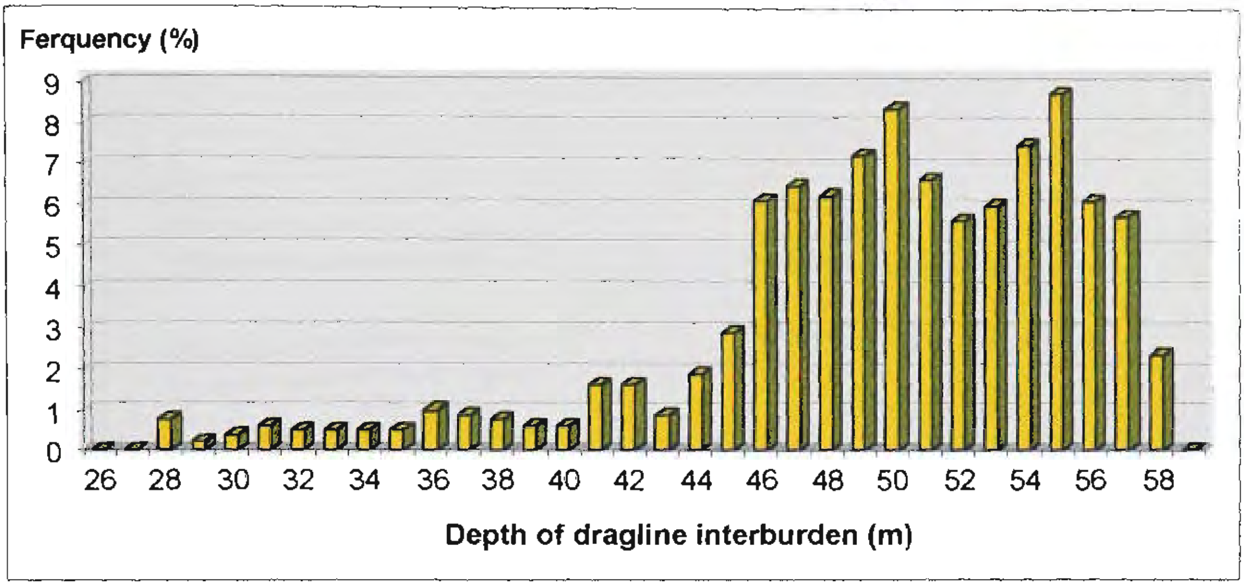


Figure 10.8- The frequency distribution of the dragline block depths.

In order to investigate the effect of depth variations on the operating parameters the first strip in the northern area was selected, since it has the widest range of depths. In this strip, the depth of the dragline interburden decreases for the mining blocks located further from the central ramp (Figure 10.9). The mining parameters were calculated for the three digging methods in each section every 30m.

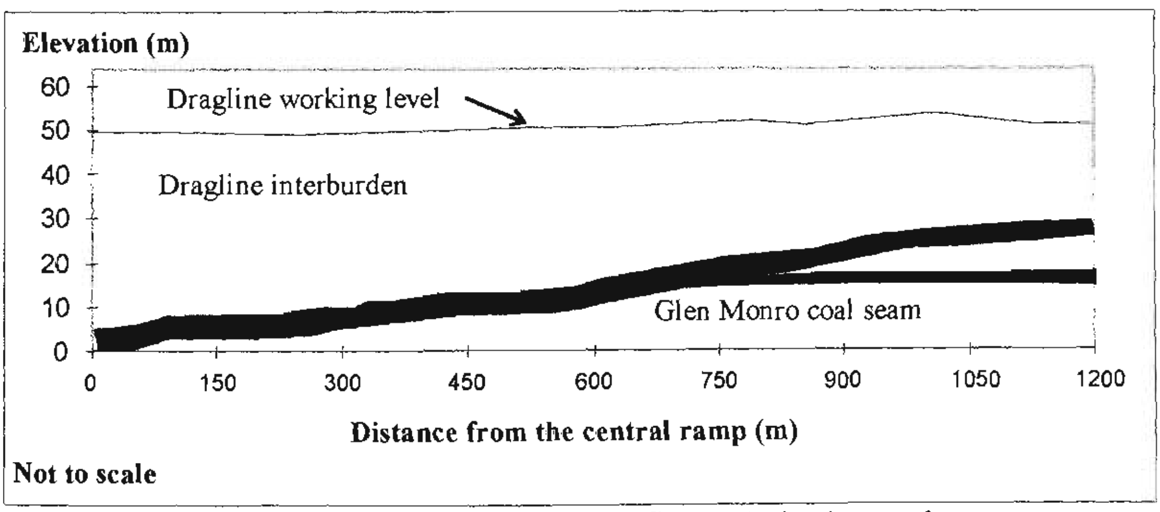


Figure 10.9- Cross section through the first strip in the northern area.

Since CADSIM generates the geology of the sections from the real geological data, the depth of dragline interburden cannot be changed arbitrarily, unless an advance bench is associated with the main dragline bench. This means that the dragline working level cannot be optimised for the Extended Bench method for this case study. Figure 10.10 shows how stripping parameters change along the strip due to the changes in dragline interburden depth for Extended Bench digging method.

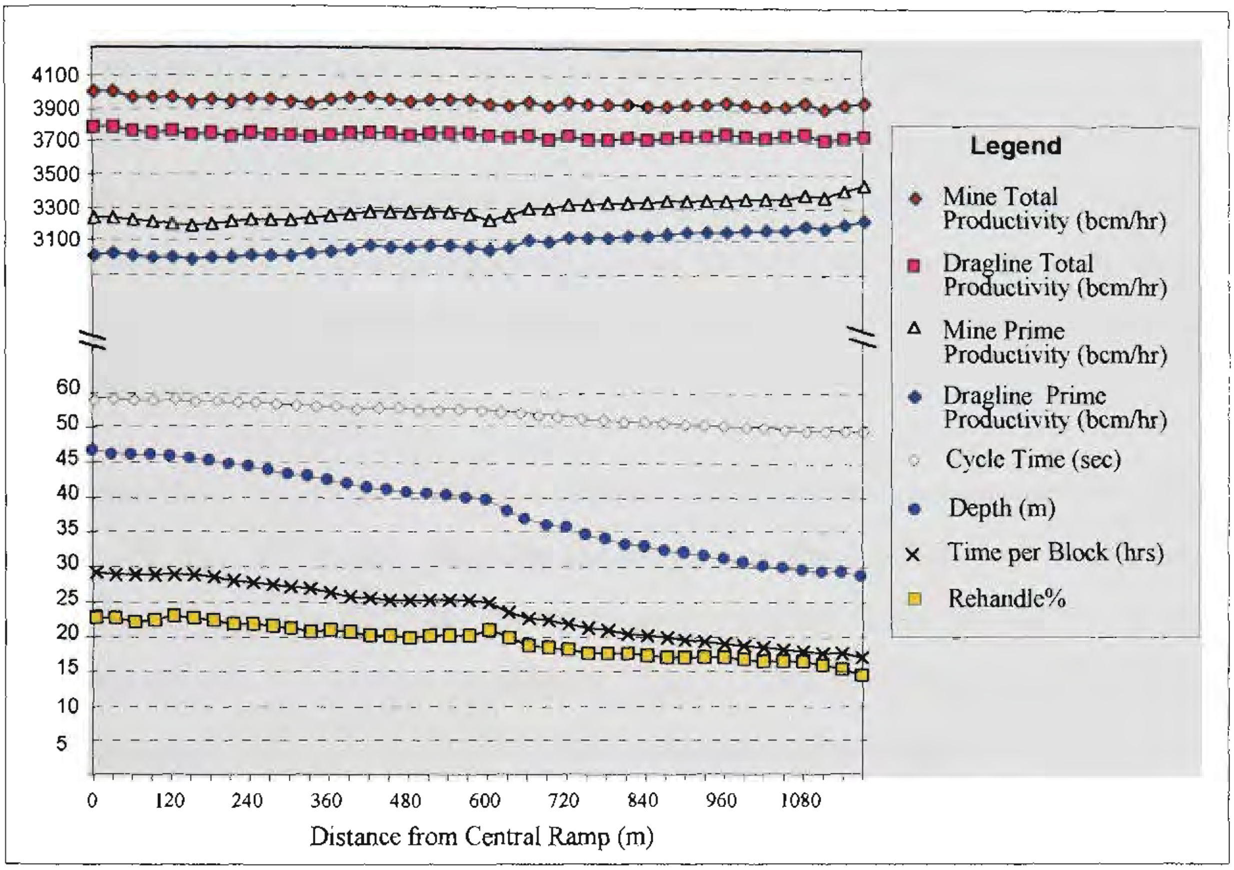


Figure 10.10- Changes in the stripping parameters for the Extended Bench method along the strip.

As can be seen in Figure 10.10 the rehandle percentage and cycle time decrease as the mining advances to shallower areas further from the central ramp. The changes in rehandle percentage and cycle time increase prime productivity for shallower blocks.

In the case of using an in-pit bench (ie. In-Pit Bench and Extended Key Cut digging methods), the dragline working level changes from the original level and it affects both the rehandle percentage and spoil available. To maximise the prime productivity the in-pit bench level must be kept minimum. On the other hand, the level of the in-pit bench cannot be reduced too much as the dragline must be able to dump all the material into the spoil area while working from this level. Other factors which must be considered while calculating the geometry of the in-pit bench are the dragline reach to the new highwall toe and a safe working width. In some cases the in-pit bench needs to be widened (ie. increased rehandle) to provide enough reach to access the new highwall toe.

Two macros SPBAL and INPIT were coded to perform the spoil balance and to calculate the optimum level of the in-pit bench. In these macros the required spoil room is first calculated based on the prime volume and swell factor and the final spoil geometry is established. In the next step a minimum level is calculated for the in-pit bench based on the post-blasting profile, required walking width and the dragline reach. This minimum level is then compared against the spoil room required. If the calculated level is not high enough for spoiling, the level is increased inside a loop until a satisfactory level is found.

Figures 10.11 and 10.12 illustrate variations in the stripping parameters along the strip for In-Pit Bench and Extended Key Cut digging methods. The in-pit bench level is a function of original depth and decreases as the original depth reduces. The rehandle percentage also decreases as the level of the in-pit bench decreases thus increasing the prime productivity. On the contrary the thrown percentage slightly reduces in shallower areas which can reduce mine prime productivity. However the effect of the reduction in rehandle percentage is more significant than the reduction in thrown percentage. The combination of changes in the rehandle, thrown percentage and cycle time causes the prime and total productivity to be increased for shallower areas for both In-Pit Bench and Extended Kay Cut digging methods.

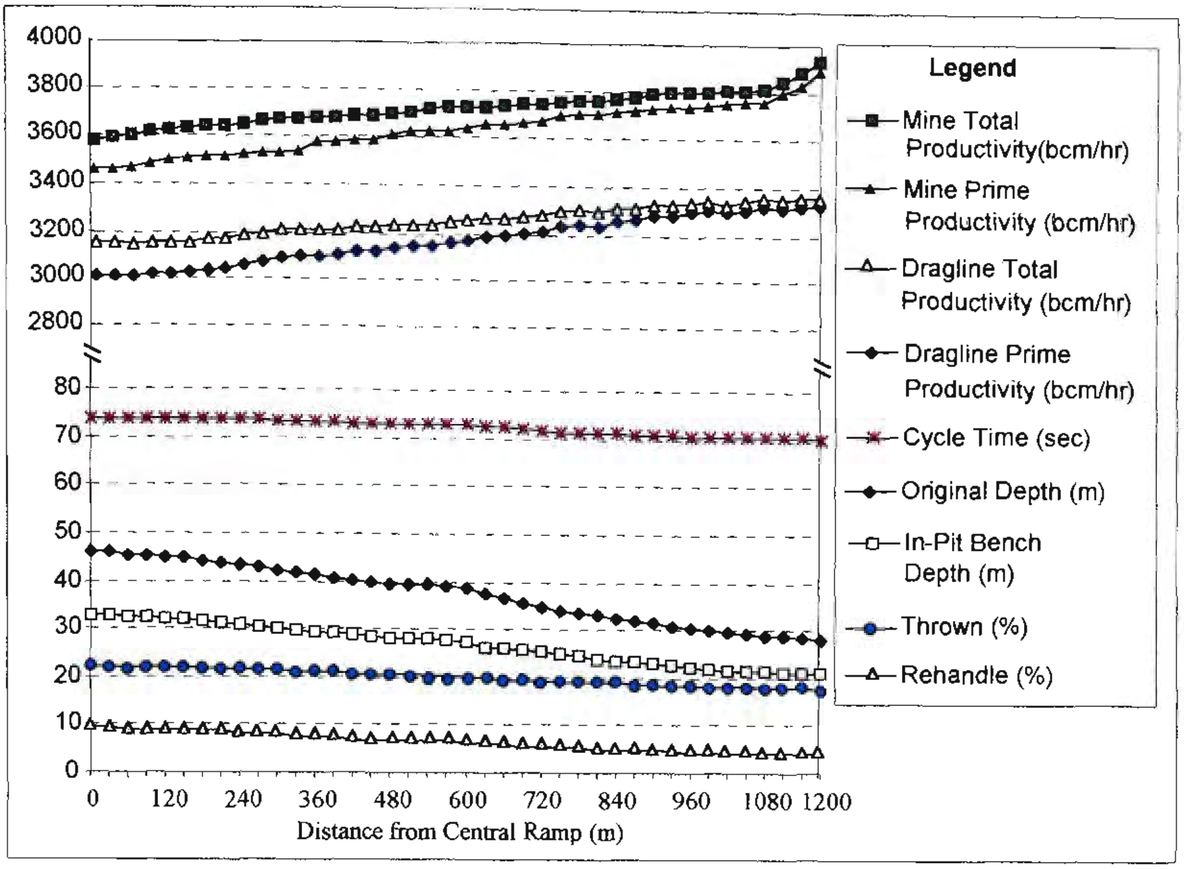


Figure 10.11- Changes in the mining parameters along the first strip for the In-Pit Bench digging method.

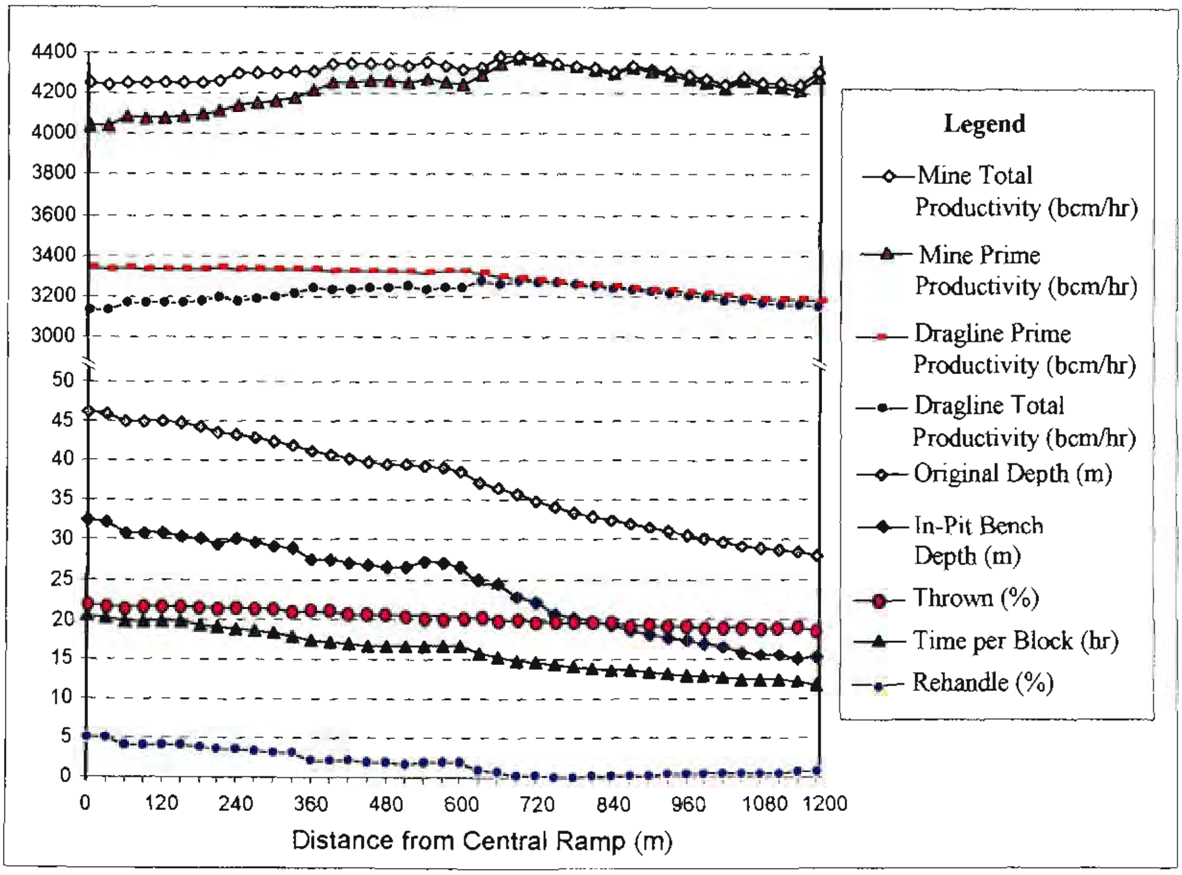


Figure 10.12- Changes in the mining parameters for the Extended Key Cut method.

10.2.4.2 Impact of Strip Width

Strip width is one of the most critical parameters influencing dragline productivity for a selected digging method. For instance, in the standard Extended Bench method as long as the key cut material can be dumped directly into the void from the previous strip, increasing the strip width will reduce the quantity of rehandle material. However, for an In-Pit Bench method narrower strips are often more productive. The effect of the strip width is discussed here separately for each of the three methods.

I) Extended Bench Digging Method: For the standard Extended Bench method, the rehandle decreases for wider strips. This reduction in rehandle is due to the relatively lower increment in the bridge volume compared to the increment in the prime volume when widening the pit. Figure 10.13 shows the trends in the rehandle, the cycle time and the coal exposure rate as a result of changes in the strip width when using Extended Bench method. The thrown percentage also slightly decreases as the strip width increases.

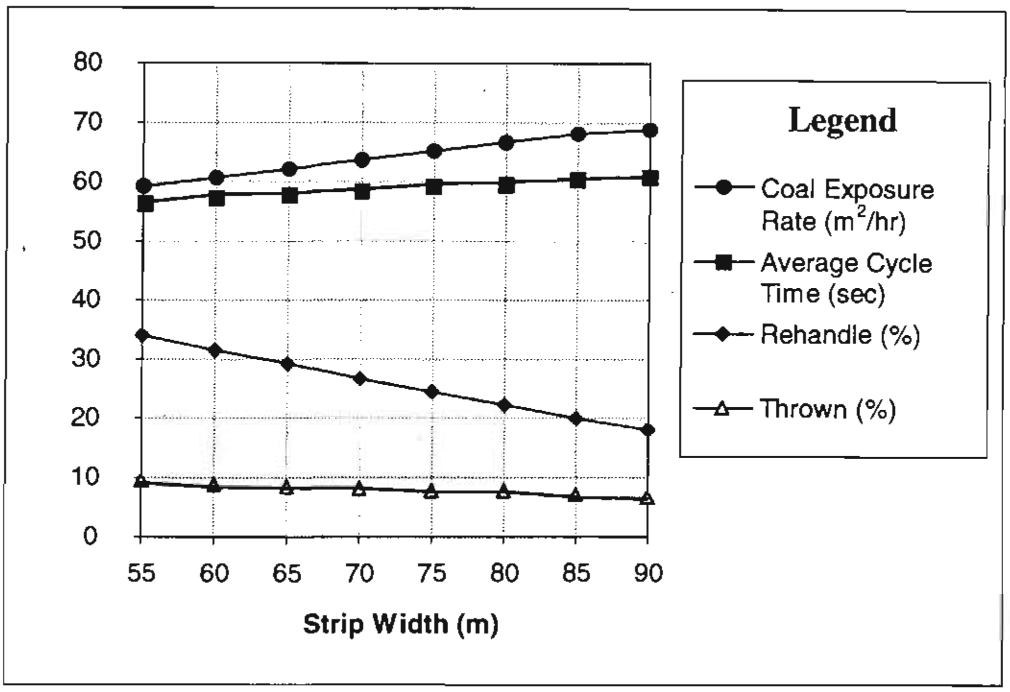


Figure 10.13- Effect of the strip width on the stripping parameters for the Extended Bench method.

In Figure 10.14 both the mine and the dragline total productivity decreases for strip widths exceeding 70m as a result of increase in cycle time. But the prime productivity continues to increase for wider pits as the effect of reduction in the rehandle is more significant and can offset the slight increase in the cycle time. In Figure 10.14 the "Mine Productivity (Total and Prime)" includes the material moved by throw blasting, while the "Dragline Productivity (Total and Prime)" refers to the rate of material movement only by the dragline. The dragline prime productivity is influenced by the rehandle percentage and the cycle time while the mine prime productivity is influenced by all the three factors, rehandle percentage, thrown percentage and cycle time. By considering the reduced rehandle, reduced thrown percentage and increased cycle times, an optimum strip width can be obtained.

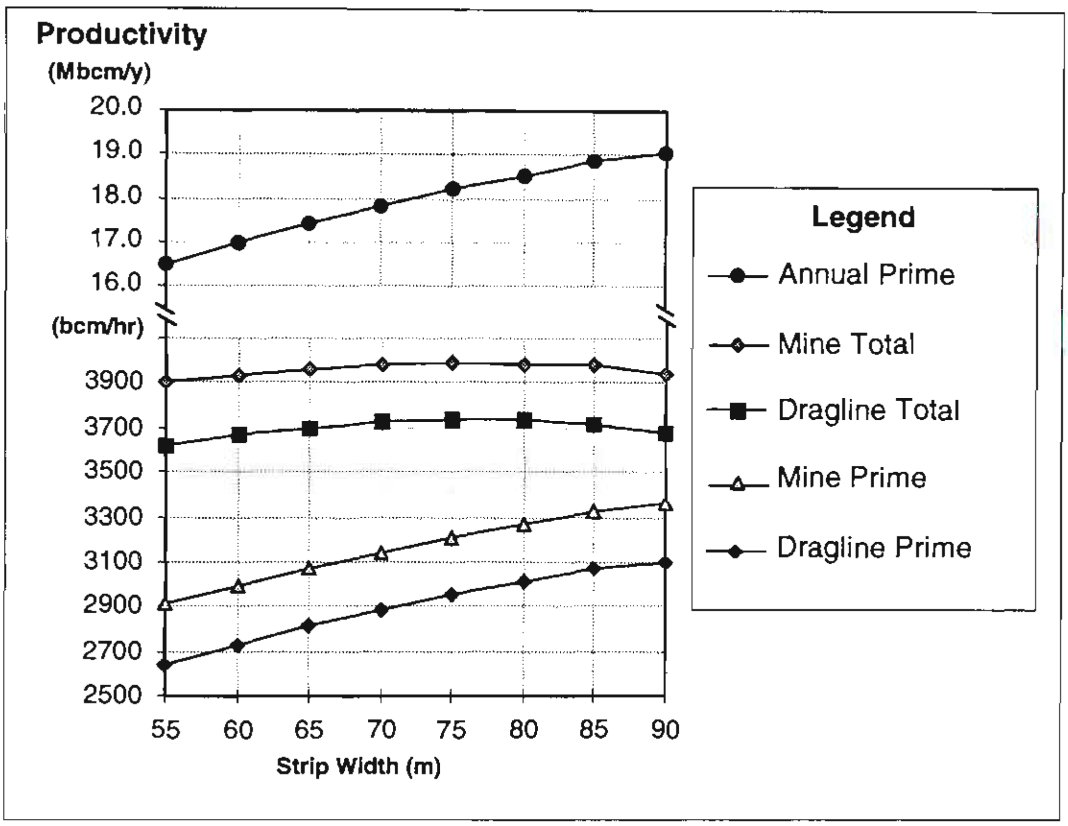


Figure 10.14- Impact of strip width on productivity for the Extended Bench method.

In practice several operational factors must also be considered in the selection of strip width. In this case study when the pit width exceeds 75m, the material from the key cut can no longer be directly placed in the new bridge. This causes additional rehandle which must be moved either by the dragline or by the dozer.

II) In-Pit Bench Digging Method: A wide pit is generally favourable for coal loading and permits greater safety for men and equipment. In practice narrower pits are more productive with digging methods which require the thrown blasting technique and lowwall side stripping. As shown in Figure 10.15, the rehandle percentage with the In-Pit Bench method increases slightly for wider pits. This is because the level of the in-pit bench increases moderately in wider pits and the dragline must be able to reach the toe of the new highwall. This is only true for strip widths less than 75m since there is a slight decrease in rehandle with pit widths exceeding 75m.

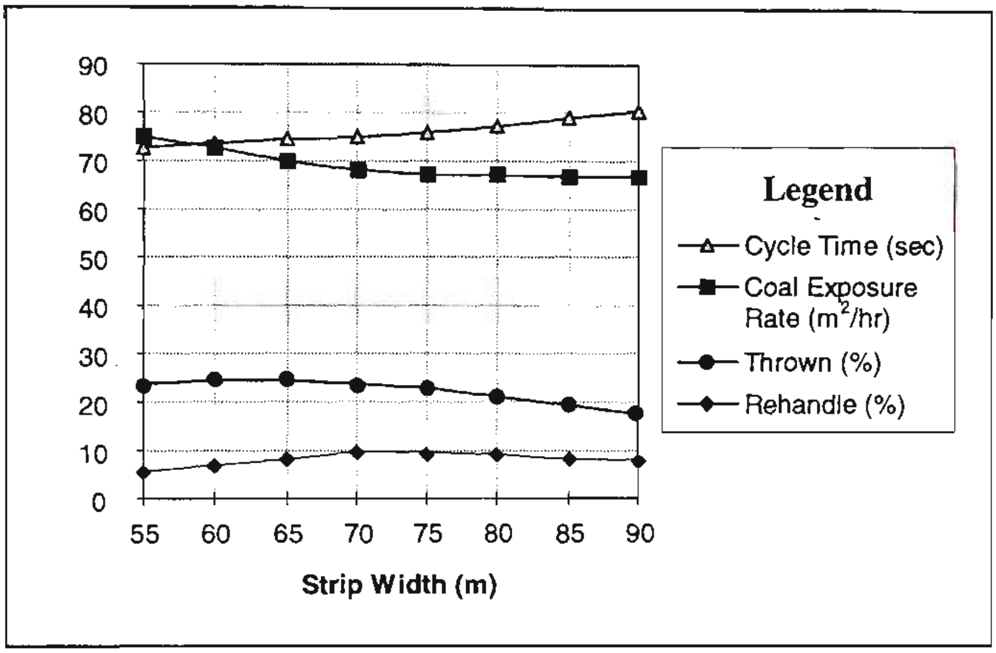


Figure 10.15- Effect of the strip width on the stripping parameters for the In-Pit Bench method.

In a strip mine the minimum practical pit width is usually dictated by the manoeuvrability of the coal loading and haulage equipment. In this case study a fleet of loader and trucks is used for the coal loading and haulage operation (Figures 10.16 and 10.17) which require a minimum 55m pit width. The effect of strip width variations on the productivity terms is depicted in Figure 10.18.

Figure 10.16- Coal loading operation at the case study mine.

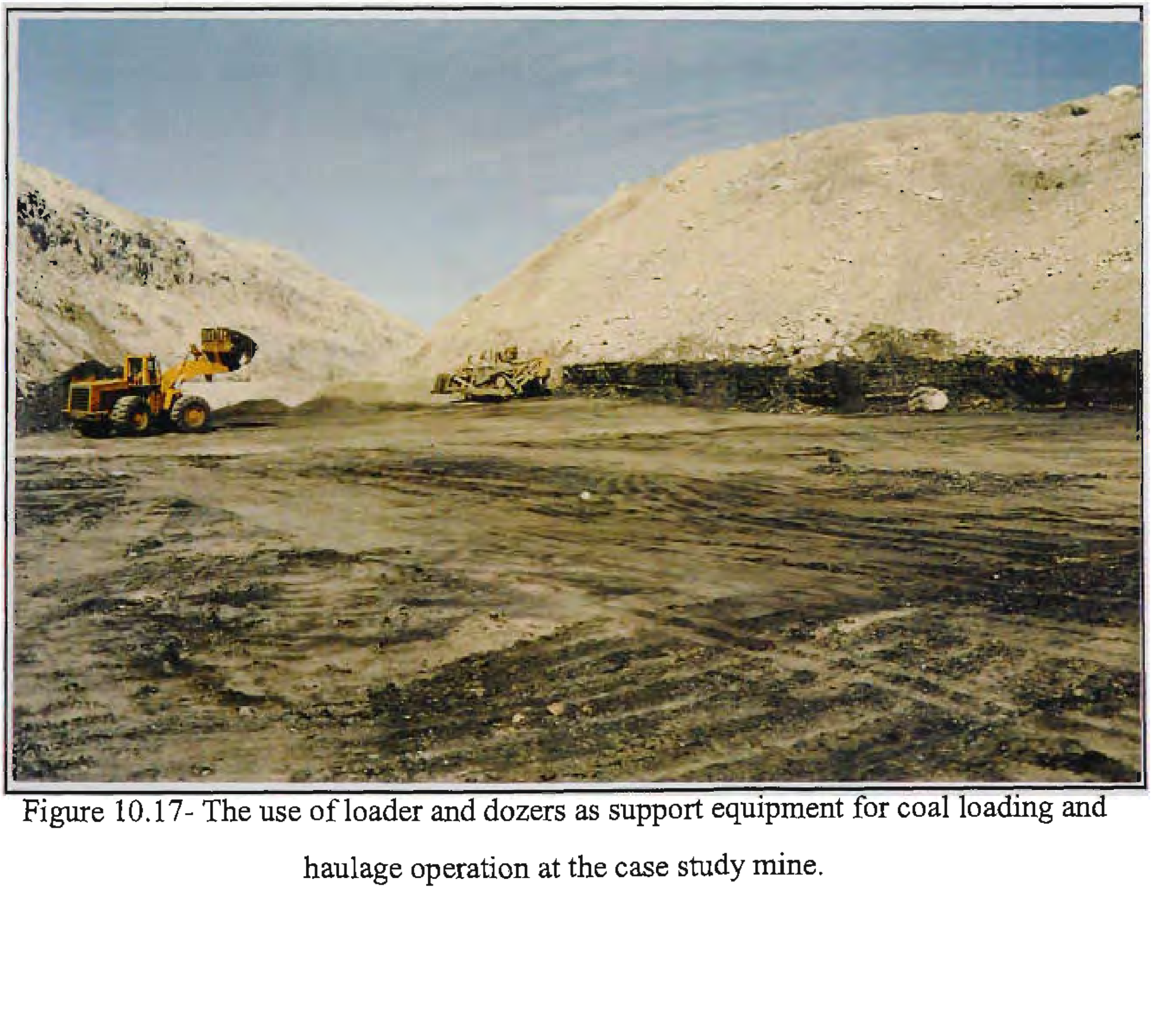


Figure 10.17- The use of loader and dozers as support equipment for coal loading and haulage operation at the case study mine.

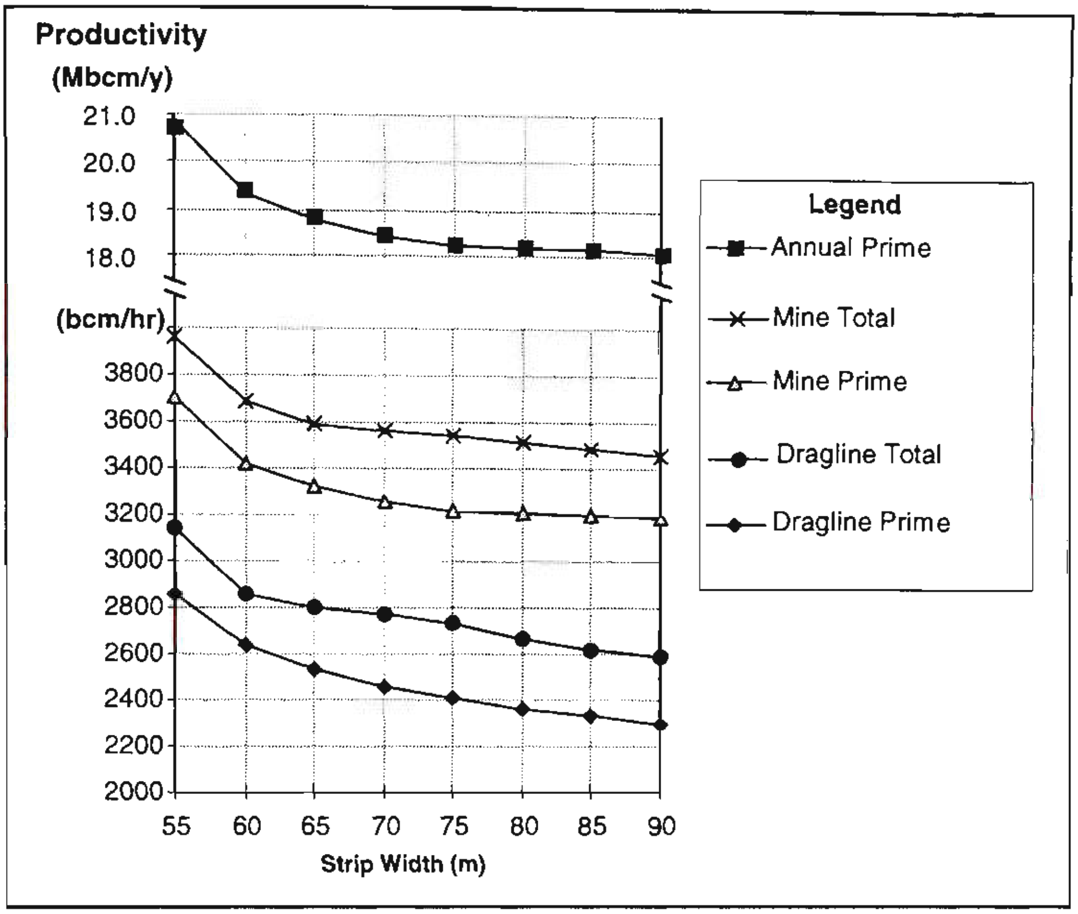


Figure 10.18- Impact of the strip width on productivity for the In-Pit Bench method.

While the results show that for the In-Pit Bench method the narrower strips are more productive, the rate of reduction in the mine prime productivity decreases and remains almost constant for strips wider than 70m mainly due to the rehandle reduction in these strips.

III) Extended Key Cut Digging Method: As with the In-Pit Bench method, the rehandle percentage increases slightly for wider pits in the Extended Key Cut method. This is because the level of the in-pit bench must be increased moderately to provide more spoil room in wider strips. In addition to rehandle percentage, the average of cycle time and thrown percentage change both in disfavour of the wide strip. The combination of all these critical parameters (rehandle, thrown and cycle time) and other operational parameters such as the dragline walking time can lead to an estimation of the coal exposure rate as shown in Figure 10.19.

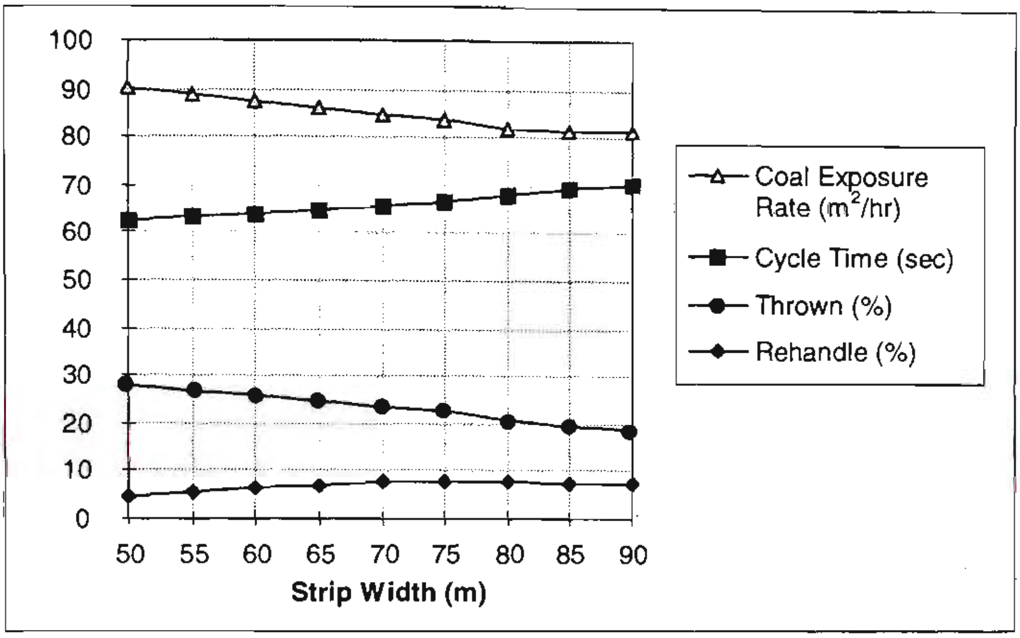


Figure 10.19- Effect of the strip width on the mining parameters for the Extended Key Cut method.

As shown in Figure 10.19 the coal exposure rate decreases as the strip width increases. This is only true for strip widths less than 80m. The coal exposure rate remains practically constant for strip widths exceeding 80m due to a slight reduction in rehandle percentage. The effect of variations in strip width on the productivity terms is depicted in Figures 10.20. The results show that the dragline productivity is sensitive to the changes in the strip width. The main reasons for the reduction in the productivity for wider strips are the reduction in the thrown percentage, increased cycle time and to some degree an increase in rehandle percentage for wider strips.

10.2.5 Dragline Productivity Calculation

Once the optimised strip width for each method is determined, numerous information relating to the operating parameters and productivity can be derived from the simulation. This kind of information may be used for analysing machine performance, scheduling and mine planning procedures, cost estimation studies and for comparison purposes to select an optimal pit layout, digging method or dragline size.

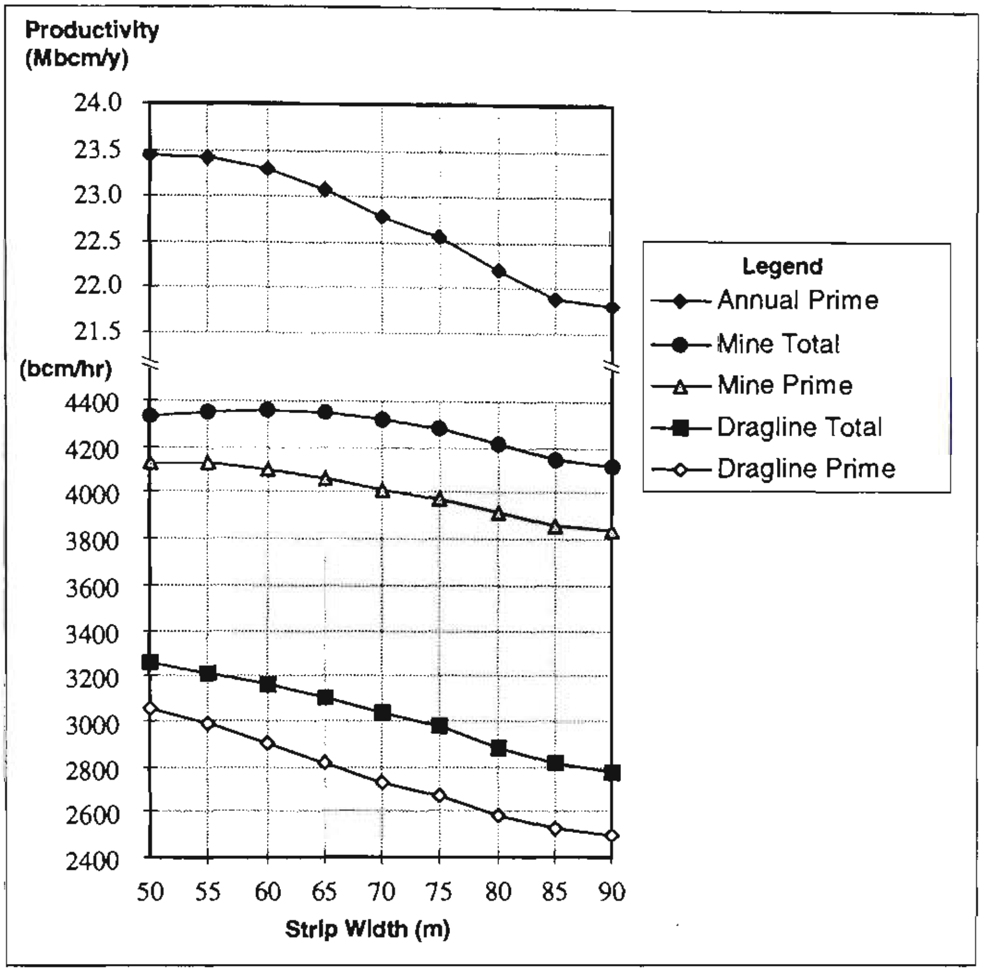


Figure 10.20- Impact of the strip width on productivity for the Extended Key Cut method.

The annual dragline productivity can be calculated either by multiplying the average of the prime productivity per hour by the estimated dig hours per year or by considering the cumulative time required for mining individual blocks. The latter seems to provide more accurate results since the effect of dragline ramps and end wall operations can also be included. The result of the second method of calculation may be directly used in scheduling and cost estimation procedures. The results of both methods of annual productivity calculation for each digging method with strip width of 90m for the standard extended bench and 55m for both in-pit bench method and extended key cut method are discussed below.

I) Using the Average Values: The following formula were used to calculate annual productivity.

$$Annual\ Productivity = Average\ Prime\ Productivity\ of\ all\ blocks\ (bcm/hr) \times Annual\ Digging\ hours$$

The estimated mine annual productivity for the three digging methods using the above equation are given below.

*1) **Extended Bench Method:*** $3205 \times 5676 = 18{,}191{,}580$ *bcm/y*

*2) **In-Pit Bench Method:*** $3459 \times 5676 = 19{,}633{,}280$ *bcm/y*

*3) **Extended Key Cut Method:*** $4055 \times 5676 = 23{,}016{,}180$ *bcm/y*

II) Using the Cumulative Time: Due to the variability of the geological conditions over the mining area, a more accurate way to estimate dragline annual productivity is to use both the time the machine spent per mining block and the volume of the block. With this method of calculation, the total time required for each strip is first calculated using the time spent to mine each block. The estimated time is compared against the annual dig hours to locate the dragline position at the end of the year as well as the number of blocks. Finally, the sum of prime bank material from the area which was stripped during the one year provides annual productivity. This method is commonly used by most scheduling software. Table 10.3 and Figure 10.21 summarise the results of the second type of annual productivity calculations (for the pit used for simulation).

Table 10.3- The estimated annual productivity using the cumulative time method.

Digging Method	Annual Productivity (Mbcm/y)							
	1st	2nd	3rd	4th*	5th	6th	Average	Variation**
Extended Bench	18,648	18,471	18,657	15,576	18,084	18,110	17,924.8	-1.5%
In-Pit Bench	20,127	19,893	19,911	16,805	19,725	19,765	19,371.2	-1.4%
Extended Key Cut	23,595	23,320	23,342	19,700	23,230	23,236	22,737.5	-1.2%

* *Major maintenance year*

** *(% of changes with the average method)* $= \dfrac{\text{Cumulative method} - \text{Average method}}{\text{Cumulative method}} \times 100$

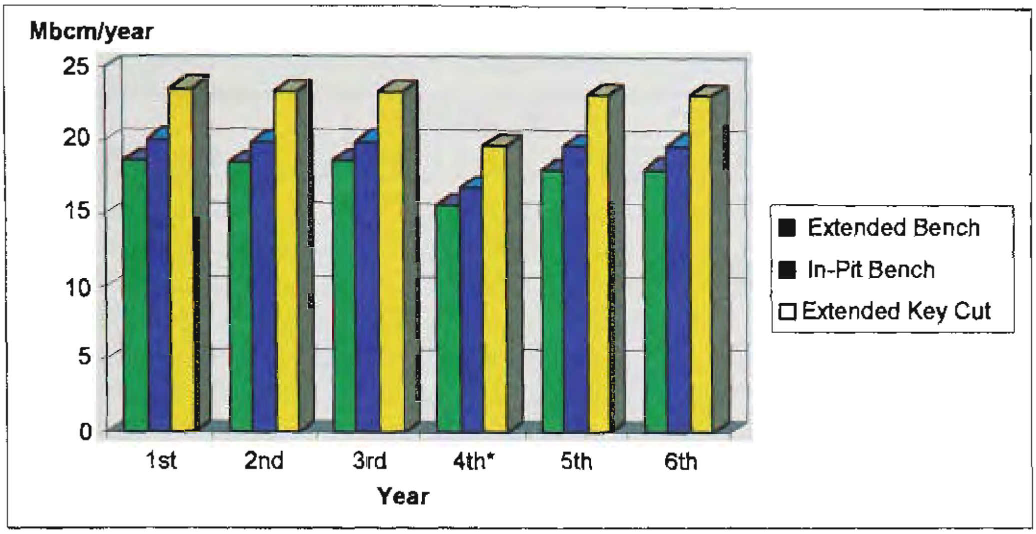


Figure 10.21-Annual productivity estimations for the first 6 years employing different digging methods.

The results show that the annual productivity remains almost constant for each digging method during the first six years of the mine is life, except for the fourth year which is a major maintenance year. The slight variation in the annual productivity is generally due to changes in the dig depth for some strips. Comparing with the results of the first method (average value), the estimated productivity from the block by block (cumulative time) method is slightly lower for all of the three digging methods. The variation between the results from the two methods is not significant for this level of study. However, the block by block method provides a more detailed type of calculation and information which may be used in short term planning studies.

10.2.7 Comparison of the Digging Methods

For this case study a detailed dragline simulation using CADSIM followed by a productivity and cost analysis was conducted to provide a reliable basis for the digging method selection. The average values for the first 15 strips in both the southern and northern areas with each digging method are given in Table 10.5. Figures 10.22 and 10.23 graphically show the same results for the critical parameters. The strip widths used as the basis for the simulated pit geometry were 90m for Extended Bench and 55m for both In-Pit Bench and Extended Key Cut digging methods.

Table 10.4- Average values of different operational parameters for the three digging methods.

Parameters	Extended Bench			In-Pit Bench			Extended Key Cut		
	South	North	Average	South	North	Average	South	North	Average
Pit width (m)	90	90	**90**	55	55	**55**	55	55	**55**
Swing angles (deg)	88.7	90.5	**89.6**	142.5	143	**142.8**	104.5	103.7	**104.1**
Cycle time (sec)	57.7	58.1	**57.9**	72.6	72.8	**72.7**	63.3	63.1	**63.2**
Walking per block (m)	161	180.1	**170.5**	108.6	112	**110.3**	102.1	106.4	**104.3**
Total time per block (hrs)	27	33.1	**30.1**	21.7	22.8	**22.2**	20	19.8	**19.8**
Original overburden depth (m)	42	49.3	**45.6**	44.2	47.2	**45.7**	44.2	47.2	**45.7**
In-pit bench depth (m)	-	-	-	34.1	35	**34.5**	32.4	33.6	**33.0**
Total coal thickness (m)	7.02	6.76	**6.9**	6.8	6.6	**6.7**	6.8	6.6	**6.7**
Total parting thickness (m)	5.8	1.05	**3.43**	3.47	0.97	**2.22**	3.47	0.97	**2.22**
Rehandle percentage (%)	26.6	26.8	**26.7**	5.4	5.8	**5.6**	5.6	5.8	**5.7**
Thrown percentage (%)	8.09	8.11	**8.1**	22.4	22.6	**22.5**	22.4	22.6	**22.5**
Dragline prime prod.* (bcm/hr)	3,010	2,882	**2,945**	2,670	2,692	**2,680**	3,148	3,140	**3,144**
Dragline total prod. (bcm/hr)	3,865	3,715	**3,798**	2,855	2,894	**2,874**	3,368	3,375	**3,371**
Mine prime prod. (bcm/hr)	3,274	3,136	**3,205**	3,440	3,478	**3,459**	4,060	4,050	**4,055**
Mine total prod. (bcm/hr)	4,125	3,976	**4,050**	3,627	3,680	**3,654**	4,280	4,284	**4,283**
Coal exposure rate (m^2/hr)	76.8	63.8	**70.3**	76.9	73.5	**75.2**	83.7	83.4	**83.6**
Annual prod. (10^6×bcm/year)	18,580	17,800	**18,192**	19,530	19,740	**19,633**	23,039	22,983	**23,016**

** prod. = productivity*

The results of the productivity analysis suggest that among the three stripping methods considered, the Extended Key Cut method removes the highest amount of the overburden due to the reduced rehandle and increased thrown percentage. Although the rehandle and thrown percentage are almost the same as for the In-Pit Bench method, due to the higher cycle times the prime productivity is lower than that of the Extended Key Cut method. The dragline total productivity is highest for the standard Extended Bench method due to the shorter swing angles and similarly cycle times. Among the three digging methods, the Extended Key Cut method is the most productive and results in highest rate of coal exposure.

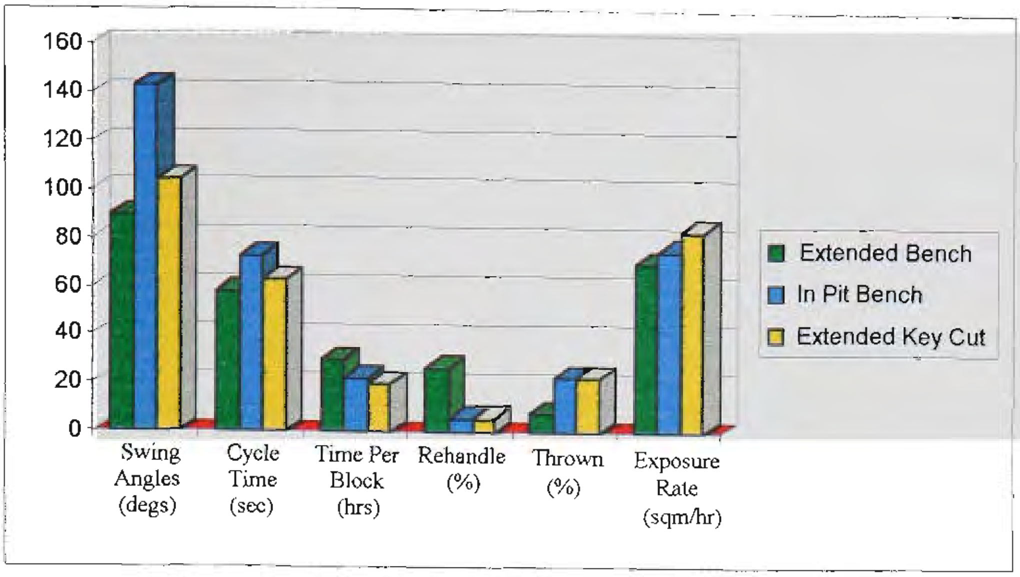


Figure 10.22- Comparison of the different operational parameters for the three digging methods.

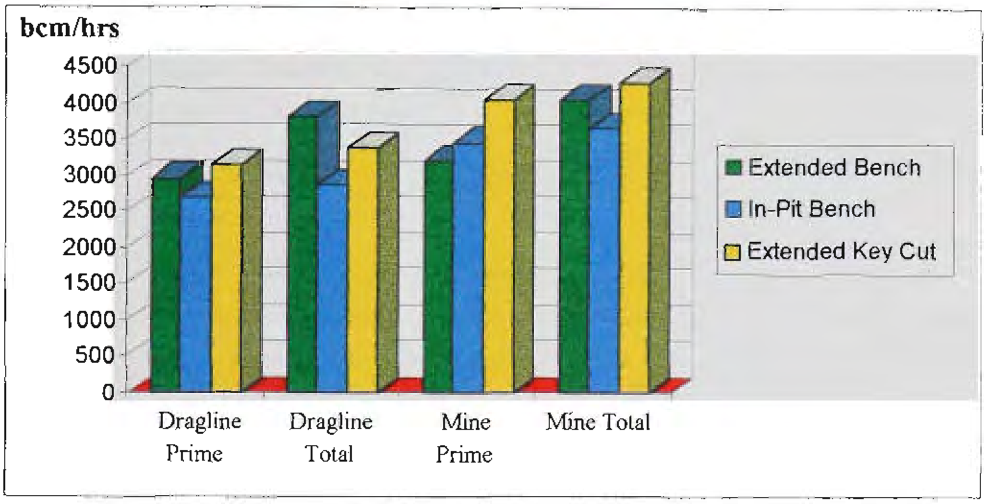


Figure 10.23- Comparison of the different productivity terms for the three digging methods.

Selection of the most cost effective digging method is usually the most critical decision in the mine planning process and requires an evaluation of the capital cost of the equipment and operating costs. In order to establish these costs for a given project a number of parameters must be defined. These parameters include the production target, mine life, strip ratio, overburden removal rate, the physical characteristics of the material to be handled, drilling and blasting patterns and so on. The dragline simulation part of the study using CADSIM provided some of the required information for a cost analysis study. Others were provided by management of the case study mine.

The discounted average production costs for the three digging methods are summarised in Table 10.5. The results show that the production cost is relatively higher for the Extended Bench method compared to those of the other two methods. This is due to a lower prime productivity, offset to some extent by the cost of drilling and blasting. The total cost for both Extended Key Cut and Extended Bench are almost the same. However, if the coal losses due to thrown blasting are included, the Extended Bench method results in the least cost per tonne of the recoverable coal. On the other hand, employing the Extended Key Cut method can considerably increase mine productivity which in turn will reduce the fixed costs such as overhead and mining lease costs. In this case study the mine management opted for the Extended Key Cut method as the case study mine targeted increased dragline productivity as first priority due to scheduling and blending problems existing within the operation.

Table 10.5 - Discounted Average Cost of the various cost centres .

		Digging Method		
Cost Centre		***Extended Bench***	***In-Pit Bench***	***Extended Key Cut***
Dragline	*Operating Cost*	0.44	0.41	0.35
	Capital	0.94	0.87	0.75
	Sub-total	1.38	1.28	1.10
Drill and Blast	*Operating. Cost*	0.28	0.60	0.60
	Capital	0.10	0.08	0.08
	Sub-total	0.38	0.68	0.68
Dozer	*Operating Cost*	0.02	0.02	0.02
	Capital	0.02	0.02	0.02
	Sub-total	0.04	0.04	0.04
Total		**1.80**	**2.00**	**1.82**

** Cost based on A$ per prime bank cubic metre*

The results of cost comparison of the digging methods is summarised in Figures 10.24 and 10.25.

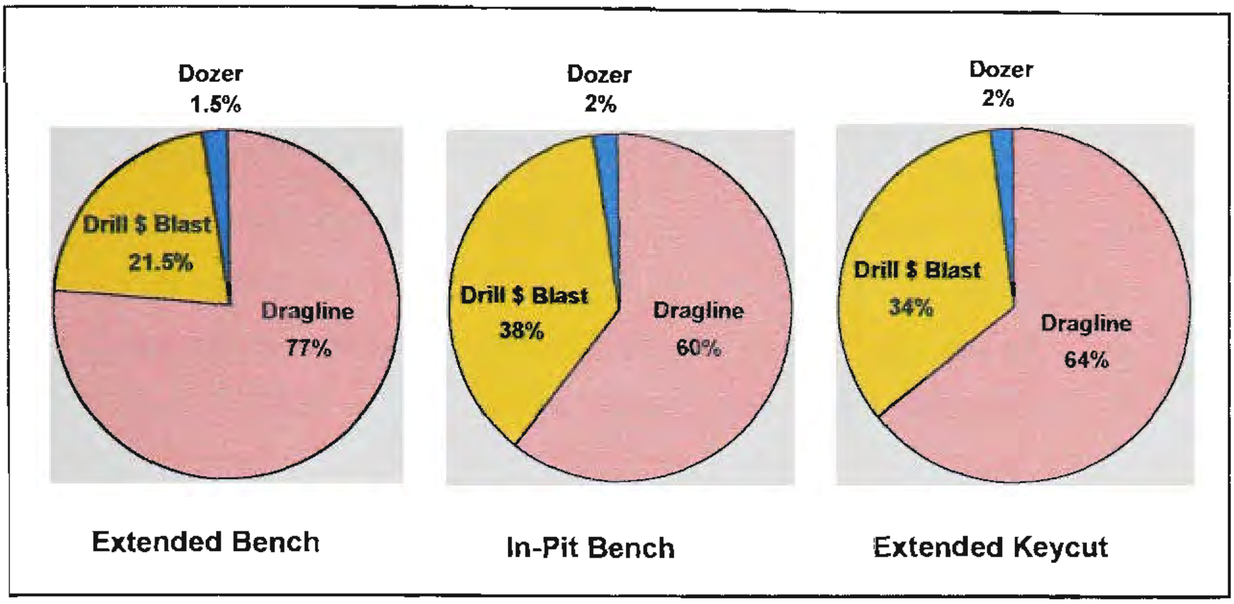


Figure 10.24 - Proportional cost of the components for the three digging methods.

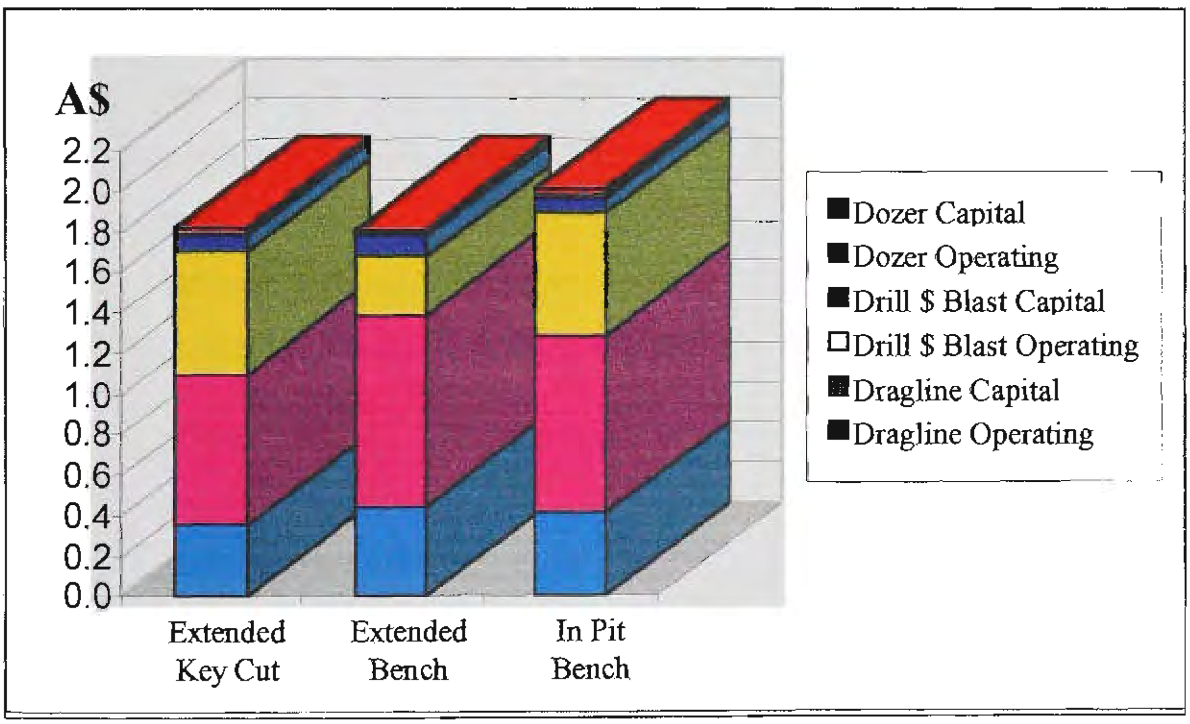


Figure 10.25- Discounted Average Cost of the various components.

10.3 CASE STUDY 2

This section deals with the results of applying the CADSIM model to a typical Central Queensland (Bowen Basin) dragline operation to remove a thick overburden covering a single thick coal seam. This case study was divided into two main phases.

1. The first phase involves an optimisation process to determine the best pit configuration for the proposed strip layout. Both the strip width and the dragline dig level in the first pass are studied in this phase, since they have a major influence on the rehandle percentage and the productivity of the dragline.
2. The second phase involves dragline simulation runs to determine productivity of the optimum strip and pit configuration for the entire deposit and to provide block by block information for a long term scheduling program.

10.3.1 Geology of the Deposit

Economic coal reserves at the mine are currently confined to a thick coal seam. The coal seam is subdivided into six splits throughout the deposit. The majority of coal production comes from the upper four splits with minor quantities for blending won from the lower two splits. A cross section of the coal seam splits is shown in Figure 10.26. The very thick coal seam that occurs in the mine is quite typical of Australian coal deposits in Central Queensland (Bowen Basin). The mineable coal seam averages 20m with occasional thicknesses up to 23m. The overburden thicknesses over the mining area range from 40 to 120m with an average of 70m (Figure 10.27).

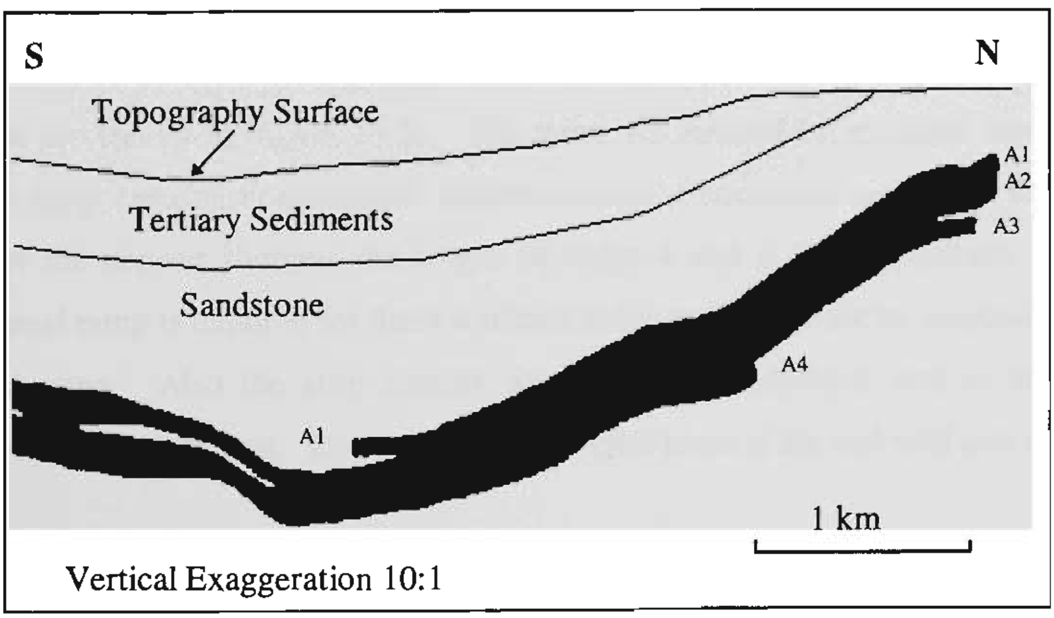


Figure 10.26- A typical cross section throughout the deposit.

Figure 10.27- Thick coal seam and thick overburden at the second case study mine.

10.3.2 Mining Layout

The mining boundary in the second case study deposit is a relatively isolated deposit with definable mining limits. The northern, western and southern limits of the deposit are characterised by oxidised coal. The north-south strike length is approximately 2km. The proposed strip layout and a set of simulation sections in the northern part of the deposit are shown in Figure 10.28. The strips are divided by a central lowwall coal access ramp creating northern and southern areas. Presence of a dyke in the western side of the deposit shortens the length of strips 4 and 6 in the southern part. An additional ramp is required for these southern strips as they cannot be accessed from the central ramp. Also the strip lengths decrease in the southern area as the mining advances toward the east. This provides more spoil room at the end wall side of strips.

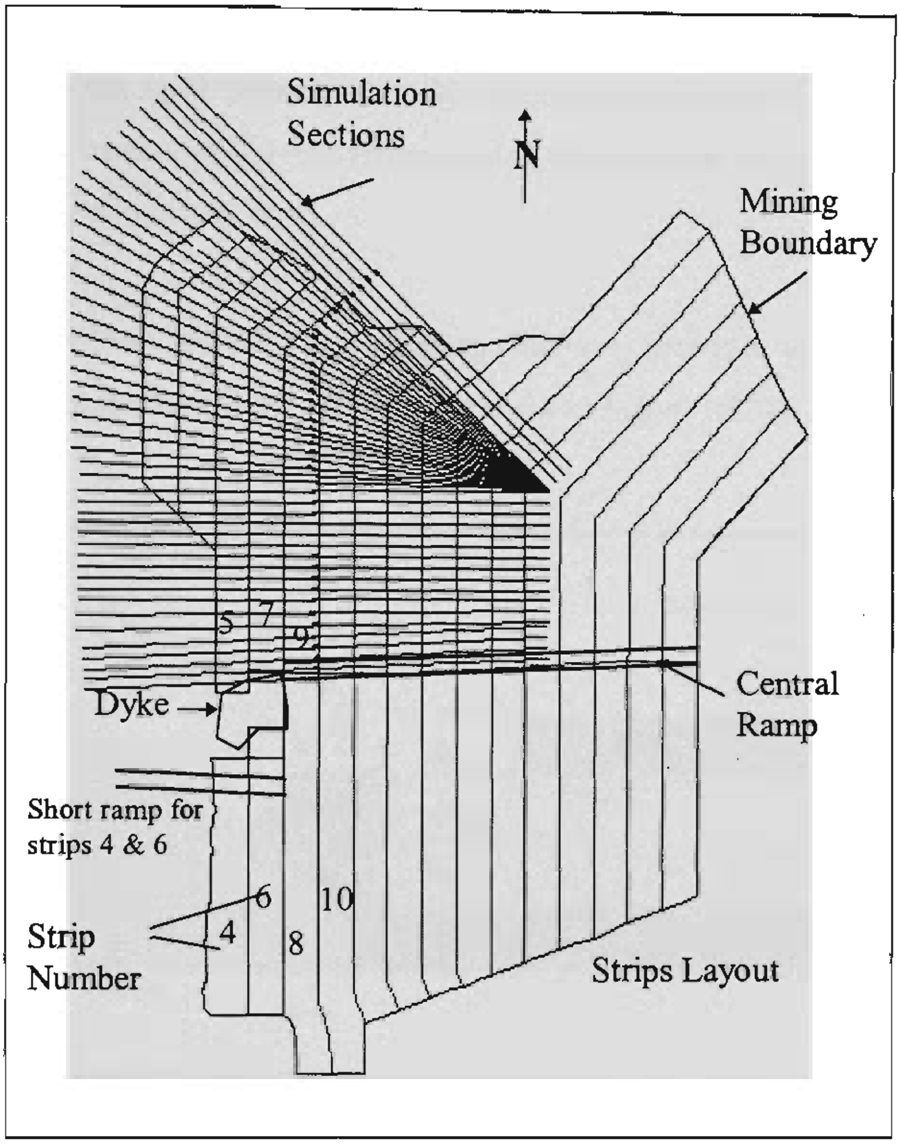


Figure 10.28- Strip layout and the northern sections used for the second case study.

10.3.3 Dragline Digging Method

The mine has been worked by truck and shovel method in the low ratio area in the western side of the deposit since critical development in 1982. A BE 1350W walking dragline equipped with a 33m^3 bucket was introduced in 1994. The dragline stripping method considered in this study is a modified split bench method. The current operating method includes:

1. A single pass extended bench digging method where overburden thicknesses are less than 45m.
2. A single pass extended bench digging method with overhand chopping where overburden thickness is between 45m and 60m.

3. A two pass extended bench digging method where overburden thicknesses exceed 60m but spoil capacity is within the operating limits of the dragline. A prestripping truck and shovel system is used ahead of the dragline in very deep area (usually more than 70m).

A typical cross section showing the digging method, together with the digging components used for the dragline simulation is shown in Figure 10.24.

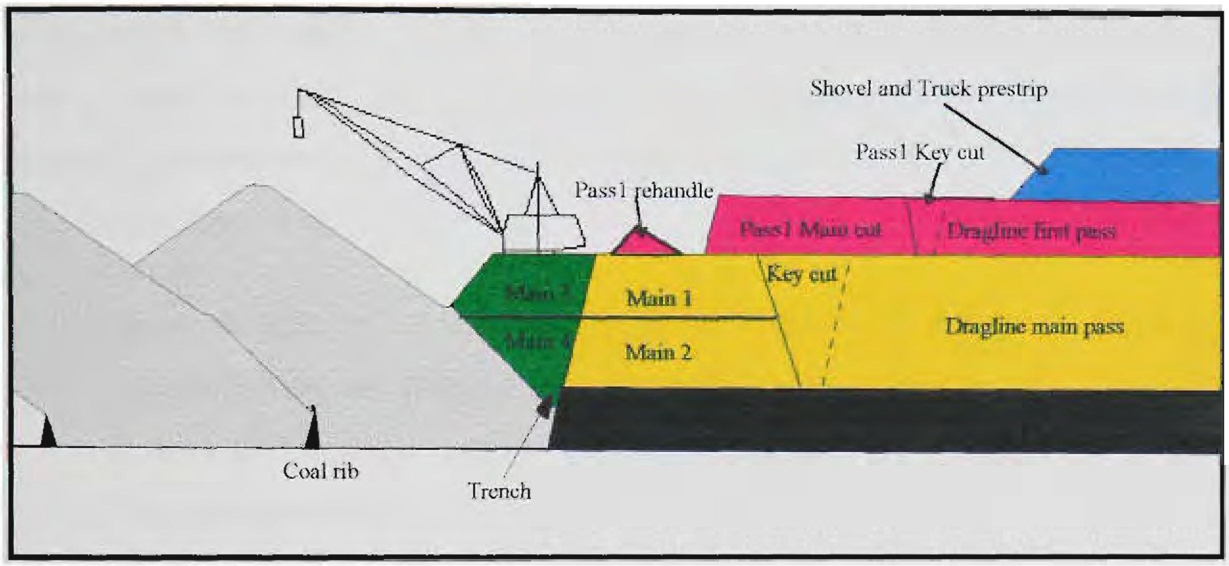


Figure 10.24- A typical cross section of the dragline digging method.

The mining sequences and the dragline walking patterns are as follows:

Southern Area: The first dragline pass commences from the end wall and digs from the south to the central ramp. At the end of its first pass, the dragline ramps down to the main pass level. The dragline strips the main pass from the central ramp and works to the south. A 40m wide walking road is provided by offsetting the first pass from the new highwall. The dragline uses this return road at the end of the main pass and walks back towards the central ramp, then follows the top pass to the topography. As the mining advances toward the east, the depth of the overburden increases. Consequently, extensive earthworks are required at the southern end wall for dragline access to the topography.

Northern Area: The dragline commences the first pass from the central ramp and mines south. At the end of its first pass, the dragline ramps down to the main pass level and walks back to the central ramp and strips in the main pass from the central ramp towards north. On completing the main pass, the dragline can then ramp out to the topography surface at the northern end of the pit as a result of shallower overburden depth in this part of deposit. This pattern relatively reduces the dragline access road earthworks in the northern area.

Volumetric Calculations: Referring to Figure 10.29, the volumetric calculations for the proposed modified split bench method can be divided up into seven components depending on the dragline positions and pit geometry. In a normal situation, the dragline mining sequence is as follows:

1. Excavate a key cut of one and a half bucket width on the first pass. The dragline dumps the key cut material as far as it can. Some rehandle (Pass 1 rehandle in Figure 10.29) may be involved in this stage due to limitation in the reach of the dragline, particularly in wide strips.
2. Continue to remove the main cut in the first pass. The material from this pass (Pass 1) is used to form the bridge in the main pass.
3. After completing the first pass, the dragline moves to the main dig level and excavates a key cut of one and a half bucket widths in the main pass. The key cut material is used to complete the bridge. Due to the coal thickness in this deposit, in most cases the key cut width must be increased to provide enough material to build the bridge.
4. Continue to remove the main cut of the second pass and the remaining old bridge into the four segments (Main 1 through Main 4 in Figure 10.29).

To run the dragline simulation with CADSIM three sets of input parameters are required. The first set is a geological model which was generated from a massive amount of exploration data. The other two sets of parameters were strip and material parameters (Table 10.6), and the dragline specifications (Table 10.7) which were provided by the mine personnel.

Table 10.6- Strip and material parameters used for the dragline simulation.

Parameter	Value
Highwall angle (deg)	75.0
Prestrip chop angle (deg)	63.0
Key cut angle (spoil side) (deg)	63.0
Spoil angle (deg)	35.0
Spoil undercut angle (deg)	45.0
Spoil flat top (m)	10.0
Spoil swell factor	1.12
Post blast swell factor	1.07
Depth of undercut trough (m)	5.0
Prestrip berm width (m)	25.0
Prestrip highwall angle (deg)	63.0

Table 10.7-Dimension terminology of the BE 1350W.

Parameter	Dimension
Dig depth (m)	45
Dump height (m)	30
Dump radius (m)	87
Tail clearance (m)	25
Walk road width (m)	40
Bucket capacity (m^3)	45

10.3.4 Dragline Pit Optimisation

The current digging method involves many strip geometry factors which are defined by the geology such as overburden thickness or coal thickness and coal dip. Only a few operational key parameters can be varied to conduct a sensitivity analysis and hence optimising the pit configuration. The most important of these parameters are strip width and the dragline working levels.

10.3.4.1 Strip Width

Two strip widths of 70 and 80m were proposed by the mine management for this case study as alternative options. These two cases were simulated for both the southern and northern areas separately. Digging depths for both main and the first pass were kept the same for the two cases. The level of the dragline working level in the main pass was 40m (maximum dragline digging depth) for both mining areas. The first pass digging depth was variable and in southern area started from 0 and increased at 5% dip to the 25m level above the main pass dig level. A 25m depth is the maximum possible depth for the first pass due to the spoil room available. In the northern area the first pass digging level in both cases was kept at 0m in the first 150m of each strip to pass the effect of the coal access ramp on the spoil room. The first pass digging depth then started at 25m and remained constant for the rest of strips toward the northern end wall. The effect of dragline digging depths on the dragline operation is discussed in more detail in the following section. Tables 10.8 and 10.9 summarise results of the simulation for the two strip width cases. The effect of changes in strip width on the rehandle percentage is also depicted in Figures 10.30 and 10.31.

Table 10.8- Summary of the simulation results in the northern area.

Strip Width (m)	Rehandle (%)				Productivity (bcm/hr)		Annual Productivity ($10^3 \times$ bcm/y)	
	Pass 1	Bridge	Ramp	Total	Prime	Total	Prime	Total
70	0.3	26.4	12.1	38.9	994	1381	6.235	8.660
80	2.0	23.8	13.4	39.2	984	1359	6.134	8.520

Table 10.9- Summary of the simulation results in the southern area.

Strip Width (m)	Rehandle (%)				Productivity (bcm/hr)		Annual Productivity ($10^3 \times$ bcm /y)	
	Pass 1	Bridge	Ramp	Total	Prime	Total	Prime	Total
70	4.3	26.8	3.8	34.8	1079	1424	6.619	8.926
80	5.1	22.6	4.3	32	1037	1400	6.644	8.771

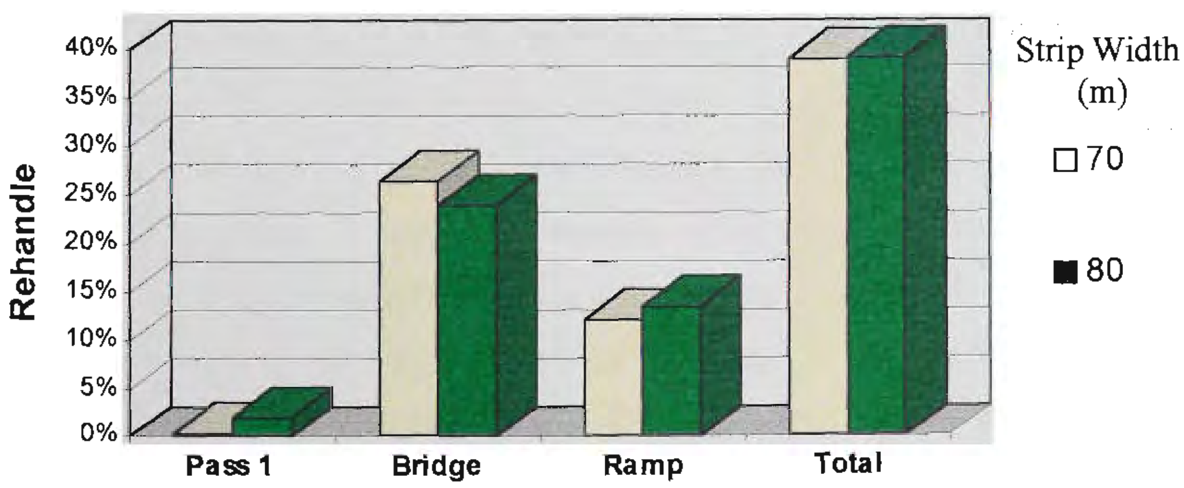


Figure 10.30- Rehandle figures for the two strip widths in the northern area.

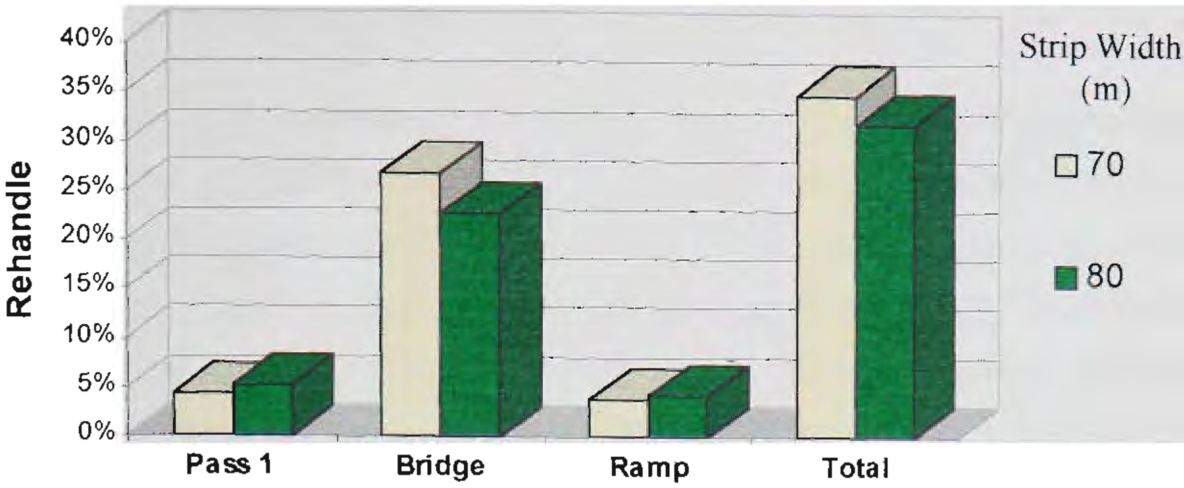


Figure 10.31- Rehandle figures for the two strip widths in the southern area.

The results show that total rehandle is almost the same for both strip widths in the northern area. This is because the reduction in the bridge rehandle for the 80m strip width case is offset by the ramp rehandle since the material from the ramp must be carried over a longer distance for a 80m strip width than a 70m case. Later simulation runs to study the effect of digging depth in the northern area showed that the amount of rehandled material due the ramp can be reduced (up to 10%) by changing the first pass digging depth configuration. Pass 1 rehandle remains the same for the two strip width cases.

As with the northern area, the results are marginal for the two strip width cases in the southern area. There is a slight reduction in total rehandle (2.8% less) for an 80m strip width. This is mainly due to a reduced bridge rehandle as a result of using a wider pit. The prime productivity and, hence, the annual prime moved by the dragline is higher for the 80m strip width since the reduction in rehandle is more significant than reduced swing angle in this case. Total productivity is higher for narrower strip widths due to the shorter swing angles. In summary, for both areas the differences in the rehandle and productivity are not very significant and both strip width cases provide almost the same results. In practice wider strips are preferable due to the reduction in the dragline walking time between strips. For example, choosing a 80m strip width for the current pit reduces number of strips from 17 to 15 strips in each area. Wide strips also provide better pit inventory and safer working conditions. A strip width of 80m was selected as optimum pit width for both areas by the mine management.

10.3.4.2 First Pass Dig Level

The dig level in the main pass is fixed and is defined by the dragline maximum digging depth and the depth of the undercut trench. The main pass level affects the spoil room available due to the changes in the dragline dump height. Increasing the main pass level increases the dragline dump height, hence providing more spoil room. On the contrary, increasing the main pass level increases rehandle percentage substantially. The current dragline operation at the mine aims to maximise the amount of overburden removed by the dragline; therefore, increasing spoil room is more important than reducing rehandle percentage for this case study. A value of 40m was used as the maximum dragline digging depth in the main pass due to digging capability of the dragline. This value is measured from the extended bench level to the top of the coal.

The digging depth of the first pass is variable and is controlled by the available spoil room, topography, and the effect of the coal access ramp on the dragline rehandle. The first simulation runs indicated that with the current pit configuration and dragline specifications a maximum of 65m overburden depth can be fitted into the spoil area. Leaving 40m for the main pass, an extra 25m can be removed by the dragline from the first pass. In the area with overburden depth exceeding 65m a shovel and truck system is required to remove the rest of overburden.

The existence of the central coal access ramp causes inadequate spoil room in an area around the ramp. A substantial proportion of the material must be carried along the strip from the sections affected by the ramp (usually six sections in each side of the ramp). This material must be rehandled and was defined as "ramp rehandle". A CADSIM module "SPFINAL" was developed to calculate the rehandle percentage due to the ramp. In this module a reference surface is used to calculate the spoil room available considering the effect of the coal access ramp. The calculated spoil room is then compared with the spoil from each section. If the spoil required is less than the spoil available the extra volume must be carried to the adjacent blocks and to the volume of the next blocks.

Reducing the rehandle percentage due to the ramp (ramp rehandle) was the criterion used in optimising the first pass working levels. Another important task which must be included in the calculation of the first pass digging level is the dragline access to the main pass considering the maximum grade that the dragline can walk. In this case study two design configurations of the first pass working level were considered to be practical for the current pit. These design alternatives are referred as Cases A and B in the following discussions. Figures 10.32 and 10.33 are the schematic north-south (NS) long cross sections showing the alternative cases for the dragline working levels.

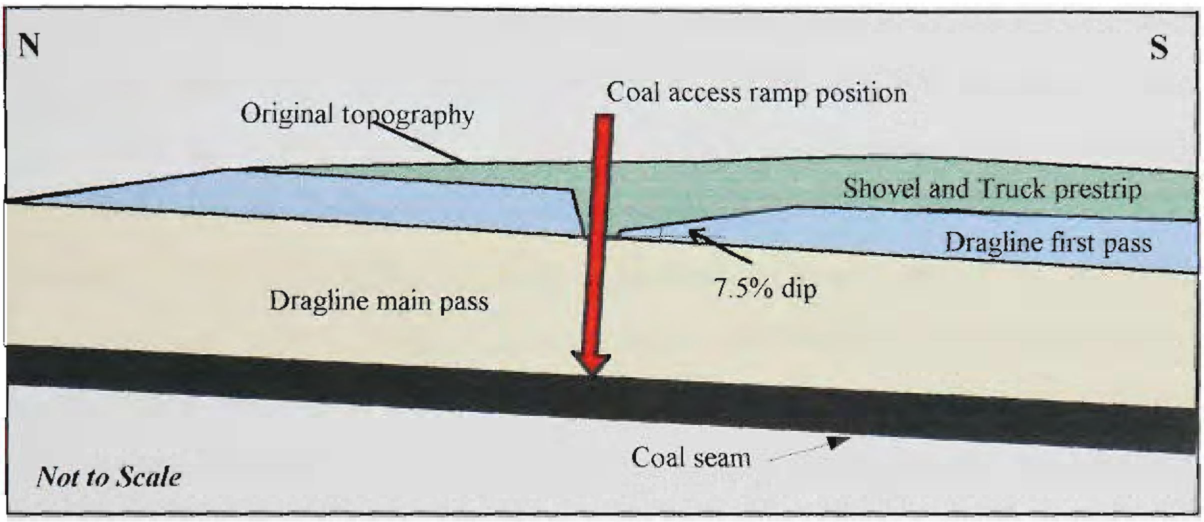


Figure 10.32- A typical long NS cross section of the first dig level (Case A).

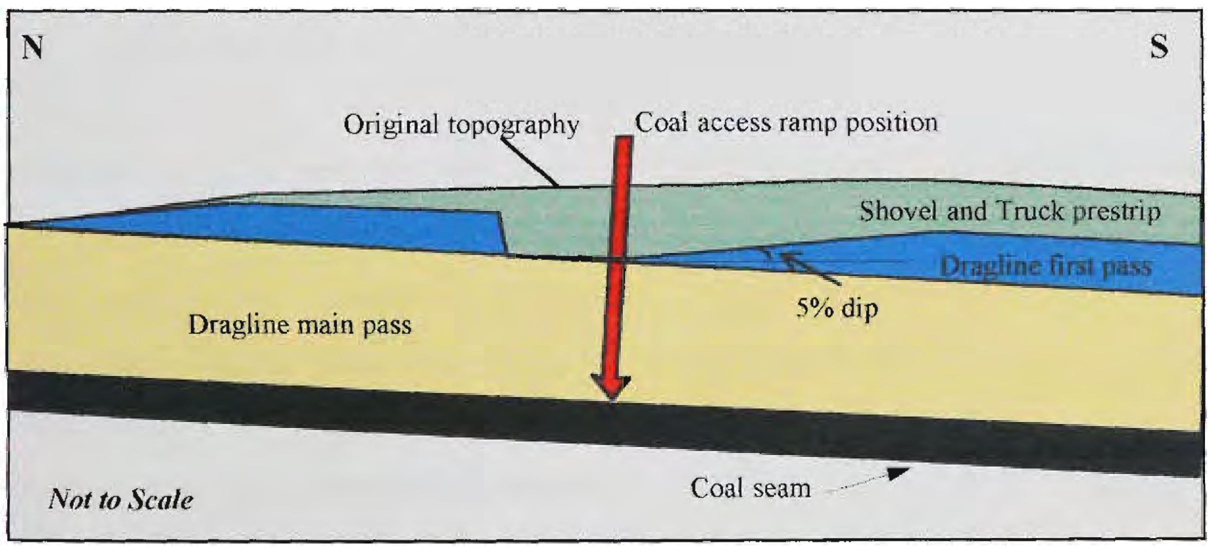


Figure 10.33- A typical long NS cross section of the second dig level (Case B).

Both cases include a dragline access to the main pass in the southern area and have the following design characteristics:

Case A: The dragline working level in the first pass starts from 25m immediately at the ramp side in the northern area and remains 25m except for the northern end wall where the dragline works from the original topography surface. The dragline access to the main pass is provided from the northern end of the pit. The first pass digging depth starts at 0m and ramps up at 7.5% dip (maximum dragline grade) to 25m above the main pass level. This allows the dragline to reach the working level quicker (during the first 200m away from the ramp), hence increasing the total amount of material removed by the dragline.

Case B: The first pass digging depth in the northern area is kept at 0m in the first six sections (150m of each strip) to pass the area affected by the central ramp. This depth would be constant to the north except at the end wall of the earlier strips, where the dragline works from the topography surface. In the southern part, the first pass digging depth starts at 0m and increases at 5% dip to 25m above the main pass level. Since coal seam also dips to the south at 5% , the overall dip is approximately 10% and the dragline reaches to the designed working level within the first ten sections (250m away from the ramp).

Several simulation runs were conducted to evaluate the effect of working level scenarios on the rehandle percentage and the dragline productivity. Initial simulation runs were performed without controlling the first pass working level from an input file while the main pass digging depth was set as 40m. This allows the CADSIM simulator to calculate maximum dig depth for the first pass based on the spoil available and create initial output report files for the calculated volumes and working levels for each block of the overburden for the entire deposit. In the next step the level of the dragline passes were controlled via an input dig level file "DIGREP.TXT" during the simulation runs on the basis of the design configurations for the two cases. Using the "DIGREP" file allows the dragline working levels to be controlled by the user, rather than be calculated by the CADSIM macros based on the spoil available. This process is used when a design is available for the working levels of dragline in different passes. A summary of the results is presented in Tables 10.10 and 10.11 and Figures 10.34 and 10.35.

passes. A summary of the results is presented in Tables 10.10 and 10.11 and Figures 10.34 and 10.35.

Table 10.10- Summary of the dragline simulation in northern area for two dig level cases.

Dig Level Case	Average Strip Length (m)	Rehandle (%)			
		Pass 1	Bridge	Ramp	Total
Case A*	925	4.6	23.8	46.2	74.6
Case B**	925	4.6	25.0	9.3	38.9

** Pass 1 digging depth starts from 25m.*
*** Pass 1 digging depth is 0m in the early 150m.*

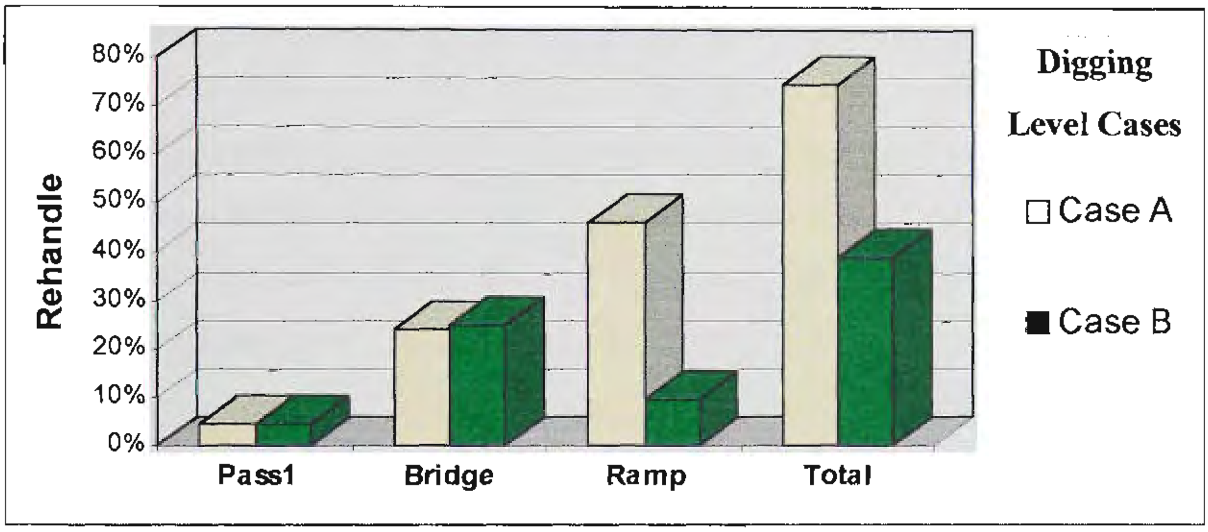


Figure 10.34- Simulation results in the northern area for the two dig level cases.

Table 10.11- Summary of dragline simulation in southern area for two dig level cases.

Dig Level Case	Average Strip Length (m)	Rehandle (%)			
		Pass 1	Bridge	Ramp	Total
Case A*	695	4.9	25.4	12.7	43.1
Case B**	695	5.0	25.5	6.1	36.6

** Pass 1 digging depth starts from 0m and increases at 7.5% dip*
*** Pass 1 digging depth starts from 0m and increases at 5% dip*

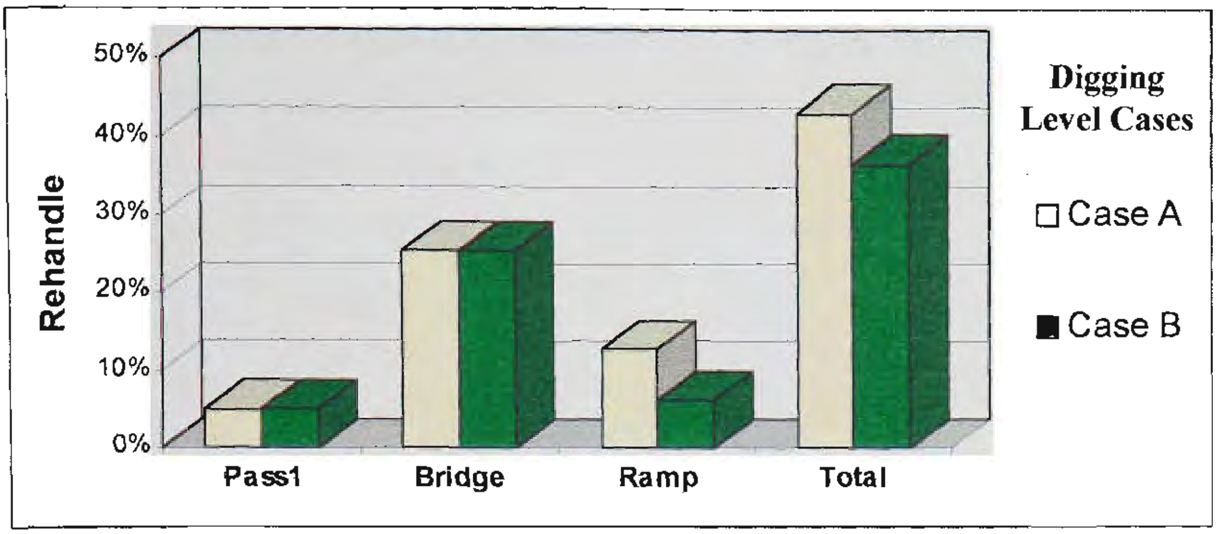


Figure 10.35- Simulation results in the southern area for the two dig level cases.

The results for both areas show that the second option (Case B) is a better option for design of the first pass working level since it reduces the total rehandle by 35.7% (from 74.6% to 38.9%) in the northern area and 6.5% (from 43.1% to 36.6%) in the southern area. The reduction in total rehandle is due to the reduced ramp rehandle as less material must be carried by the dragline in Case B. On the other hand, there is an increase in the amount of material that must be removed by truck and shovel in Case B. The extra material is about 7.8% and 2.5% of the total prime of both passes for northern and southern areas, respectively. According to the information provided by the mine staff, the cost of moving material by the truck and shovel system is between 1.5 and 2 times higher than that of the dragline. This means that if an option reduces the dragline rehandle more than twice of the amount of extra material that must be removed by the truck and shovel, it is cost effective to use such an option. The results suggest that the second option (Case B) provides a more cost effective scenario than the first option (Case A). Case B was used as optimum design configuration for the detailed simulation. The detailed simulation aimed at the productivity estimation for the entire mine life and creating a block by block waste and coal volume information for a long term scheduling program.

10.3.4.3 Simulation Results for Strip 4

Due to the location of a dyke and since a separate access ramp was required for strip 4 (Figure 10.28), it was decided to consider the strip as a special case when optimising the pit configurations. Rehandle figures and productivity of the dragline were calculated for both 70 and 80m strip widths in strip 4. Two cases of different design configurations for the first pass working level were also simulated for this strip. The main design criterion for the calculation of the first pass working level was to reduce the dragline prime production in the area affected by the ramp. In the first case, the first pass digging depth was kept at 0 m in the first 150m of the strip and then ramped up with 7.5% dip to the 25m level above the main working level. In the second case, the first pass digging depth started from 0m and increased at 5% upward to the 25m. The results for these four cases are summarised in Table 10.12. A 3D view of the pit after the simulation of strip 4 is shown in Figure 10.36.

Table 10.12- Summary of the dragline simulation results for strip 4 in the south.

	Pass 1 Dig Level			
	150m @ 0m & Ramps Up		***Start From 0m& Ramps Up***	
	Strip Width			
	70m	***80m***	***70m***	***80m***
Rehandle Percentage				
Pass 1	1.7	3.5	1.6	3.7
Bridge	37.5	33.3	36.7	31.0
Ramp	39.1	50.0	61.9	75.4
Total	78.3	86.9	100.2	110.1
Productivity				
Prime (bcm/hr)	777	738	686	654
Total (bcm/hr)	1373	1379	1373	1374
Annual prime ($10^6 \times$ bcm)	4.870	4.626	4.299	4.099
Annual total ($10^6 \times$ bcm)	8.637	8.645	8.607	8.613

The results of the dragline simulation for strip 4 suggest that for this strip a wide pit is not suitable since there is an increase in the total rehandle mainly due to the increased rehandled material around the ramp. Comparing with the results of the simulation for other strips in the southern area, this strip has very large amount of carried material due to the ramp (40%, in the best case 6%). This is because in a normal case half of the spoil from the ramp area must be carried in either side of the deposit, but in strip 4 all the extra material from the blocks affected by the ramp must be carried along the strip toward the south. Another factor which increases ramp rehandle for this strip is the strip length. strip 4 has one of the shortest strip lengths (575m) among all the strips due to the presence of the dyke and this reduces the dragline prime volume while the carried material remains the same as for other strips.

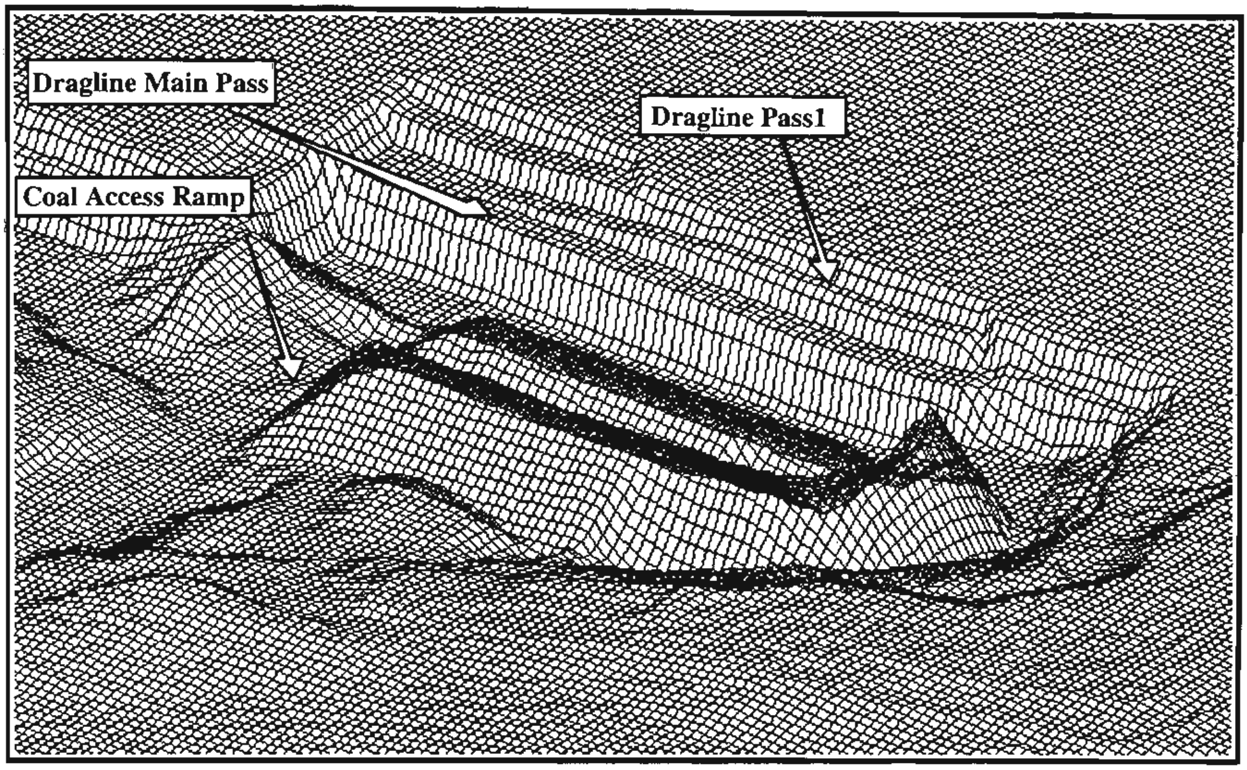


Figure 10.36- Final view of the strip 4 after simulation.

Because of the effect of ramp in this strip, a narrower strip (70m) is preferable as the total rehandle is reduced by 8.6% (from 86.9% to 78.3%). It must be noted that the strip width for the subsequent strips is 80m. Normally, a narrower old pit will reduce the available spoil room, hence increasing the rehandle percentage. This is not a case for the current pit design as the closure of the short ramp in strip 4 provides extra spoil room for strip 6 which can offset the effect of narrow old pit.

10.3.5 Detailed Simulation Results for the Optimised Pit

Detailed rehandle and productivity estimations were performed for all the strips in the southern and northern areas. Following is a summary of the results from the detailed CADSIM runs on the entire deposit.

10.3.5.1 Detail Simulation Results in the Southern Area

Simulation sections for CADSIM were created in the southern area with 25m offset. The strip width chosen for all strips was 80m and the first pass digging depth started from 0m and increased to 25m at 5% dip for all strips except strip 4 as discussed in the previous sections. The first pass dig level in strip 4 is 0m in the first 150m of the strip and then ramped up to 25m. Table 10.13 details a breakdown of the rehandle components and productivity terms for all strips in the southern area. The results are summarised in the associated graphs (Figures 10.37 and 10.38). A general 3D view of the dragline pit in the southern area is shown in Figure 10.39.

Table 10.13- Simulation results of the optimum pit in the southern area.

Strip No	Strip Length (m)	Rehandle (%) Pass 1	Bridge	Ramp	Total	Productivity (bcm/hr) Prime	Total	Annual Productivity ($10^6 \times$ bcm/y) Prime	Total
4	575	3.5	33.3	50.0	86.9	737	1379	4.626	8.645
6	625	3.5	25.0	12.9	41.4	1013	1433	6.350	8.980
8	925	4.6	23.7	4.2	32.5	1053	1395	6.592	8.742
10	900	5.1	22.6	4.4	32.1	1059	1399	6.642	8.801
12	775	5.3	22.8	6.4	34.5	1040	1399	6.520	8.613
14	750	5.1	24.0	7.0	36.1	1016	1383	6.368	8.565
16	725	4.2	28.0	6.0	38.2	1017	1405	6.371	8.671
18	700	4.2	28.6	6.2	39.0	1008	1401	6.353	8.780
20	675	4.9	27.4	6.8	39.1	1004	1397	6.298	8.754
22	650	5.3	26.8	7.2	39.3	1002	1396	6.280	8.735
24	625	5.4	26.4	7.1	38.9	996	1384	6.246	8.701
26	575	5.4	25.7	6.1	37.2	1010	1386	6.330	8.792
28	550	5.2	24.8	6.2	36.2	1016	1384	6.370	8.740
Average	**695**	**5.0**	**25.5**	**6.1**	**36.6**	**1002**	**1395**	**6.214**	**8.713**

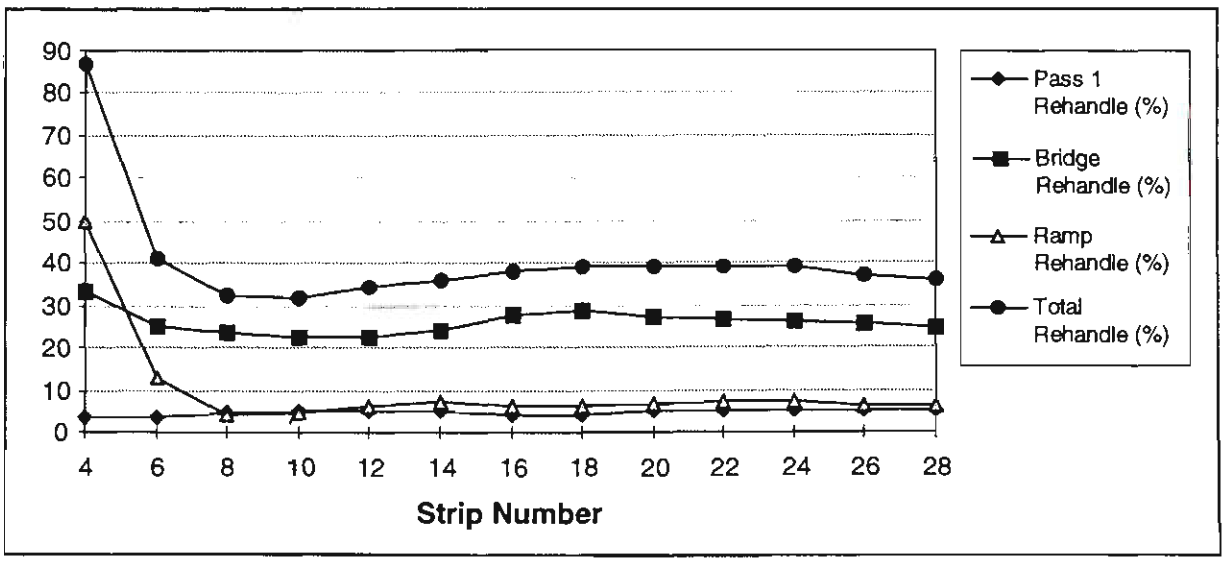


Figure 10.37- Rehandle components for the optimum pit in the southern area.

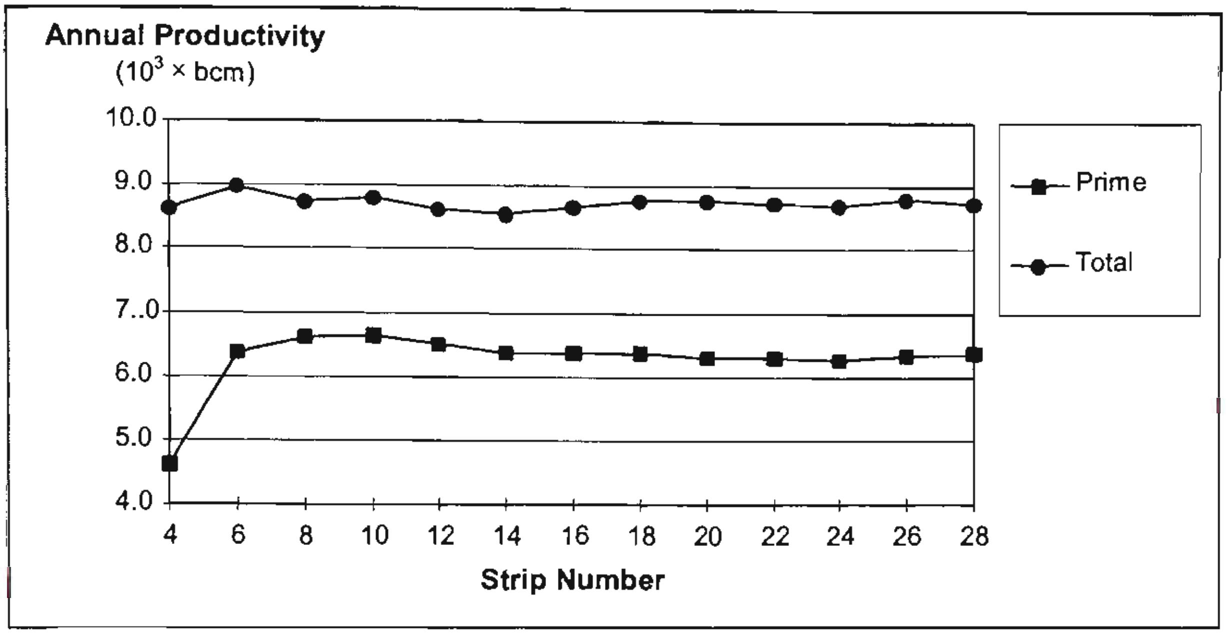


Figure 10.38- Annual productivity estimation for the optimum pit in the southern area.

10.3.5.2 Comments from Simulation in the Southern area

Some of the findings from the dragline simulation results using CADSIM in the southern area presented below:

1. Strip 4 has the highest rehandle figures and, as a result, the lowest prime productivity compared with other strips. The main reasons for this are the short length of this strip and the use of a separate ramp.
2. There is a significant reduction in rehandle percentage for strip 6 compared with strip 4. Strip 6 contains a dyke and, hence, a shorter pit. In addition spoil room shortage at the southern end wall causes additional rehandle for this strip. However, there is an additional spoil room in the middle of the strip due to the closure of Strip 4 ramp.
3. A steady state rehandle percentage and productivity is reached after strip 8.
4. Strips 8 and 10 have the lowest total rehandle mainly due to the reduced rehandle due to the ramp. The prime productivity is high in these two strips because of rehandle reduction and relatively longer strip length.
5. The first pass working level starts from 0m at the ramp side and ramps up with 5% dip resulting in a ramp rehandle about 6 to 7%.
6. There is a constant reduction in the strip lengths as the mining advances to the east. This slightly increases the rehandle due to the ramp.

7. Because of the increased overburden depth at the eastern side, the dragline access to the topography becomes a difficulty. This requires substantial earthworks for access ramps and sharp increases in the prestrip volumes.

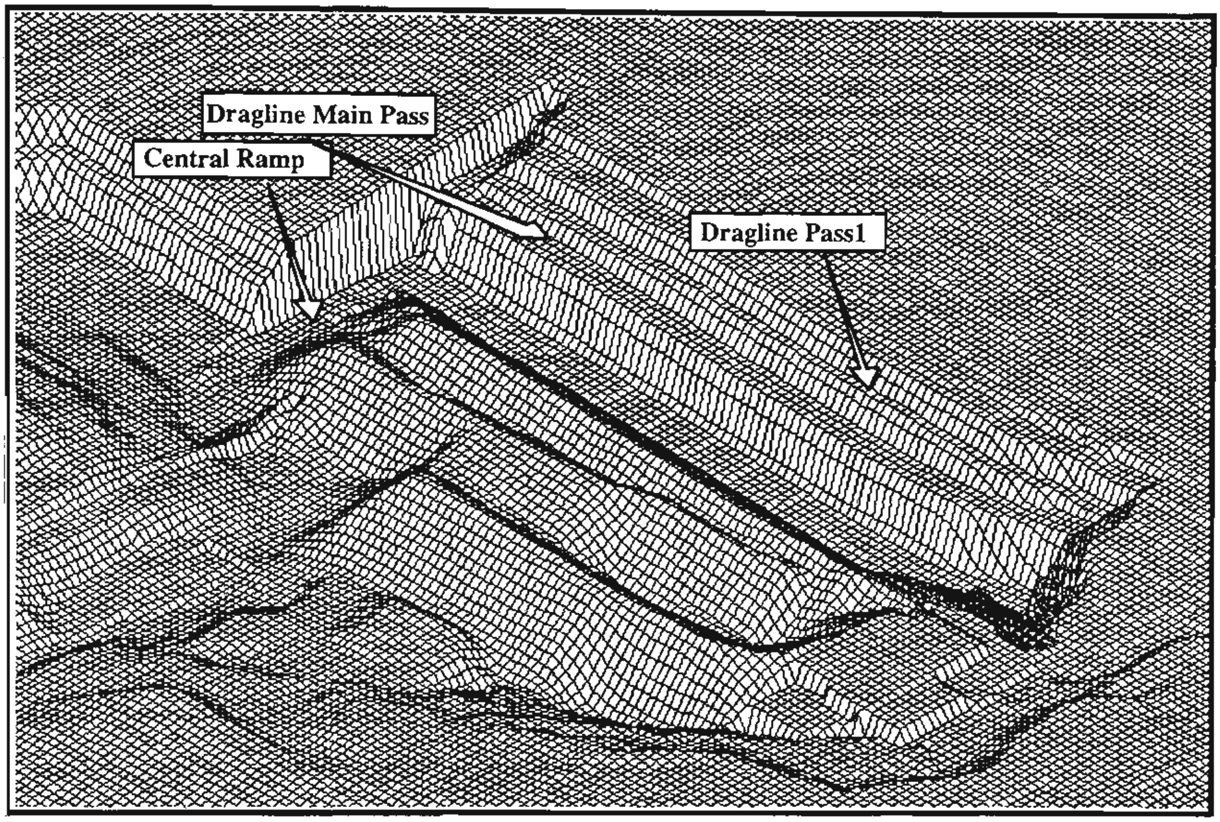


Figure 10.39- A general view of the dragline pit in the southern area after simulation strip 8.

10.3.5.3 Detail Simulation Results in the Northern Area

Two sets of parallel and radial sections were created in the north to cater for the strip bend. Strip widths were 80m except near the bend where widths are slightly more than 80m. The first pass digging depth is 0m in the first 150m and 25m for the rest of the strip, until it reaches the topography surface. A general 3D view of the dragline pit in the north is shown in Figure 10.40. Table 10.14 details rehandle components and productivity for all strips in the north. The results are graphically summarised in Figures 10.41 and 10.42.

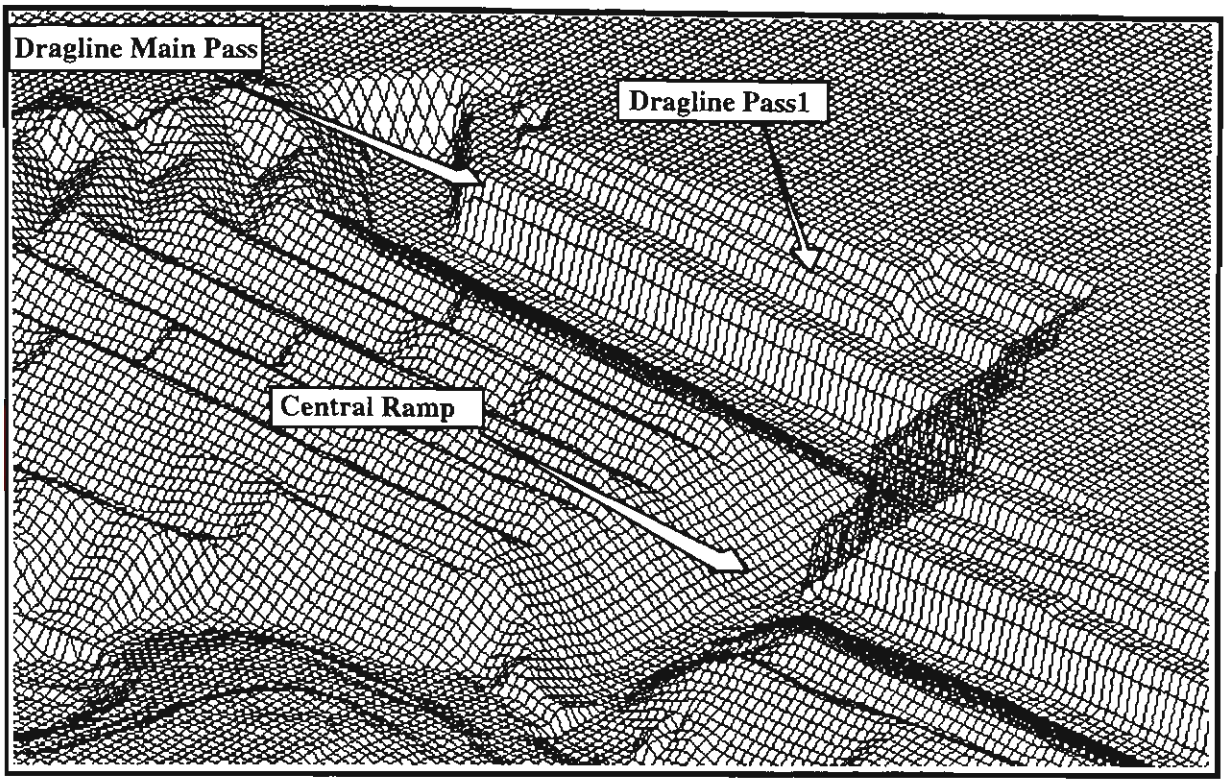


Figure 10.40- A general 3D view of the dragline pit in the northern area after simulation strip 15.

Table 10.14- Simulation results of the optimum pit in the northern area.

Strip No	Strip Length (m)	Rehandle (%)				Productivity (bcm/hr)		Annual Productivity ($10^6 \times bcm/y$)	
		Pass 1	Bridge	Ramp	Total	Prime	Total	Prime	Total
5	990	2.7	27.2	0.1	30.0	1106	1438	6.933	9.013
7	925	5.1	25.1	20.2	50.4	905	1361	5.674	8.533
9	850	5.2	23.8	15.6	44.6	949	1372	5.947	8.600
11	875	4.8	24.1	9.0	38.0	1016	1402	6.367	8.787
13	825	4.6	24.7	4.0	33.2	1071	1427	6.714	8.943
15	850	5.3	25.4	6.6	37.3	1018	1398	6.383	8.764
17	860	4.6	24.3	5.7	34.6	1100	1480	6.146	8.273
19	1240	3.2	26.5	50.3	80.1	847	1526	4.755	8.564
21	1150	3.9	28.5	0.9	33.3	1151	1534	6.461	8.612
23	1060	3.7	27.8	1.0	32.4	1126	1491	6.301	8.343
25	980	5.4	25.0	1.7	32.1	1058	1398	6.634	8.763
27	890	5.1	24.8	4.1	34.1	1001	1343	6.277	8.418
29	770	4.9	25.2	3.2	33.2	1011	1347	6.336	8.439
Average	**925**	**4.47**	**25.64**	**9.93**	**40.04**	**1026**	**1427**	**6.200**	**8.608**

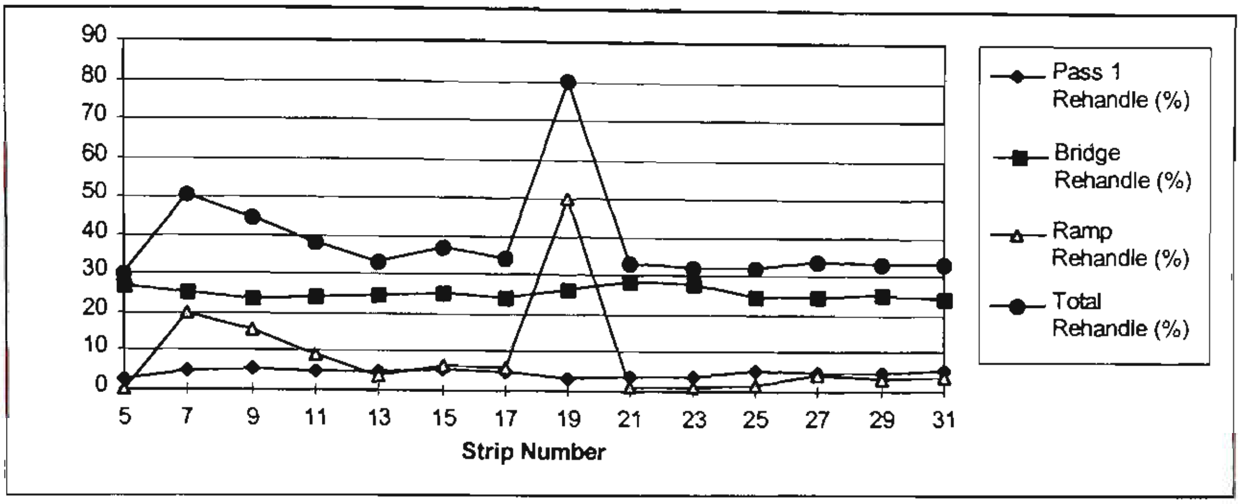


Figure 10.41- Rehandle components for the optimum pit in the northern area.

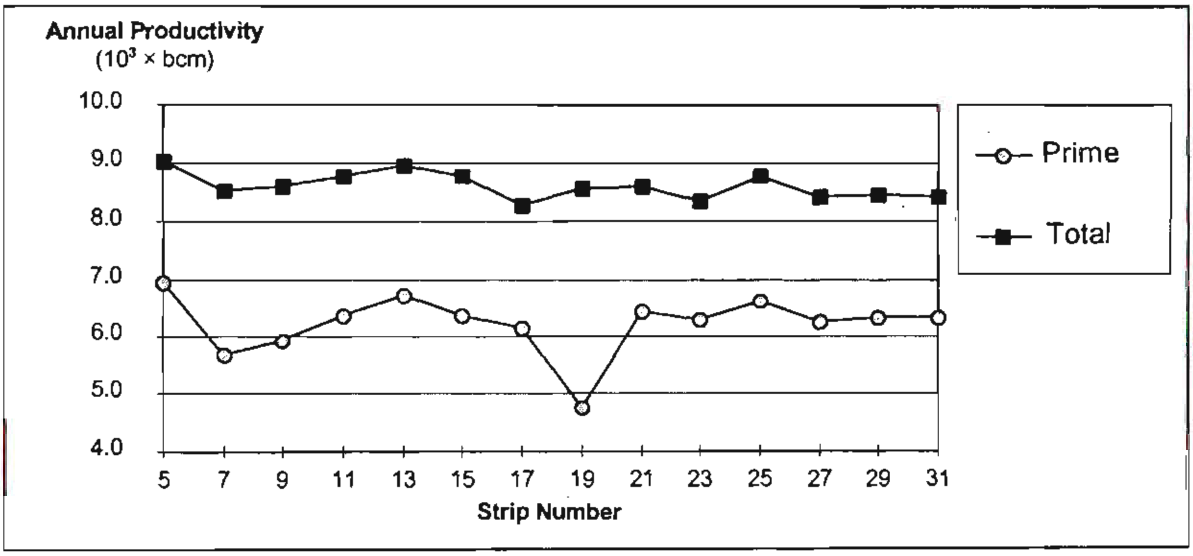


Figure 10.42- Annual productivity estimation of the optimum pit in the northern area.

10.3.5.4 Comments from the Dragline Simulation in the Northern area

The findings from the dragline simulation using the CADSIM model in the northern area are presented below:

1. Strip 5 has the lowest total rehandle and, hence, the highest prime productivity of all of the strips. The total rehandle is low in this strip because of the reduction in ramp rehandle. The dragline has enough spoil room in this strip as a result of a wide old shovel and truck pit.
2. The increase in ramp rehandle for Strip 7 and to some extent in Strip 9, is due to the changes in available spoil room in the first 300m of these strips. The main

reason for this is the coal seam dip and the coal roof changes that reduce the dragline main dig level and hence the effective dump height.

3. A possible way to reduce rehandle in StripS 7 and 9 would be to reduce the first pass digging depth to a maximum of 22-23m.
4. The increased total rehandle in Strip 19 is due to a short old pit. At the end of Strip 19 a box cut is required and some material should be removed by truck and shovel.

10.4 SUMMARY

A study was carried out to demonstrate and test the capabilities of the CADSIM model developed in this thesis by applying it to two real mining operations. In the first case study the variations occurring in the different operating parameters were examined for the three digging methods, Standard Extended Bench, In-Pit Bench and Extended Key Cut to find the optimum configurations. The results of the study indicated that an average productivity of 17.9 million bcm per annum would be achieved by employing the Standard Extended Bench method during the first six years. Alternatively, employing an In-Pit Bench method results in an average productivity of 19.3 million bcm per annum. But the highest prime and annual productivity is achievable using an Extended Key Cut method which can result in a 22.7 Mbcm per annum.

The study aimed at optimising pit design showed that a very wide pit suitable for the Extended Bench method is not desirable for the current pit. The maximum prime productivity is obtained with a pit 100 to105m wide. The study also suggests that increasing the width of the pit to more than 105m has an adverse effect on the dragline operation and will reduce its prime productivity due to the higher swing angles.

Applying the CADSIM model to the second case study demonstrated the capabilities of the model in evaluating the effect of coal access ramps and dragline dig levels. Strip width and dragline dig level in the first pass were changed and the effect of these changes were analysed to optimise pit configurations for the second case study. Reduction in rehandle and, hence, improving productivity of the dragline was the main criterion in this study.

CHAPTER ELEVEN

SUMMARY AND CONCLUSIONS

11.1 SUMMARY

Increased overburden depths and more complex geological conditions adversely affect efficiency of the strip mining equipment. Many operations report that the productivity of a dragline operation can be increased by modifications to the digging methods, optimised pit geometry and better planning for the strip layout and location of access ramps. Selection of an appropriate digging method and optimisation of strip layout and pit geometry for a dragline operation require that a number of options be examined based on consideration of the geological factors and dragline size. Optimising the operating parameters of a pit requires an evaluation of a range of options. With the use of computer based techniques such as 3D CAD, it is possible to study a large number of options with a high degree of reliability to obtain an optimum plan for the mining operation.

A computerised dragline simulator (CADSIM) has been developed during the course of this thesis, and this can be used in evaluating different dragline mining scenarios so that an optimum solution can be identified and selected for a given geology. The success of this study has been demonstrated by the use of the model as a planning tool by three large dragline operations both in New South Wales and Queensland coal mines. The results from these case studies showed that the CADSIM model could be used to determine the optimum strip mine design for a given operation.

The CADSIM dragline simulation model incorporates the current capabilities of computer technology to improve the efficiency and accuracy of the planning of a dragline mining operation. The procedure used for the simulation of a dragline operation is an automated approach that permits various options be evaluated quickly and thoroughly to identify the best option. An emphasis was placed on changes in dragline digging methods as the most efficient way to improve stripping capabilities of a given dragline operation.

As the first step in the model development a computer based geological model was established. Topographical maps and drill hole data were used to generate a gridded seam model. The geological model used for this thesis stores data as a series of two-dimensional grids or triangles which is the most efficient modelling technique for strata type deposits such as coal. In a gridded seam technique the combination of grids and triangles allows all of the required features to be modelled. Operational features such as existing spoil and cut surfaces are well represented as triangles while regular, undisturbed coal roof and floor surfaces can be represented using grids. Geological sections can be constructed by intersecting the plane of the section and grids or triangles which represent different mining layers in that section. The resultants of this intersection are strings which define the geological condition of the gridded surfaces in each section. The coordinates of the strings are then stored into ASCII files to be used during the simulation of the dragline digging methods.

A survey conducted as a part of this thesis showed that most open cut operations are considering innovative and modified mining methods in order to cope with the complex geological situations. From the information received, seven digging methods were identified as representative of most of the Australian dragline operations. These methods, along with their sub-applications, were coded in DSLX language to provide a library of stripping methods which can be used for the purpose of dragline digging method selection or evaluation of several stripping methods for a given geology.

The process of the dragline simulation for this study is a three step procedure as follows.

1. reading input data by the CADSIM model to reproduce the geology of the sections and establish the initial pit design,
2. simulating the dragline operations, cut and spoil sequences, calculation of volumes and dragline positions, and
3. generating 3D graphic images outputs and reports containing volumetric data for both prime and rehandled, swing angle and hoist distance data.

The process of simulation commences with retrieving critical strings from the ASCII files created in the geological modelling phase. The location of critical points such as the toe and crest of the old and the new highwall was then used to form the initial design for dragline and truck and shovel benches. In the next step, an excavation method is defined as a sequence of steps through the use of CAD functions. The simulation of cut and fill operations was carried out by a developed simulator CADSIM using DSLX macros. The fundamental logic of the model is working with strings and points to generate cut and fill profiles. This approach automates every movement process and displays the resultant cut and spoil geometry, while managing volume calculations. The combination of different cut and spoil procedures in a logical sequence was used to mimic the removal of a block of waste material by the dragline. Once defined, a digging method can be replayed as a full simulation on multiple sections. It is also possible to produce the final 3D surfaces that is a result of integration of final spoil profiles after simulation of a pit.

This new approach, used in the CADSIM model, links the geological cross sections and transfers the information such as material being carried and dragline dig depth between the geological sections. The use of this approach allows most of problems associated with the traditional 2D range diagrams be solved. The most common design problems which can not be included by the traditional 2D range diagram approach are:

- effect of coal access ramps on rehandle and design of pit geometry,
- effect of curved strips on the spoil room available and actual prime volume, and
- effect of dragline gradeability and grade control between sections along the strip.

During the dragline simulation of the cut and fill processes by CADSIM, all volumes of material moved from sub-components (eg. top of key cut) along with the associated swing angles and hoist distances are progressively written to a report file. The report file can be formatted so that the data are readily imported to the other data analysing packages such as spreadsheets. The imported data are then used in the productivity calculations and further analyses. This valuable block by block information on the whole deposit may also be used for other strip mine planning processes such as detailed scheduling programs. In addition to the report files, a 3D graphic output of the updated surface after the dragline simulation may be produced by the CADSIM model at any stage.

A field time study using more than 100,000 cycles provided the data required for productivity analysis stage and also to validate the CADSIM results. The monitoring data were statistically analysed for different dragline activities such as fill time, swing angle and swing time, etc. The time study results were used in a spreadsheet to calculate the productivity of the dragline simulated operations.

When estimating a dragline productivity the two major types of data required were volumetric and time based data. The volumetric information including rehandle was obtained from the dragline simulation phase. The calculation of cycle time involved the estimation of elements such as swing and hoist time components from the dragline simulation and fixed elements such as filling and dumping times from field time study. Correction and adjustment factors may also be applied to the estimated values, depending upon mode of operation and operator skill. The computed cycle time was based on the swing or hoist dependency of the individual cycle. Different productivity terms such as prime, total and annual productivity were computed for comparison purposes. However, the key element in the productivity analysis was the mine prime productivity which can be defined as a measure of coal exposure rate.

Traditionally, productivity calculations rely on single-value estimates for a number of operating parameters. The results of the time study conducted in this thesis showed that some cycle time components such as filling and dump time were not deterministic and were not related to parameters which could be calculated with a dragline simulator. When uncertain input parameters are involved, a probabilistic approach offers the advantage of quantifying the risk associated with the uncertainty of the input variables, hence permitting better decision making. A Monte Carlo method was used to predict a probability distribution of the prime and total productivity. The risk analysis results on a case study showed that prime and total productivity could vary within a considerably wide range due to the variability of the operational parameters.

The process of selecting an optimum dragline digging method for a given geology is incomplete without performing a financial analysis when various techniques with different costs are involved. The process of cost estimation was generally conducted in this thesis for comparison purposes on the basis of either cost per hour, cost per bcm, or cost per tonne of coal exposed. The relative stripping and mining costs for the proposed digging methods were evaluated in terms of:

- drilling and blasting requirements,
- total dragline operation, and
- auxiliary equipment and services.

A Discounted Average Cost (DAC) method was used to compare alternative digging methods. The DAC method is more suitable for decision making processes as it does not include the revenues from the coal which tend to bias the decision.

11.2 CONCLUSIONS

Three case studies of various coal deposit types were used to validate the CADSIM's results and to illustrate the major capabilities of the model. The major applications of CADSIM tested in this thesis were:

1. dragline digging method selection based on the costs of the operation,
2. optimisation of the dragline pit dimensions and configuration, and
3. evaluation of the coal access ramp position and strips layout.

The main objective for the dragline simulation in the first case study was to validate the CADSIM results. The mine was a multi-seam dragline operation utilising a single highwall, double lowwall pass method. As a measure of the ability of the model to simulate mining operations and generate reliable results, a comparison was made with the actual data captured by a dragline monitoring system. The comparison showed that the CADSIM model was able to predict most of the operational parameters within an acceptable range (less than 5%).

In addition to validation of the model, the results of the first case study indicate that:

- In a multi-seam dragline operation the thickness of overburden and interburdens is the main factor in selecting an optimum digging method. Other important factors also worth considering are the dragline size and pit geometry.
- There is a clear difference between most of the operating parameters in the highwall and the lowwall side mining and the dragline performance is considerably affected by the mode of operation.

The purpose of the second case study was to illustrate the CADSIM's ability to simulate different digging methods and the selection of the most cost effective method. The strip mine was a multi-seam operation, but only the last interburden was allocated to a new dragline. The coal thickness varies between 2 to 7m with an average of 5m and the interburden thickness ranges between 25 to 57m with an average of 48m. Three digging methods were simulated in detail for two separate areas. The productivity and cost of each method were estimated and compared to allow selection of the most efficient method for dragline operation. The results showed that the Extended Key Cut method was the most productive method followed by the In-Pit Bench method, while the standard Extended Bench method was the cheapest method for waste removal.

The results of the second case study indicate that:

- The selection of the most appropriate digging method for a dragline operation depends on many factors including the geology of the deposit, dragline size, production requirement and cost per bcm of waste moved.
- Although in most cases the use of an advance bench causes a longer swing angle and reduced bucket fill factor, this could reduce the average depth of the dragline working bench in the mine and consequently a considerable reduction in rehandle material.
- The optimum pit geometry depends on the digging method.
- The dragline size has a significant effect on the selection of a digging method and the optimum pit geometry. Digging methods with higher rehandle, but shorter swing angle (eg. Extended Bench method) may be more suitable for a dragline with a very large bucket (eg. greater than 75 bcm) and a medium boom length. Conversely, a digging method with less rehandle but larger swing angle (eg. In-Pit Bench chop cut) are more suitable for a dragline with medium size bucket and longer boom length.
- The most productive digging method may not necessarily be the cheapest one, particularly when different drilling and blasting techniques with different costs are involved.
- The throw blasting technique can significantly improve productivity but it also increases operating costs mainly due to the increased drilling costs and powder factor.

The intention in the third case study was to test the capability of the CADSIM model to simulate a whole deposit while investigating the effect of different mine design features on dragline operation. The parameters considered were strip layout, the existence of disturbing structures such as dykes or basements, coal seam dip, pit dimensions and the location of coal access ramps. The type of deposit for this case was a single thick coal seam (20 to 25m) overlaid by a deep overburden (45 to 65m).

The results of the third case study also indicate that:

- When using an Extended Bench method to uncover a thick coal seam, the dragline reach becomes more critical than the dragline dump height. In such a case the use of digging methods with an in-pit bench lower than the original level may be more appropriate.
- The effect of coal access ramps on the available spoil room is more substantial in the case of a thick overburden. In the case of a two pass digging method, rehandling due to the ramp can be minimised by optimising the depth of the first pass (or chop bench).
- Using a coal trench to clear the edge of a thick coal seam during dragline mining can reduce the dragline's maximum digging depth in the main pass.

All the case studies showed that the geological conditions have a major impact on the selection of an appropriate strategy for a dragline operation. As the geological conditions are unique to each coal deposit, generalisation of the results from the case studies to all dragline operations is inappropriate. This emphasises the need to treat each operation as a unique case and the process of the dragline simulation must be performed thoroughly for each case. Finally, testing the CADSIM model in the various case studies clearly showed that the simulation model developed in this study was able to provide valid and useful information for selection of optimum strategies for a strip mine planning scheme.

11.3 RECOMMENDATIONS FOR FURTHER STUDIES

This thesis has provided and promoted advances in overburden removal system by using an innovative approach but there is still scope for further work in this area.

1. A significant improvement can be made by applying this process to 3D blocks rather than cross sections. Gridded surfaces such as a topography grid can still be used for the dragline simulation and its volumetric calculations.

2. The process of creating 3D outputs from the dragline simulation is not fully automated and it is time consuming. A possible means of improvement of the 3D graphic outputs is to automate the process. The automation process can be accomplished by adding an option menu to the CADSIM main menu to allow access to the ASCII outputs from the dragline simulator in a standard format, processing the data to final stages to create the gridded surface and output graphics.

3. This thesis has emphasised the lack of detailed knowledge which exists for dragline operations, especially in the area of digging method characteristics. There are certainly many other mine site specific factors which can be looked at. There is a need to expand the work to gather more detailed information.

4. Further work must be done in the area of time study to realise the potential which dragline monitors have for improving productivity. This study looked at statistics of different cycle time variables over a period of time and compared results from two different operating modes. The first step toward a more detailed analysis of data is to make sure that each part of the cycle time is evaluated on the basis of a consistent digging method. It is also useful if elements of the overburden removal are to be compared in much more detail so that the specific causes of sub-optimum performance can be identified. To determine areas of productivity loss, a much more detailed approach is required, whereby the individual parts of the mining block such as chop, key cut and rehandle digging are examined and their effects analysed.

5. Dragline monitor data can also be used in the estimation and evaluation of the delay times for a dragline operation. The delay times can be both scheduled and unscheduled shutdowns. Such data are very useful in annual productivity calculations and for long term planning of a mine. To accomplish this task, data must be collected and processed over a longer period of time (at least 4-5 years). In order to identify the source of operational delays, it is also important to make sure that the correct monitoring codes have been entered by the dragline operators for each delay events.

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APPENDIX A

"DSLX'S LANGUAGE SYNTAXES AND FUNCTIONS"

A.1 VARIABLES IN DSLX

DSLX allows the user to manipulate strings or points and differentiate between these types of data by having different types of data variables available. DSLX uses five different types of variables for different operations. These five types of variables are 1) Scalars, 2) Points, 3) Arrays, 4) Strings, 5)Tables. The various types of data variables, how they are declared and accessed is described in this section.

Scalar Variables: Consider an operation such as swelling a volume to calculate what spoil room space is required. Assuming that the volume is calculated then a bulking factor or swell factor is required. The bulking factor in this case is considered to be a constant for the project and a statement such as SWELL =1.35 could assign the value to the variable SWELL. SWELL is called a scalar. Scalars are frequently constants. For example SWELL may never change. However this is not always true SWELL could change with area thus we could say:

Example A.1:

```
IF STRIP>= 10
SWELL = SWELL +0.1
ENDIF
```

Other scalar variables could be dragline dimensions, geotechnical parameters such as spoil and highwall repose angles.

Array Variables: In the previous example the swell factor was assumed constant. However, it is frequently true that the swell factor varies with material type. For example alluvial material on top of hard rock has a lower swell factor than hard rock. In such cases the swell factor may be considered to vary by the layer. In a particular simulation exercise the user may have three layers e.g. overburden, ROCK1 and till ROCK2. Three bulking factors could be set up using scalar variables such as SWELL1 = 1.1, SWELL2 = 1.3, SWELL3 = 1.35. In this case an ARRAY could be defined as an array of three scalars, for example, SWELL1, SWELL2, SWELL3.

In the example shown below an array of 16 is set up to store colours. This array is then allocated colours. So array COL1 is colour 1 COL2 is colour 2, etc.

Example A.2:

```
GLOBAL X1,COLP
GLOBAL_ARRAY COL[16]
X1 =0
REPEAT
        X1=X1+1
        COL[X1] =X1
        COLP =COL[X1]
        PRINT " COLOUR INDEX,COLOUR " X1,COLP
UNTIL X1>=16
```

Point Variables: Coordinates or points on a cross-section such as the crest point of the bench or the apex of the spoil peak are defined as the point variables. Points in DSLX have coordinates in X along section and Z vertically. Each point can therefore be considered a coordinate of the form [X,Z]. The coordinate [0,0] is the origin of the section at the lower left coordinate with X increasing to the right and Z increasing to the top of the section. Point variables can be computed by various DSL's routines, by intersection of strings or can be allocated numbers based on calculations within the program. In the example below P1 is defined by giving the X and Z values and P2 is calculated by projecting P1 at an angle (37^{o}) for a distance (DIST = 50m).

Example A.3:

```
GLOBAL_POINT P1,P2
LOCAL  DIST
DIST = 50
P1 = 0,0
P2 = P1 + {37} * DIST
PRINT "NEW VALUE" P1,P2
```

String Variables: String variables are essentially a list of points which are connected in sequence and thus define lines. A string variable can have two or more points and can be created in many ways using DSLX's' routines. String variables are used to describe geological surfaces such as topography and the coal surfaces; mining surfaces such as the cut surface and the spoil surface; and man-made surfaces such as key cut. Many of the built in functions of DSLX relate to handling of string intersections, string volumetric, and other string calculations. In the Example below, the first string Str1 is created by concatating two points P1 and P2 and the second string Str2 is created by concatating points and the first string.

Example A.4:

```
GLOBAL_POINT P1,P2
LOCAL_STRING Str1[2], Str2[5]
P1 = 0,0; P2 = 10,100; P3 = 25,120
P4 = P1 + {45} * 70; P5 = P3 + {45} * 70
Str1 = P1//P2
Str2 = P3// Str1//P4//P5
```

Table Variables: Table variables are essentially lists of strings. They have three dimensions as shown in the example below. The first dimension is the number of strings the table can hold. The second dimension is the number of points each string can hold. The third dimension is always 2, and refers to the number of scalars each point can hold (X and Y). Table variables are used to manipulate multiple surfaces, such as strings representing topography, roof of coal, floor of coal, etc.

In the following example, first geology information of a section (S1) is loaded to a table variable (SECT) and then different surfaces (strings) are extracted from the table.

Example A.5:

```
GLOBAL_TABLE Sect [3][10][2]
GLOBAL_STRING Topo[10], Roof [10], Floor[10]
SECTLOADN (Sect,S1)
Topo = Sect[1]
Roof = Sect[2]
Floor = Sect[3]
```

In the above example file S1.LAY must exist in the working directory. This file contains the layer information for the section. The number of columns in this file depends on the number of surfaces in this section.

A typical example of a layer file contents is shown below:

column 1	chainage		
column 2	elevation of *Topo* layer at that chainage		
column 3	elevation of *Roof* layer at that chainage		
column 4	elevation of *Floor* layer at that chainage		
0.0	214.94	207.18	190.42
5.02	214.00	206.73	190.16
10.04	213.03	206.30	189.90
15.06	212.70	205.76	189.63
20.08	211.90	205.25	189.37
25.10	210.93	204.81	189.14
30.12	209.94	204.39	188.93
35.15	209.05	203.93	188.70
40.17	208.14	203.47	188.45
45.19	207.21	203.06	188.18
50.21	206.16	202.67	187.88

A.1.1 Declaration of Variables

All the variables need to be declared at the start in DSL's routines. Declarations create space in memory and label that space with the variable name. Two type of declaration are supported, they are "global" and "local" declaration. Global variables and local variables are quite different. A global variable can be called by any sub-routine as its is common and available to all sub-routines in the DSLX. Local variables are unique to their own sub-routines and they are not available for other sub-routines. For example, if two draglines are used in tandem and these two draglines have different dumping heights then two dumping sub-routines could be defined with each dumping sub-routine having a declared variable list. Local variables are extremely useful in writing DSLX sub-routines as they avoid the necessity for stringent naming conventions.

The declaration of a scalar is of the form:

Example A.6: *GLOBAL VARNAME,..*
LOCAL VARNAME,..

Allocations of memory is given to an array variable by its name and size of the array.

Example A.7: *GLOBAL_ARRAY STRIP[100],..*
LOCAL_ARRAY COL[10],..

Point variables are allocated to memory as having two parts. These are the X and the Z components. On declaration point variables are automatically allocated the values NULL,0. Thus, the X dimension of the point is given a null value, the Z dimension is given a 0. This NULL value indicates the point is undeclared and prevents its use in subsequent calculations. This is much safer than declaring a point 0,0 or some other arbitrary initialisation value.

Points are declared by the statements shown on the following example.

Example A.8: *GLOBAL_POINT VARNAME,..*
LOCAL_POINT VARNAME,..

Strings are defined in the format global string or local string as shown below.

Example A.9: *GLOBAL_STRING VARNAME[SIZE],..*
LOCAL_STRING VARNAME[SIZE], ..

String variables are managed in memory differently to other variables. On declaration the string variable is allocated space in memory and this space is given a label. The memory is initialised with a NULL value in the first position of the string. This NULL acts as an end of string marker and when the string is subsequently plotted or accessed the software looks for this NULL as a terminator. If a string of 100 points is declared only the initial point or the first point is filled with a NULL. If subsequently the first 10 points of the string are allocated values, then these points are given values and the 11th point is allocated a NULL point. On subsequent display or access of the string, the string is read until the NULL is found and the valid 10 points are displayed or accessed.

The declaration of the table variables is of the form:

Example A.10: *GLOBAL_TABLE VARNAME[SIZE1][SIZE2][2],..*
LOCAL_TABLE VARNAME[SIZE1][SIZE2][2],..

The arithmetic manipulation of variables, for example, projecting P1 at angle A1 to create P3, follows particular formats and structures which depend on the type of variable in use and the particular arithmetic operation being conducted.

A.2 DSLX'S LANGUAGE FUNCTIONS

The special language of DSLX software, uses a number of subroutines and library functions to perform the simulation process. Therefore, it is simpler and shorter to write a program compared with a general purpose language like FORTRAN or C. With the aid of graphical interface, debugging is faster and more efficient. Complex calculations such as the calculation of volume between two strings is handled in DSLX via the functions. Functions require arguments such as the top string and the base string. Functions avoid the user having to build these complex calculations. Tables A.1 and A.2 list the arithmetic and trigonometric functions used by DSLX. Other functions are described in more detail below.

A.2.1 Read and Write Functions

A number of functions is available to read data from an external file and write outputs to the report files. These functions are frequently used to import design parameters and to write various types of reports. Some of the common functions are:

CLOSE ( )
Closes the file opened for writing by the OPEN command. Only one file can be opened for writing at any time. No argument is required.

GETR
GETR (VAR,istart,ilength) reads a real value number from the text buffer. The text buffer is a memory element containing one line of text from an input file. The text is read into the buffer using READR. Three required arguments are:

VAR = Name of variable to hold value
istart = The column number where value exists in buffer.
ilength = The number of digits in the location.

Table A.1- List of the Arithmetic functions used by DSLX.

ARITHMETIC FUNCTIONS		
Statement	**Description**	**Example**
ABS (X)	Calculates the absolute value of variables in the argument list. At least one argument is required.	ABS (X) = 5.4 where X= -5.4
CHS (X)	Changes the sign of variables in the argument list. At least one argument is required.	CHS (A,B) is equivalent to: A = 0-1*A and B= -1 *B
COPY (X)	Copies the result of an expression into all variables in the argument list. At least two arguments are required.	COPY (X/2+3,Y,Z) is equivalent to: Y = X/2+3 and Z = X/2+3
DECR (X)	Decreases the value of variables in the argument list by 1. At least one argument is required.	DECR (X,Y) is equivalent to: X = X-1 and Y = Y-1
FRAC (X)	Returns the fractional component of values in the argument list. At least one argument is required.	FRAC (X,Y), If X = 3.4 and Y = 2.0 the resultant values of X and Y would be 0.4 and 0 respectively
INCR (X)	Increments variables in the argument list by 1. At least one argument is required.	INCR (X,Y) is equivalent to: X = X+1 and Y = Y+1
INT (X)	Returns the integer portion of a real number. At least one argument is required.	INT (X), If the original value of X was 3.54 then X will move to 3.0.
INV (X)	Calculates the inverse of values in the argument list. At least one argument is required.	INV (X,Y) is equivalent to: X = 1/X and Y = 1/Y
LN (X)	Returns the natural logarithm of variables in the argument list. At least one argument is required.	LN (X), If X was equal to 100 the resultant value of X would be 4.6.
LOG (X)	Returns the base 10 logarithm of variables in the argument list. At least one argument is required.	LOG (X), If X was equal to 1000 the resultant value of X would be 3.
MOD (A,B,C)	Calculates the integer remainder of (A) divided by (B). Three arguments are required.	MOD (A,B,C): (C) is the value returned. If A=10 and B=2, then C=0 ; If A=10 and B=3, then C=1
POWER (X)	Calculates the natural exponent of variables in the argument list. At least one argument is required.	POWER (X), If X was equal to 2.3 the resultant value of X would be 10.
POWER10 (X)	Calculates 10 raised to the power of each variable in the argument list. At least one argument is required.	POWER10 (X), If X was equal to 2 the resultant value of X would be 100.
SQRT (X)	Calculates the second root of values in the argument list. At least one argument is required.	SQRT(X,Y) is equivalent to: X = X^1/2 and Y = Y^1/2
SQUARE (X)	Calculates the square of values in the argument list. At least one argument is required.	SQUARE(X,Y) is equivalent to: X = X^2 and Y = Y^2

Table A.2 - List of the Trigonometric functions used by DSLX.

TRIGONOMETRIC FUNCTIONS		
Statement	**Description**	**Example**
ACOS (X)	Calculates the arc cosine of the variables in the argument list. At least one argument is required.	ACOS(X) If X is equal to 0.5 the resultant value of X would be 60.
ASIN (X)	Calculates the arc sine of the variables in the argument list. At least one argument is required.	ASIN(X) If X is equal to 0.5 the resultant value of X would be 30.
ATAN (X)	Calculates the arc tan of the variables in the argument list. At least one argument is required.	ATAN(X) If X is equal to 1 the resultant value of X would be 45.
COS (X)	Calculates the cosine of variables in the argument list. At least one argument is required.	COS (X) sets X to 0.5 if X is equal to 60.
SIN (X)	Calculates the sine of variables in the argument list.	SIN (X) sets X to 0.5 if X is equal to 30.
TAN (X)	Calculates the tangent of variables in the argument list. At least one argument is required.	TAN (X) sets X to 1 if X is equal to 45.

INQUIRE

Prompts for a character input. This is used for interactive data input by the user. This input is then assigned to the variable.

Format: INQUIRE "Please enter value for highwall angle" HWANG
This would assign the typed value to HWANG.

OPEN (["]FILENAME["],ISIZE)

Opens a file ready for ASCII data to be written to, such as a report file. Files are closed using the CLOSE command. If an OPEN command is issued, any previously opened file will be automatically closed.

PUTC

PUTC (String[I],POS) Places character variables into location specified by POS.

PUTR (rvalue, istart, ilength, ndecp)

Places a real value number into location specified by *istart,* into the text buffer. The text buffer is a memory element used to store a line of text for writing to an external file, using a series of PUTR and PUTS commands. Complex formatting at output text can be achieved.

rvalue = variable to be placed into location specified
istart = the column number for the start location
ilength = the number of digits in the location

ndecp = number of decimal points

PUTS ("text ",istart)

Places text into location specified by *istart*, into the text buffer.

READR (IOSTAT)

Reads a record from an external file (one line) into the text buffer. Returns a number >0 if an error occurs, e.g., EOF. This is used in conjunction with GETR to read data from input files. Input files are opened with OPENR. If IOSTAT is greater than 0, it means all records are read to the buffer.

WRITE ( )

Writes the most recent text buffer contents for the presently opened file and clears the buffer. The text buffer is filled using PUTS and PUTR commands.

WRITES ( )

Writes the most recent text buffer contents to the screen.

A.2.2 Points Operational Functions

PNTATTR

PNTATTR (P1,ATTR,VAR) returns either the X or Z attribute of a point depending on the value of ATTR. The three arguments required are:

P1 = Point

ATTR = Control switch that defines which attribute value is required.

1 = X value and 2 = Z value.

VAR = Scalar variable into which the attribute value is placed

Example A.11: *P1 = 3,2*
PNTATTR (P1,1,X) SETS X TO 3
PNTATTR (P1,2,X) SETS X TO 2

PNTDIST

PNTDIST (P1,P2,DIST) returns the distance between two points P1 and P2.

Three arguments are required. They are:

P1 = First point

P2 = Second point

DIST = Scalar variable into which the result of function is placed.

PNTINTS

PNTINTS (STR1,P1,ANG,PN) calculates a new point PN which is the intersection of a ray projected from point P1 at angle ANG with string STR1. Four arguments are required.

Examplea.12:
```
LOCAL ANG
LOCAL_POINT P1,P2,P3,PN
LOCAL_STRING STR1[5],STR2[5]
P1 = 250,180
P2 = 100,250
P3 = 500,300
STR1 = p2//p3
ANG = 75
PNTINTS (STR1,P1,ANG,PN)
PRINT "INTERSECTION POINT IS " PN
STR2 = P1//PN
```

Figure A.1 illustrates concepts used in the above example.

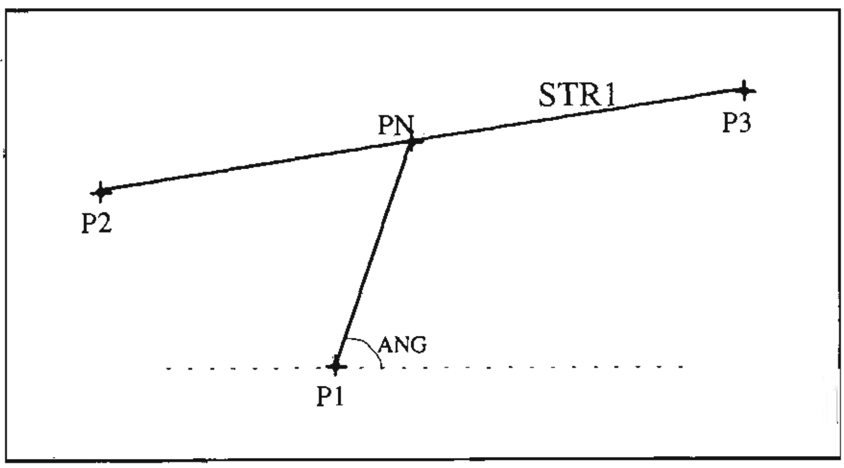


Figure A.1- Concepts of the "PNTINTS" function.

PNTINTSB

PNTINTSB (PT1,DIR,ANG,STR1,INTS,PT2,NUMINT) returns a point, P2, and the number of intersections (NUMINT) of a ray and a string. This function requires seven arguments. They are:

PT1 = Known point

DIR = A switch to determine if projection of the ray from PT1 is to be in one direction or two. DIR =1 - project one way and DIR = 2 - project both ways

ANG = Projection angle

STR1 = *String with which ray is to be intersected*

INTS = *An integer specifying which intersection is to be calculated i.e. if ints = 3 the third intersection will be calculated.*

PT2 = *Intersection point*

Example A.13:

```
LOCAL ANG,INTS,NUMINT
LOCAL_POINT PT2,PT1
LOCAL_STRING STR1[10],STR2[5]
INQUIRE "Angle for projected line is required, " ANG
INQUIRE "Which intersection, eg 1st, 3rd, " INTS
PNTINTSB (PT1,1,ANG,STR1,INTS,PT2,NUMINT)
PRINT "INTERSECTION POINT IS " PT2
PRINT "NUMBER OF INTERSECTION POINTS ARE " NUMINT
STR2 = PT1//PT2
DRAWSTR(STR2,2,1)
```

Figure A.2 illustrates concepts used in the above example.

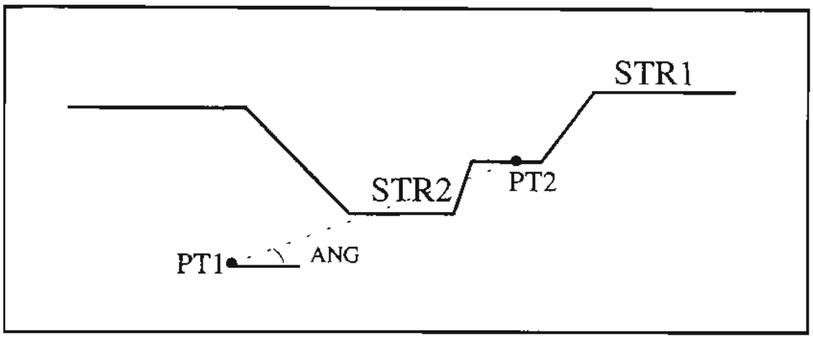


Figure A.2- Concepts of the "PNTINTSB" function.

A.2.3 String Functions

Most of dragline operation simulation in DSLX is performed through the use of strings. Strings are frequently used in design of cut and fill profiles and volumetric calculations. A great number of DSLX's functions is allocated to string operations. Some of the more important sting functions are described below.

CENTROID

CENTROID (STR,POINT) finds the centroid of a closed string. The major application of this function is in calculation of swing angles and hoist distances.

STR = *Closed string*

POINT = *Location of centroid*

REVERSE

REVERSE (STRING) reverses the numbering order of a string.

STRATTR

STRATTR (STR,DIST,IATTR,VALUE) calculates the X or ZY value of a string given a distance (XDIST) from the origin. Four arguments are required. They are:

STR = The string from which the value is to be obtained.
DIST = Distance from 0 along the X axis at which the attribute is to be calculated.
IATTR = A switch to determine which attribute is to be calculated. IATTR may only have one of two values, 1 or 2. IAATR = 1, Returns X value and IATTR = 2, Returns Z value
VALUE = Variable into which the calculated value is placed.

STREXTR

STREXTR (STR1,P1,P2,STR2) extracts a sub-string from an existing string. Four arguments are required. They are:

STR1 = Original string from which the sub-string is to be extracted.
P1 = Left hand limiting point.
P2 = Right hand limiting point.
STR2 = Extracted string.

Figure A.3 illustrates concepts used in STREXTR function.

STRFILTER

STRFILTER (STRING) deletes duplicate points on the string.

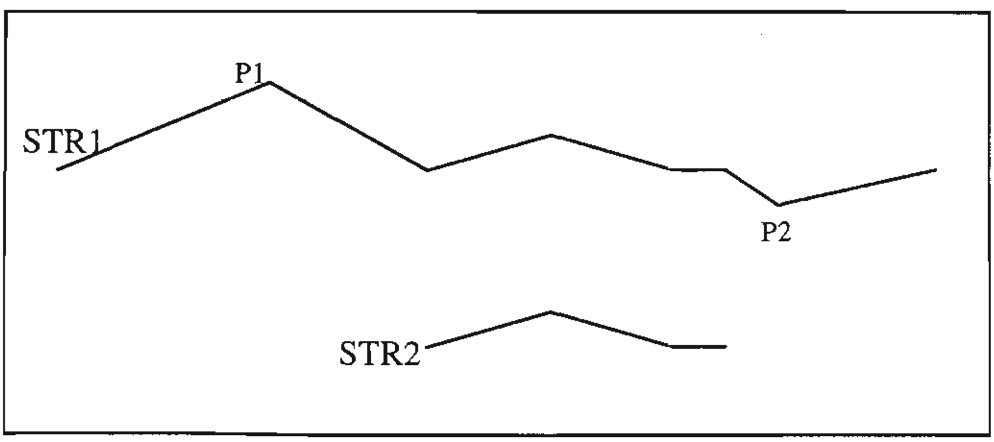


Figure A.3 - Concepts of the "STREXTR" function.

STRGETF

STRGETF (STR,["]FILENAME["]) reads data from an existing data file into a string. Each line consists of two value as X and Z values of a point. The format of an input file is shown below. The resultant string would have six points.

SAMPLE.STR

0	*90*
30	*100*
50	*97*
100	*90*
140	*89*
210	*93*

STRINGOPER

STRINGOPER (STR1,STR2,OPER,STR3) finds the maximum or minimum of two strings and generates a new string.

STR1 = First string.
STR2 = Second string.
OPER = Control Switch, 1 = Min of the two strings and 2 = Max of the two strings.
STR3 = Output string

Figure A.4 illustrates concepts used in STROPER function.

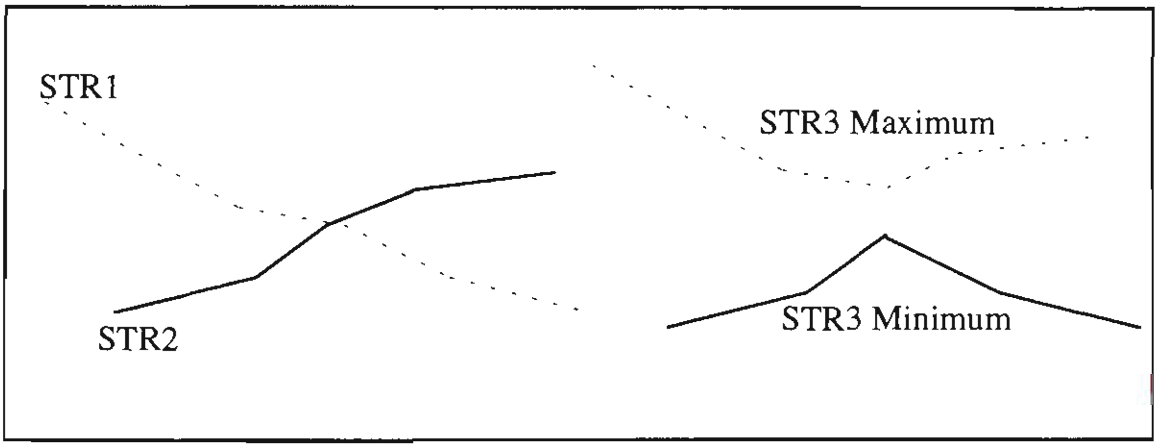


Figure A.4 - Concepts of the "STROPER" function.

STRINSCS

STRINSCS (STR1,STR2,TOL,STR3) inserts string 2 into string 1 to give a closed string 3. Four arguments are required. They are:

STR1 = Original string.

STR2 = Sub-string to be inserted into STR1.

TOL = Tolerance in metres to control search for insertion.

STR3 = Resultant string.

Figure A.5 illustrates concepts used in STRINSCS function.

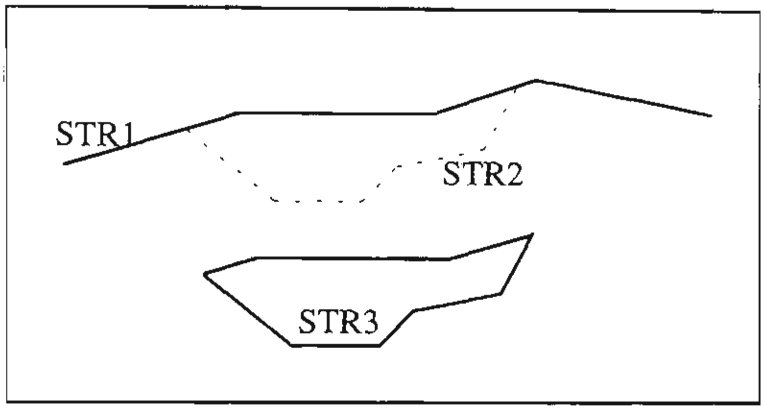


Figure A.5 - Concepts of the "STRINSCS" function.

STRINSOS

STRINSOS (STR1,STR2,TOL,STR3) inserts string 2 into string 1 to give an open string 3. Four arguments are required. They are:

STR1 = Original string.

STR2 = Sub-string to be inserted into STR1.

TOL = Tolerance in metres to control search for insertion.

STR3 = Resultant string.

Figure A.6 illustrates concepts used in STRINSOS function.

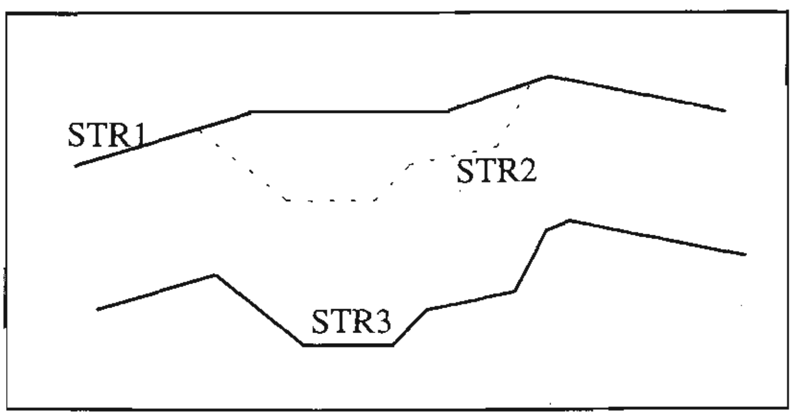


Figure A.6 - Concepts of the "STRINSOS" function.

STRINSP

STRINSP (P1,TOL,STR) inserts a point in a string if it falls within a given tolerance. Three arguments are required. They are:

P1 = Point to be inserted.

TOL = Tolerance for insertion in metres.

STR = String into which point is to be inserted.

STRINTS

STRINTS (STR1,STR2,P1) finds the intersection of two strings and returns it as a point. The first intersection is the one returned. Three arguments are required. They are:

STR1 = String 1.

STR2 = String 2.

P1 = Intersection point of STR1 and STR2.

STRLEN

STRLEN (STR,VAR) returns the number of data points currently stored in a string. Two arguments are required. They are:

STR = String.

VAR = Variable into which the number of points is placed.

STRVOL

STRVOL (STR,VOL) calculates the volume inside a closed string. Two arguments are required. They are:

STR = Closed string.

VOL = Calculated volume.

STRWRITE

STRWRITE (STR,["]FILENAME["]) writes a string to an external file. The default file extension is .STR. The format output is the same as that shown in the STRGETF function.

STRZRANGE

STRZRANGE (STR,D1,D2,ZMIN,ZMAX) calculates the maximum and minimum Z values for a string between two X limits. Five arguments are required. They are:

STR = String for which limits are to be calculated.

D1 = Left X limit.

D2 = Right X limit.
ZMN = Minimum Z value of string.
ZMX = Maximum Z value of string.

A.2.4 Drawing Functions

DRAWDR

DRAWDR (EQN,P1,SCALE,ROT) draws the currently defined dragline on the graphics screen. A set of dragline parameters must be defined prior to use of this function. The dragline parameters are defined through SETDRAGP function. Five arguments are required. They are:

EQN = Equipment number.
P1 = DL base position.
SCALE = Proportion of full scale as defined in SETDRAGP.
ROT = Rotation of dragline in degrees: (eg. 180 = facing to left).

SETDRAGP

SETDRAGP (H,RAD,ECLR,FCLR,CCLR,MDD,BW) sets the current dragline parameters for plotting. Seven arguments are required. They are:

H = Dump Height
RAD = Dump radius
ECLR = Rear end clearance
FCLR = Front clearance
CCLR = Crest clearance
MDD = Maximum dig depth
BW = Bucket width

DRAWGRID

DRAWGRID (P,ZT,ZA,XT,XA) draws coordinate grid only on the X & Z axis. Five arguments are required. They are:

P = Grid pen colour
ZT = Tick spacing on the Z-axis
ZA = Annotation spacing on the Z-axis
XT = Tick spacing on the X-axis
XA = Annotation spacing on the X-axis.

DRAWPT

DRAWPT (P1,SYM,COL,SZE) draws a point as a symbol or as text along side the point. The text drawn is taken from the most recent text buffer opened by one of the PUT commands.

P1 = Point to be drawn
SYM = Symbol of Point or text, for symbol use symbol number
COL = Colour of Point
SZE = Size of Point

DRAWSTR

DRAWSTR (STR,LCOL,LTYPE) draws a string on the screen. Three arguments are required. They are;

STR = The string which is to be drawn
LCOL = Line pen number
LTYPE = Line type code.

FILLSTR

FILLSTR (S1,S2,LP1,LP2,FP,FT) fills between two strings. This function is only valid if a graphics device has been selected and a section mounted (SECTION). The two strings to be filled between must have the same lateral extents. That is, the same minimum and maximum X values. The six required arguments are:

S1 = First string
S2 = Second string
LP1 = Line pen for first string
LP2 = Line pen for second string
FP = Fill pen number
FT = Fill type (0 = solid fill, 1 = hatch)

APPENDIX B

LIST OF COMPUTER PROGRAMS

This appendix contains the listing of five main computer programs developed during this study. The computer program cods are provided on a floppy disk in a packet at the back of the thesis. These programs can be used for various dragline operating techniques and for different configurations from a simple single seam to complex multi seam operations. Each of these main programs represents a different dragline mining method. In these programs the main routine is used to read input data, retrieve sections and geology of the section, controls the sequence of operation and also to call other sub-programs.

The basic principals to develop these programs are the digging method specifications and sequence of a dragline operation. These specifications are gathered through the digging method survey described in Chapter 1 and combined into one main computer programs for each major method. A main program may also be able to simulate various versions of a digging method with slight modifications.

APPENDIX C

"EXAMPLES OF THE OUTPUT REPORTS FROM THE CADSIM MODEL"

One of the unique aspects of the CADSIM model is generation enormous amount of information regarding mine design details, volumetric calculations and dragline performance data. The CADSIM model is totally flexible in generation and formatting output reports which are to be read into other softwares such as spreadsheet, mining reserve database and scheduling software. The followings are some sample outputs from the model.

C.1 GENERAL REPORT FILE

A general report file contains all volumetric information on a sectional basis. It also includes definition and default value of critical parameters used for each simulation run.

```
***************Dragline parameters*************************************************
Dump height:   30.0;   Dump radius:   87.0;   Dig depth:              45.0
Hwall clear    25.0    Rear clear     25.0    Crest clear             6.0
Bucket width   6.0     Tub radius     9.0     Working gradient        5.0
***********************************************************************************
```

```
******************Material parameters*********************************************
Repose angle   35.0    Coal trench angle   45.0    Spoil cut angle  45.0
Swell factor   1.2     Prime cut angle     75.0    Coal rib angle   75.0
***********************************************************************************
```

```
****************** Strip parameters ***********************************************
Strip width                  80.0     High wall angle                      75.0
Walk road width              40.0     Spoil bench width                    5.0
Maximum spoil flat top       10.0     Vertical distance to trench base     5.0
Max. overhand depth          15.0
 2.0 % extra rehandle allowed for first pass cleanup
***********************************************************************************
```

NOTE :- All volumes are in BCM

Sect Name	Str No.	D/line U/H	D/line Chop	Truck Vol.	Spoil Room	Spoil Req.	Increm Spoil	Cumul. Spoil	Rehan. Vol	Rehand. (%)	Coal Loss
S1	6	77325	1480	124089	652	78804	-78152	-78152	92575	141.0	2923
S3	6	79267	1507	121894	50514	214520	-30260	-164006	38599	57.3	1607
S4	6	79200	1507	120207	72480	244713	-8228	-172234	35561	52.9	1607
S5	6	71874	808	101132	92402	244916	19719	-152514	32591	53.8	1502
S6	6	70427	1313	93819	113826	224254	42087	-110427	32951	55.1	1488
S7	6	83887	53043	57903	139473	247358	2543	-107884	43278	37.9	1725
S8	6	85740	53857	55597	141764	247481	2167	-105717	44184	38.0	1733
S9	6	79612	49997	48941	135033	235326	5424	-100293	35508	32.9	1575
S10	6	79663	49817	44830	135660	229773	6180	-94113	36048	33.4	1572
S11	6	79614	49385	42342	136600	223112	7602	-86511	36990	34.4	1573
S12	6	79553	49134	39784	137563	215199	8875	-77636	37585	35.0	1572
S13	6	79389	48897	37146	138623	205922	10337	-67299	38155	35.7	1571
S14	6	79268	48655	34330	139580	195222	11657	-55643	38945	36.5	1571
S15	6	79039	48420	30775	140353	183101	12895	-42747	39696	37.4	1570
S16	6	79000	48313	26858	140776	170060	13463	-29284	40158	37.9	1571
S17	6	79074	48102	22644	141138	156460	13961	-15323	40902	38.6	1574
S18	6	64718	38366	14864	122270	118406	19187	0	32819	38.2	1323
S19	6	41311	22694	7397	90015	64005	26010	0	21443	40.2	900
S20	6	30744	15979	4624	73025	46723	26302	0	16053	41.2	704

C.2 DIG LEVELS REPORT FILE

Sect	Strip		High Wall Crest					Chop Bench		
No	No	Easting	Northing	R.L.	Grade	Depth	Easting	Northing	R.L.	Grade
S1	1	245736	7320684	222.6	0.0	41.9	245776	7320686	222.6	0.0
S2	1	245737	7320660	222.3	-1.3	42.2	245777	7320662	222.3	-1.3
S3	1	245737	7320634	221.7	-2.6	42.5	245777	7320637	221.7	-2.7
S4	1	245737	7320610	221.1	-2.3	42.9	245777	7320611	221.1	-2.2
S5	1	245737	7320584	220.5	-2.3	43.3	245777	7320586	220.5	-2.4
S6	1	245738	7320559	219.9	-2.6	43.7	245778	7320561	219.9	-2.5
S7	1	245738	7320534	219.2	-2.8	44.1	245779	7320536	225.2	1.1
S8	1	245738	7320509	218.5	-2.8	44.7	245790	7320512	223.8	4.8
S9	1	245738	7320484	217.3	-4.4	44.9	245790	7320486	222.6	3.6
S10	1	245738	7320459	216.1	-5.1	45.0	245789	7320461	226.1	-4.2
S11	1	245739	7320434	214.8	-5.1	45.0	245789	7320436	227.8	-3.4
S12	1	245739	7320409	213.5	-5.1	45.0	245788	7320412	228.5	-2.1
S13	1	245739	7320384	212.2	-5.1	45.0	245788	7320386	225.2	-2.9
S14	1	245739	7320359	211.0	-4.9	45.0	245788	7320361	225.0	-0.9
S15	1	245739	7320334	209.8	-4.7	45.0	245788	7320336	223.8	-4.7
S16	1	245739	7320309	208.6	-4.8	45.0	245788	7320311	222.6	-4.8
S17	1	245739	7320284	207.5	-4.6	45.0	245788	7320286	222.5	-0.6
S18	1	245739	7320259	206.4	-4.4	45.0	245788	7320261	220.4	-4.4
S19	1	245739	7320234	205.4	-3.9	44.9	245788	7320236	220.4	0.0
S20	1	245739	7320208	204.6	-3.3	44.7	245788	7320211	228.6	-3.1

C.3 COAL AND PARTING VOLUMES REPORT

Strip No.	Sect No.	Seam No.	Coal Vol	Parting Vol
1	1	1	42785	133
2	1	1	47703	145
1	2	1	43158	52
2	2	1	53621	58
1	3	1	41583	0
2	3	1	50169	21
1	4	1	37401	10
2	4	1	48412	93
1	5	1	40452	37
2	5	1	48050	52
1	6	1	40610	384
2	6	1	46893	84
1	7	1	42050	37
2	7	1	48337	87
1	8	1	39852	56
2	8	1	45384	79
1	9	1	35984	36
2	9	1	38437	48
1	10	1	36873	116
2	10	1	38201	70
1	11	1	37534	48
2	11	1	38158	68
1	12	1	38772	75
2	12	1	38492	94
1	13	1	40414	43
2	13	1	39331	103
1	14	1	38742	54
2	14	1	40657	78
1	15	1	40358	67
2	15	1	42334	86
1	16	1	41813	49
2	16	1	44051	84
1	17	1	43024	63
2	17	1	45483	134
1	18	1	43195	45
2	18	1	44346	89
1	19	1	41036	67
2	19	1	39945	94
1	20	1	37244	68
2	20	1	35459	56

C.4 REHANDLE REPORT

Strip	Section	Easting	Northing	Rehandle %
6	1	245717	7320684	98.56
6	2	245718	7320659	76.01
6	3	245718	7320634	68.81
6	4	245718	7320609	53.21
6	5	245719	7320584	59.70
6	6	245718	7320558	59.51
6	7	245718	7320533	50.62
6	8	245719	7320508	37.22
6	9	245719	7320483	37.01
6	10	245718	7320458	34.72
6	11	245719	7320433	35.99
6	12	245718	7320408	35.44
6	13	245717	7320383	35.27
6	14	245717	7320358	34.90
6	15	245717	7320333	34.67
6	16	245718	7320308	34.21
6	17	245718	7320283	33.86
6	18	245719	7320258	33.71
6	19	245719	7320233	35.09
6	20	245719	7320208	35.90
7	1	245797	7320688	65.12
7	2	245797	7320662	57.45
7	3	245797	7320638	52.32
7	4	245797	7320613	45.55
7	5	245798	7320588	42.68
7	6	245798	7320562	44.53
7	7	245778	7320536	44.45
7	8	245778	7320511	42.55
7	9	245778	7320486	34.74
7	10	245778	7320461	34.70
7	11	245779	7320436	34.84
7	12	245779	7320411	34.97
7	13	245779	7320386	34.89
7	14	245779	7320361	34.67
7	15	245779	7320336	34.80
7	16	245799	7320312	34.56
7	17	245799	7320287	34.87
7	18	245799	7320262	34.80
7	19	245799	7320237	34.98
7	20	245799	7320212	35.24

C.5 DETAILED REPORT

Strip No.	Sec No.	D/line Vol (U/H)	OVERHAND CHOP Vol(chop)	SWG1	HST1	UNDERHAND KEY CUT VOL	SWG1	HST1	SWG2	HST2	VOLUME	SWG1	UNDERHAND MAIN SEGMENT1 HST1	SWG2	HST2	SWG1	HST1	SW2	HST2
1	1	81352	0	0	0	25511	12	23	78	0	21024	16	13	103	0	27	13	90	0
1	2	80692	0	0	0	25200	12	23	78	0	21067	15	13	102	0	27	13	90	0
1	3	81647	0	0	0	25023	12	23	78	0	22470	15	13	102	0	26	13	90	0
1	4	81620	0	0	0	24699	12	23	78	0	23361	14	12	78	0	26	12	91	4
1	5	80381	0	0	0	24312	11	23	79	0	22855	14	12	44	0	25	12	85	23
1	6	80658	0	0	0	23919	11	22	79	0	23980	13	11	45	0	24	11	50	29
1	7	82059	7255	116	0	24402	11	23	79	0	24767	13	11	45	0	25	11	49	30
1	8	80106	46247	90	29	23000	11	22	79	0	25766	17	10	40	0	26	10	47	30
1	9	79520	47788	90	23	22791	11	22	79	0	25638	17	10	39	0	26	10	46	30
1	10	79221	47972	90	21	22756	11	22	79	0	25428	16	10	39	0	25	10	46	31
2	1	77353	0	0	0	22670	11	22	79	0	24971	16	10	102	0	26	10	94	0
2	2	78924	0	0	0	22695	11	22	79	0	25841	17	10	101	0	26	10	92	0
2	3	80324	0	0	0	22632	11	22	79	0	26639	18	10	100	0	26	10	91	0
2	4	80429	0	0	0	22644	11	22	79	0	26701	18	10	100	0	26	10	91	3
2	5	80512	0	0	0	22713	11	22	79	0	26691	18	10	38	0	26	10	91	24
2	6	80422	0	0	0	22704	11	22	79	0	26633	17	10	38	0	26	10	45	31
2	7	80411	0	0	0	22687	11	22	79	0	26566	17	10	37	0	26	10	44	30
2	8	80434	39222	110	12	22684	11	22	79	0	26482	17	10	37	0	26	10	45	32
2	9	80474	49334	108	15	22696	11	22	79	0	26397	17	10	37	0	26	10	45	32
2	10	80587	47422	110	15	22732	11	22	79	0	26346	17	10	37	0	26	10	46	32

C.5 DETAILED REPORT (CONTINUED)

UNDERHAND MAIN SEGMENT3								UNDERHAND MAIN SEGMENT4								PARTLY DUMPED	
VOLUME	SWG1	HST1	SWG2	HST2	VOLUME	SWG1	HST1	SWG2	HST2	VOLUME	SWG1	HST	VOLUME	SWG	HST	VOLUME	VOLUME
25606	12	10	113	0	84161	37	47	88	0	1883	90	47	97626	125	47	0	106064
24703	12	10	113	0	55579	30	40	95	0	20448	90	40	169939	125	40	0	78330
23053	11	10	114	0	25531	22	32	103	0	52666	88	32	204656	125	32	0	46787
21720	10	10	77	0	14639	23	33	64	0	0	0	0	200698	125	16	0	35104
21379	11	10	78	2	16200	24	34	63	2	0	0	0	150229	125	36	0	34023
19961	10	10	77	20	14490	22	32	64	21	0	0	0	71781	125	53	0	33427
19731	10	10	62	23	15130	23	33	63	30	0	0	0	0	0	0	380	32126
17163	9	11	57	24	12989	22	31	65	30	0	0	0	0	0	0	9045	29593
16710	9	10	55	24	11772	22	30	66	30	0	0	0	0	0	0	9887	26928
16606	9	10	54	24	10699	21	29	66	30	0	0	0	0	0	0	7096	27162
19194	9	11	116	0	79110	35	45	90	0	2959	86	45	89272	125	45	0	94132
18244	9	11	117	0	53302	29	38	96	0	17762	85	38	162667	125	38	0	70522
17526	8	11	117	0	22244	20	29	105	0	46818	84	29	202941	125	29	0	39684
16903	8	11	76	0	9901	22	30	67	0	0	0	0	202768	125	13	0	26725
16083	9	10	77	1	10168	21	29	66	2	0	0	0	160517	125	31	0	26178
16020	9	10	74	20	10006	21	29	67	20	0	0	0	91904	125	49	0	25967
15940	9	10	50	23	9805	21	29	67	30	0	0	0	17860	125	59	0	25704
15852	9	10	53	25	9570	21	29	63	30	0	0	0	0	0	0	6550	25399
15869	9	10	53	25	9567	21	29	63	30	0	0	0	0	0	0	9995	25412
15948	9	10	56	26	9692	21	29	67	30	0	0	0	0	0	0	9463	25605

APPENDIX D

"FREQUENCY HISTOGRAMS OF THE PERFORMANCE PARAMETERS AND BEST FIT RESULTS"

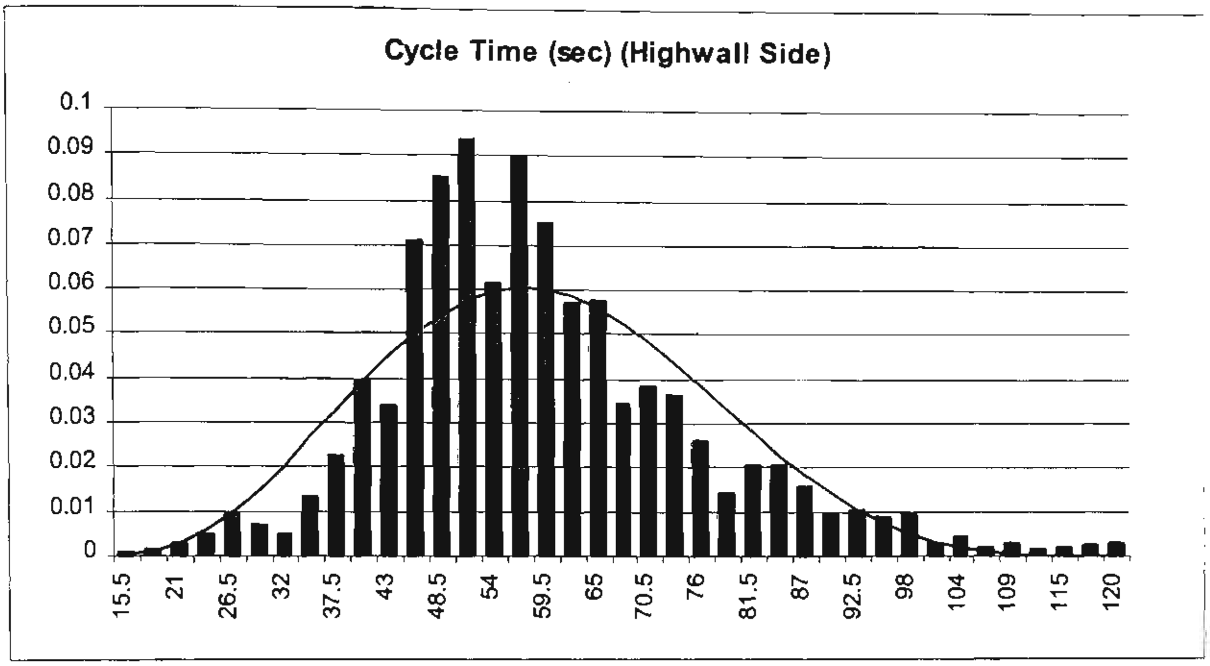
This appendix presents the detailed results of the frequency histograms of the dragline performance data captured by the dragline monitoring system as described in Chapter 7. It also covers the results of the Best Fit analysis using Input Data Analysis option in ARENA software.

The distributions are listed from best to worst based upon the values of the respective squared errors. The quality of a curve fit is based primarily on the square error criterion, which is defined as the sum of $[f_i - f(x_i)]$, summed over all histogram intervals. In this expression f_i refers to the relative frequency of the data for the i^{th} interval, and $f(x_i)$ refers to the relative frequency for the fitted probability distribution function.

For most of the distributions supported by the software, the curve fitting is based on the use of maximum likelihood estimators. Exceptions to this rule are the Beta, triangular and Uniform distributions. The Beta distribution is fitted in two different ways, first using maximum likelihood estimators, and then the method of moments. The results corresponding to the best of these fits are then retained. The Triangular and Uniform distributions use empirical rules to fit the distribution to the data.

The results of chi-square and (for non-integer data) Kolmogrov-Smirnov goodness-of-fit tests are also shown. These results are presented in the form of p values. These are based upon the probability of committing a type I error (i.e., the probability that rejection of the distribution function will be an incorrect decision).

Cycle Time (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45823
Min Data Value	= 10
Max Data Value	= 120
Sample Mean	= 57.7
Sample Std Dev	= 16.7

Best Fit Results:

Function	*Sq Error*
Gamma	0.00365
Erlang	0.00414
Normal	0.00533
Beta	0.00562
Lognormal	0.00606
Triangular	0.0106
Weibull	0.0169
Uniform	0.0303
Exponential	0.039

Distribution Summary

Distribution:	Gamma
Expression:	10 + GAMM(6.38, 7.48)
Square Error:	0.003653

Chi Square Test

Number of intervals	= 34
Degrees of freedom	= 31
Test Statistic	= 605
Corresponding p-value	< 0.005

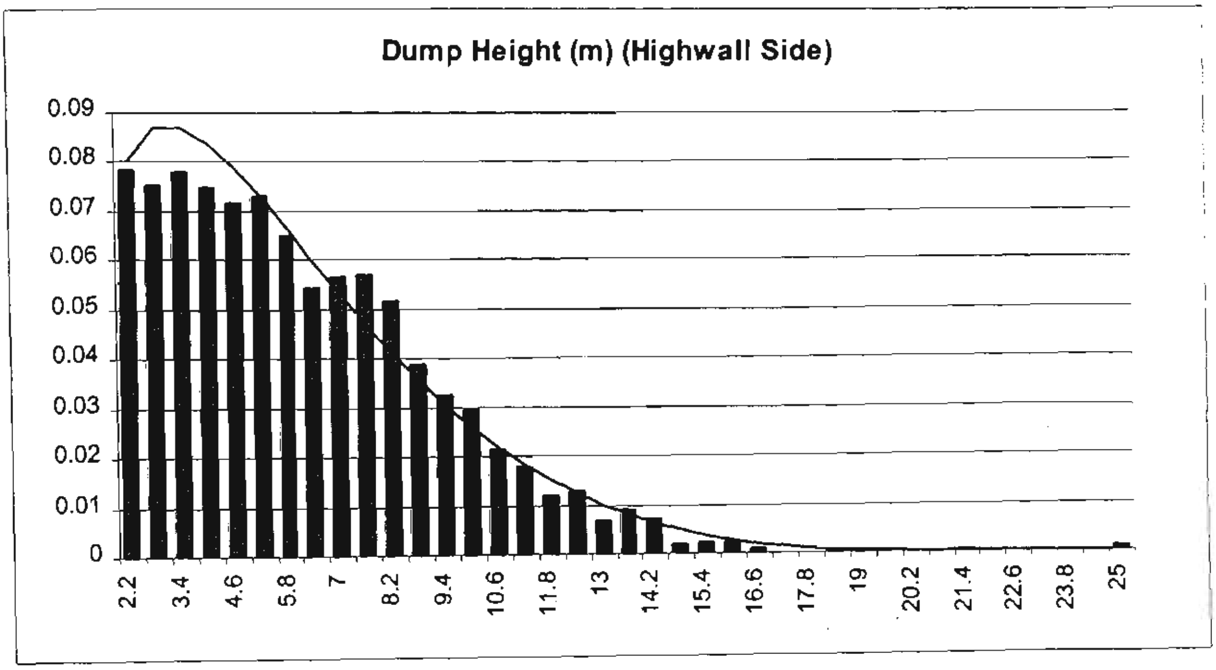
Kolmogorov-Smirnov Test

Test Statistic = 0.0525
Corresponding p-value < 0.01

Histogram Summary

Histogram Range	= 10 to 120
Number of Intervals	= 40

Dump Height (Highwall Side Stripping)



Data Summary

No. of Data Points	= 42560
Min Data Value	= 1.01
Max Data Value	= 24.9
Sample Mean	= 5.57
Sample Std Dev	= 3.19

Best Fit Results

Function	*Sq Error*
Beta	0.000782
Weibull	0.00168
Gamma	0.00187
Erlang	0.00334
Normal	0.00536
Exponential	0.00717
Lognormal	0.00763
Triangular	0.0139
Uniform	0.0345

Distribution Summary

Distribution: Beta
Expression: 1 + 24 * BETA(1.4, 5.97)
Square Error: 0.000782

Chi Square Test

Number of intervals = 25
Degrees of freedom = 22
Test Statistic = 54.8
Corresponding p-value < 0.005

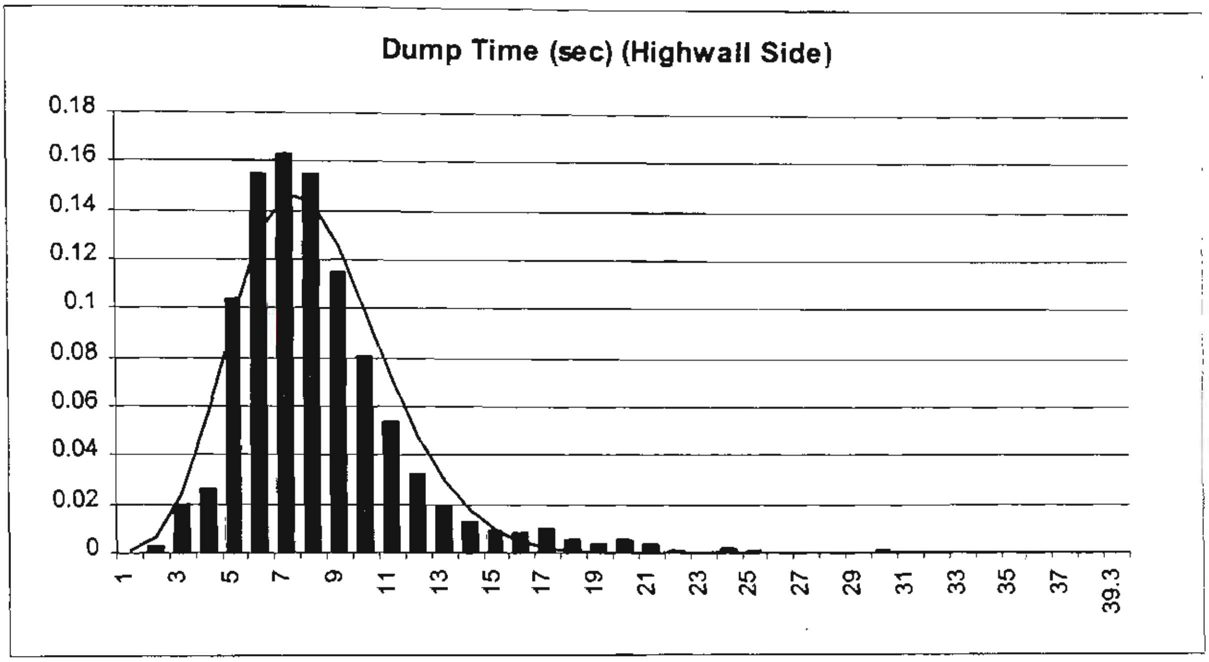
Kolmogorov-Smirnov Test

Test Statistic = 0.579
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 1 to 25
Number of Intervals = 40

Dump Time (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45886
Min Data Value	= 0
Max Data Value	= 35
Sample Mean	= 8.16
Sample Std Dev	= 3.43

Best Fit Results

Function	*Sq Error*
Beta	0.0035
Poisson	0.00553
Gamma	0.00883
Erlang	0.00946
Lognormal	0.0128
Normal	0.0136
Weibull	0.0442
Triangular	0.0589
Exponential	0.078
Uniform	0.0868

Distribution Summary

Distribution: Beta
Expression: -0.5 + 40.5 * BETA(7.39, 27.8)
Square Error: 0.003499

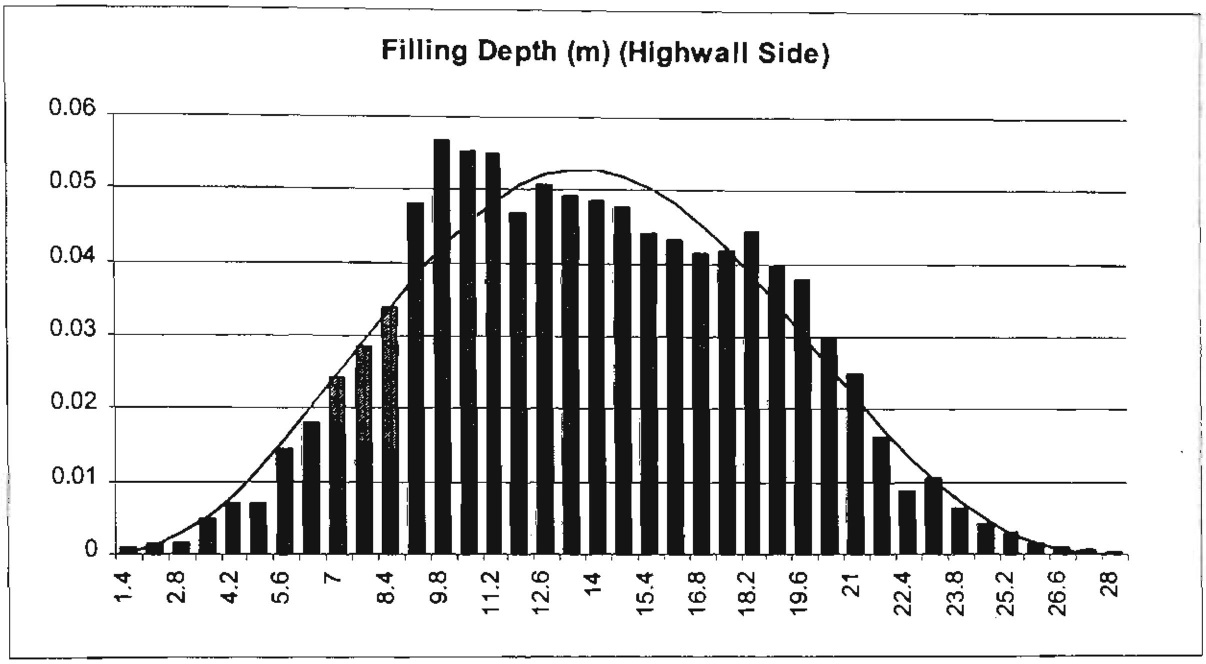
Chi Square Test
Number of intervals = 16
Degrees of freedom = 13
Test Statistic = 1.85e+003
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 0.579
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = -0.5 to 40
Number of Intervals = 40

Filling Depth (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45890
Min Data Value	= 0
Max Data Value	= 27.4
Sample Mean	= 13.4
Sample Std Dev	= 4.73

Best Fit Results

Function	*Sq Error*
Beta	0.000807
Gamma	0.00134
Erlang	0.00134
Normal	0.00148
Triangular	0.0022
Lognormal	0.00271
Weibull	0.00819
Uniform	0.0159
Exponential	0.0258

Distribution Summary

Distribution: Beta
Expression: -0.001 + 28 * BETA(3.72, 4.02)
Square Error: 0.000807

Chi Square Test

Number of intervals = 34
Degrees of freedom = 31
Test Statistic = 139
Corresponding p-value < 0.005

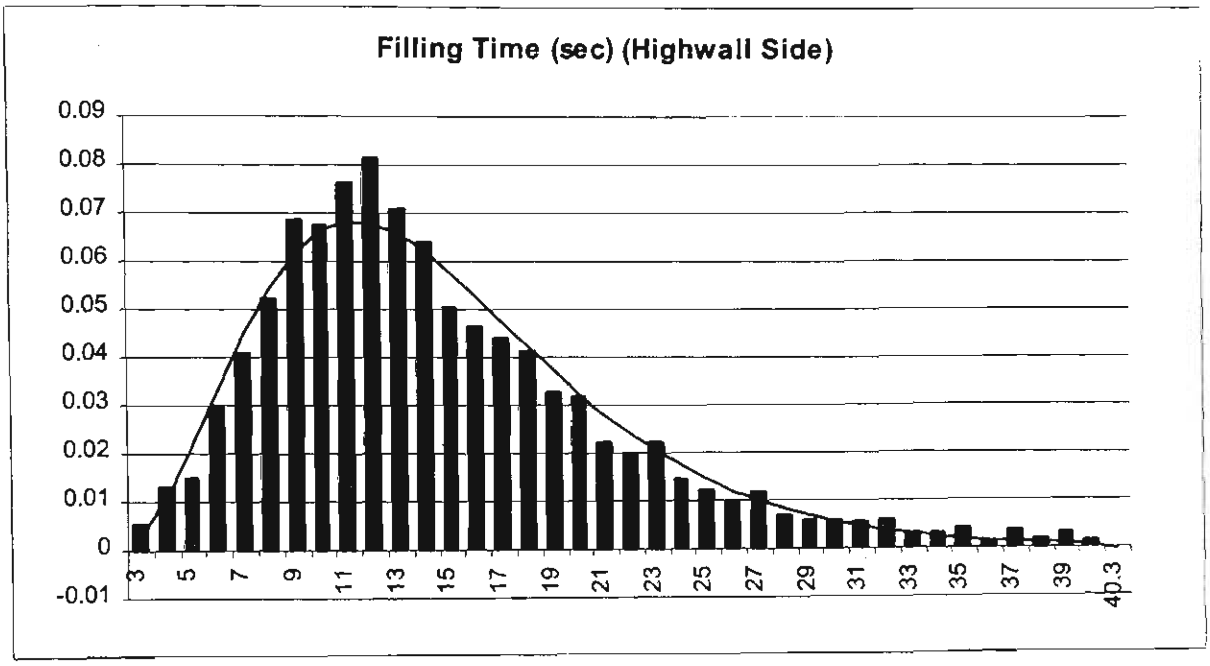
Kolmogorov-Smirnov Test

Test Statistic = 0.0314
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = -0.001 to 28
Number of Intervals = 40

Filling Time (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45732
Min Data Value	= 2
Max Data Value	= 40
Sample Mean	= 14.6
Sample Std Dev	= 6.72

Best Fit Results

Function	*Sq Error*
Erlang	0.000826
Lognormal	0.00088
Gamma	0.00101
Beta	0.00305
Normal	0.00598
Triangular	0.0123
Poisson	0.016
Exponential	0.0234
Uniform	0.0274
Weibull	0.0697

Distribution Summary

Distribution: Erlang
Expression: 1.5 + ERLA(3.27, 4)
Square Error: 0.000621

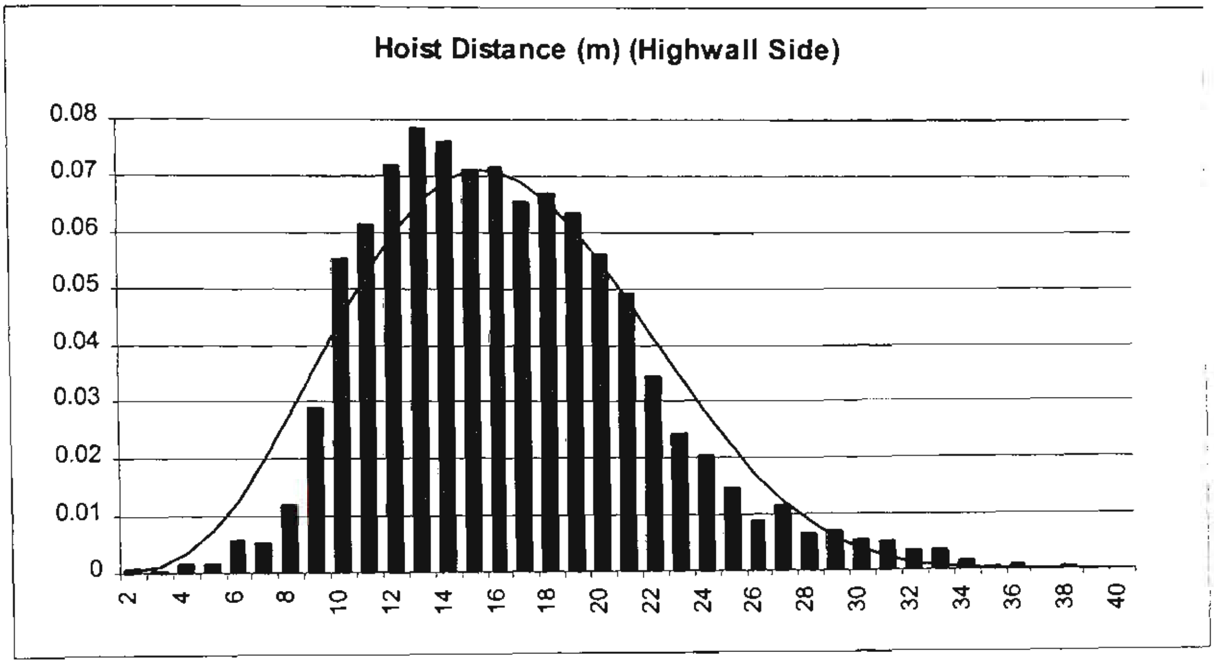
Chi Square Test
Number of intervals = 35
Degrees of freedom = 32
Test Statistic = 202
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 0.0314
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 1.5 to 40
Number of Intervals = 40

Hoist Distance (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45934
Min Data Value	= 0
Max Data Value	= 38.5
Sample Mean	= 15.8
Sample Std Dev	= 5.41

Best Fit Results

Function	*Sq Error*
Beta	0.00159
Normal	0.00204
Lognormal	0.00487
Erlang	0.00782
Gamma	0.00788
Triangular	0.0104
Uniform	0.0313
Exponential	0.0385
Weibull	0.103

Distribution Summary

Distribution: Beta
Expression: -0.001 + 40 * BETA(4.75, 7.31)
Square Error: 0.001588

Chi Square Test

Number of intervals = 29
Degrees of freedom = 26
Test Statistic = 608
Corresponding p-value < 0.005

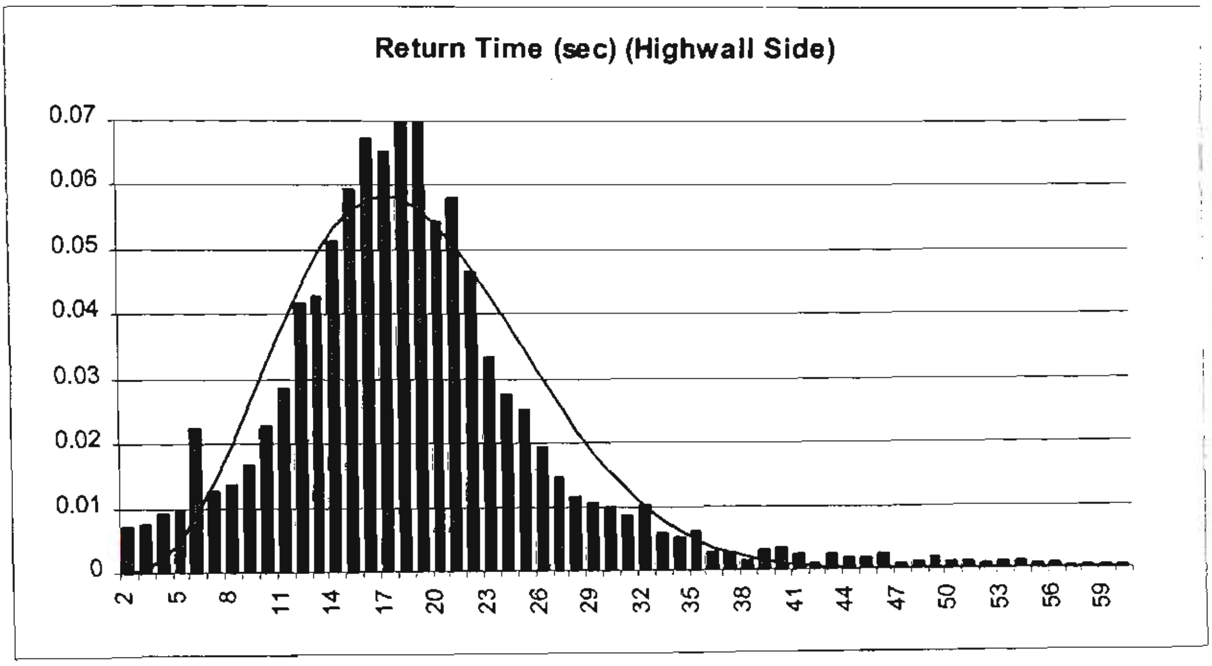
Kolmogorov-Smirnov Test

Test Statistic = 0.0436
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = -0.001 to 40
Number of Intervals = 40

Return Time (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45757
Min Data Value	= 1
Max Data Value	= 60
Sample Mean	= 18.5
Sample Std Dev	= 8.05

Best Fit Results

Function	*Sq Error*
Beta	0.00206
Normal	0.00339
Erlang	0.00349
Gamma	0.00421
Poisson	0.00554
Lognormal	0.00706
Triangular	0.0146
Exponential	0.0264
Uniform	0.0266
Weibull	0.0592

Distribution Summary

Distribution: Beta
Expression: 0.5 + 60 * BETA(4.72, 10.9)
Square Error: 0.002065

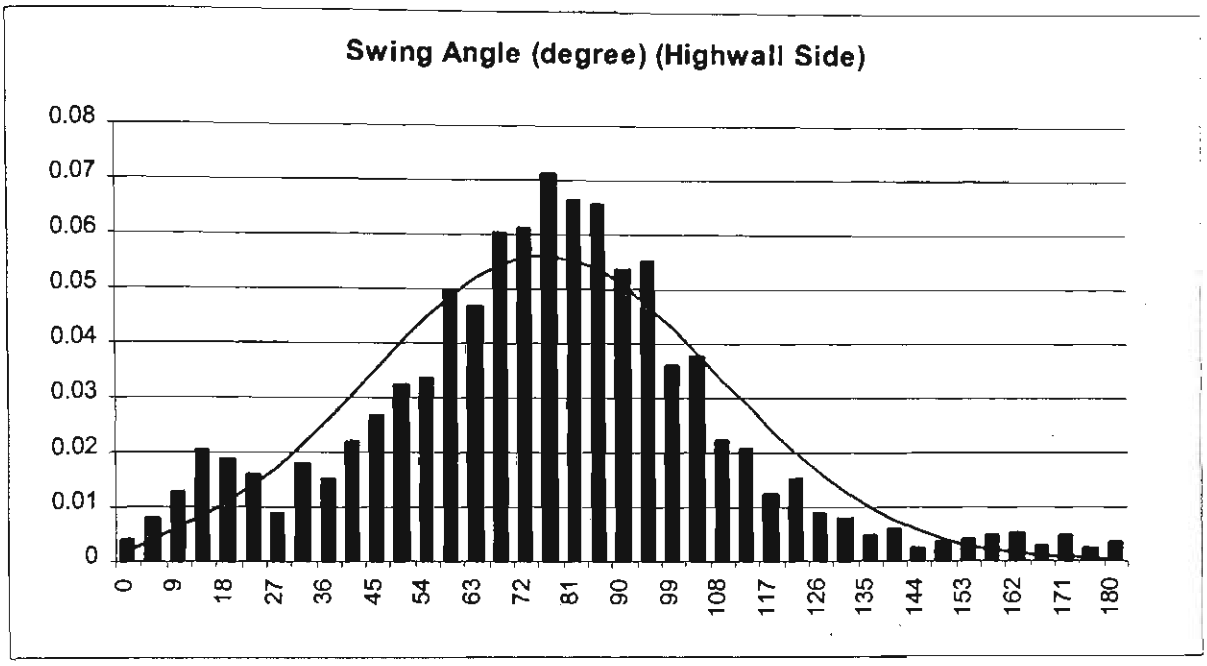
Chi Square Test

Number of intervals	= 36
Degrees of freedom	= 33
Test Statistic	= 2.14e+003
Corresponding p-value	< 0.005

Histogram Summary

Histogram Range	= 0.5 to 60.5
Number of Intervals	= 60

Swing Angle (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45812
Min Data Value	= 1
Max Data Value	= 180
Sample Mean	= 73.2
Sample Std Dev	= 31.9

Best Fit Results

Function	*Sq Error*
Normal	0.00325
Beta	0.00368
Triangular	0.00688
Erlang	0.00832
Gamma	0.00912
Lognormal	0.015
Uniform	0.021
Exponential	0.0271
Weibull	0.0833

Distribution Summary

Distribution: Normal
Expression: NORM(73.2, 31.9)
Square Error: 0.003248

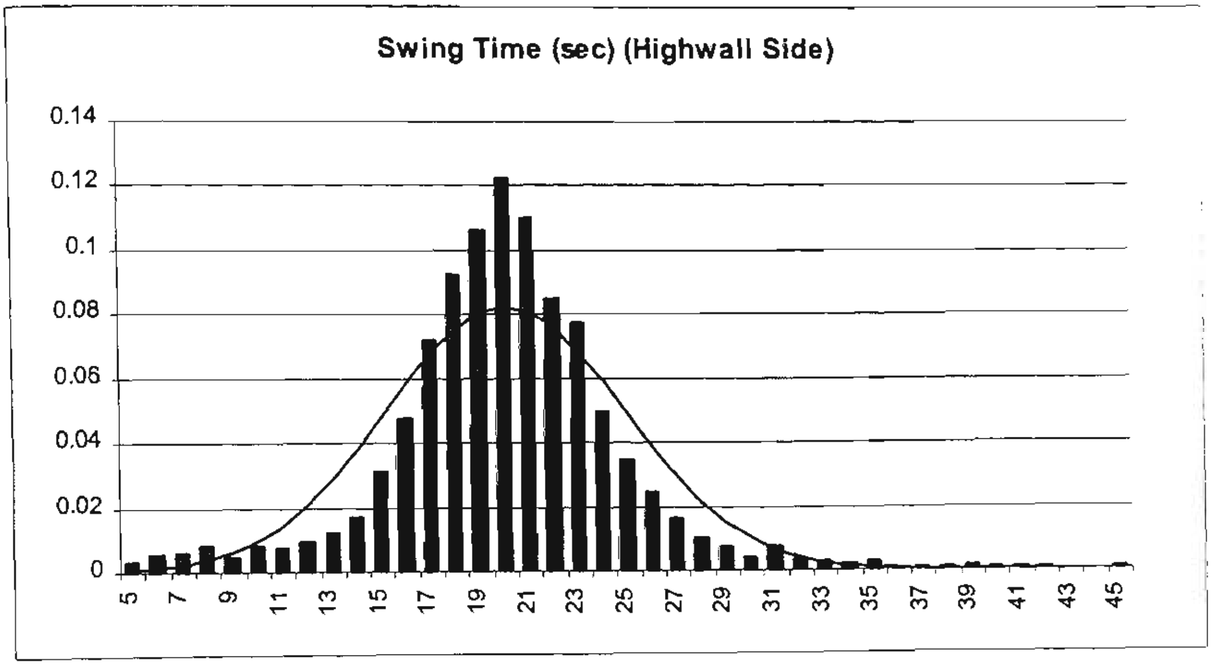
Chi Square Test

Number of intervals = 36
Degrees of freedom = 33
Test Statistic = 1.09e+003
Corresponding p-value < 0.005

Histogram Summary

Histogram Range = -0.001 to 180
Number of Intervals = 40

Swing Time (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45888
Min Data Value	= 3
Max Data Value	= 68
Sample Mean	= 20
Sample Std Dev	= 4.83

Best Fit Results

Function	*Sq Error*
Normal	0.00567
Poisson	0.00623
Beta	0.00683
Gamma	0.0108
Erlang	0.0111
Lognormal	0.016
Triangular	0.0414
Weibull	0.0444
Uniform	0.0587
Exponential	0.0588

Distribution Summary

Distribution:	Normal
Expression:	NORM(20, 4.83)
Square Error:	0.005671

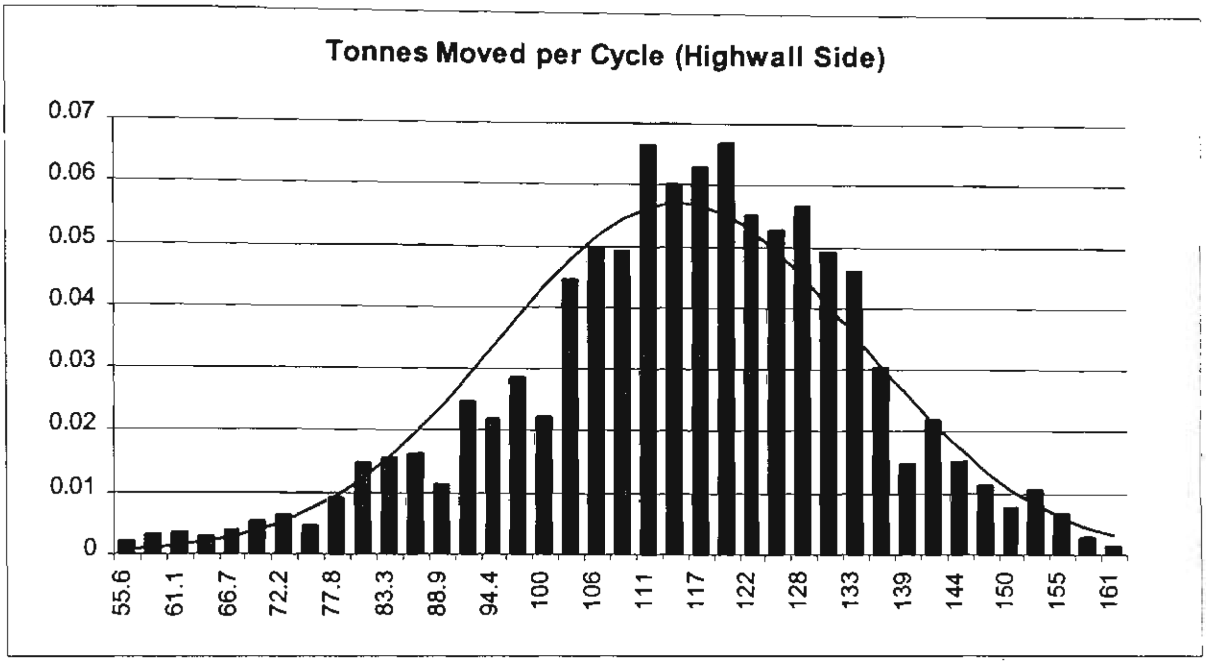
Chi Square Test

Number of intervals	= 29
Degrees of freedom	= 26
Test Statistic	= 1.08e+003
Corresponding p-value	< 0.005

Histogram Summary

Histogram Range	= 2.5 to 68.5
Number of Intervals	= 66

Tonnes Moved per Cycle (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45777
Min Data Value	= 50
Max Data Value	= 161
Sample Mean	= 113
Sample Std Dev	= 19.4

Best Fit Results

Function	*Sq Error*
Normal	0.00336
Beta	0.00365
Triangular	0.00479
Gamma	0.00739
Erlang	0.00758
Lognormal	0.0108
Uniform	0.0195
Exponential	0.0318
Weibull	0.117

Distribution Summary

Distribution:	Normal
Expression:	NORM(113, 19.4)
Square Error:	0.003361

Chi Square Test

Number of intervals = 37
Degrees of freedom = 34
Test Statistic = 752
Corresponding p-value < 0.005

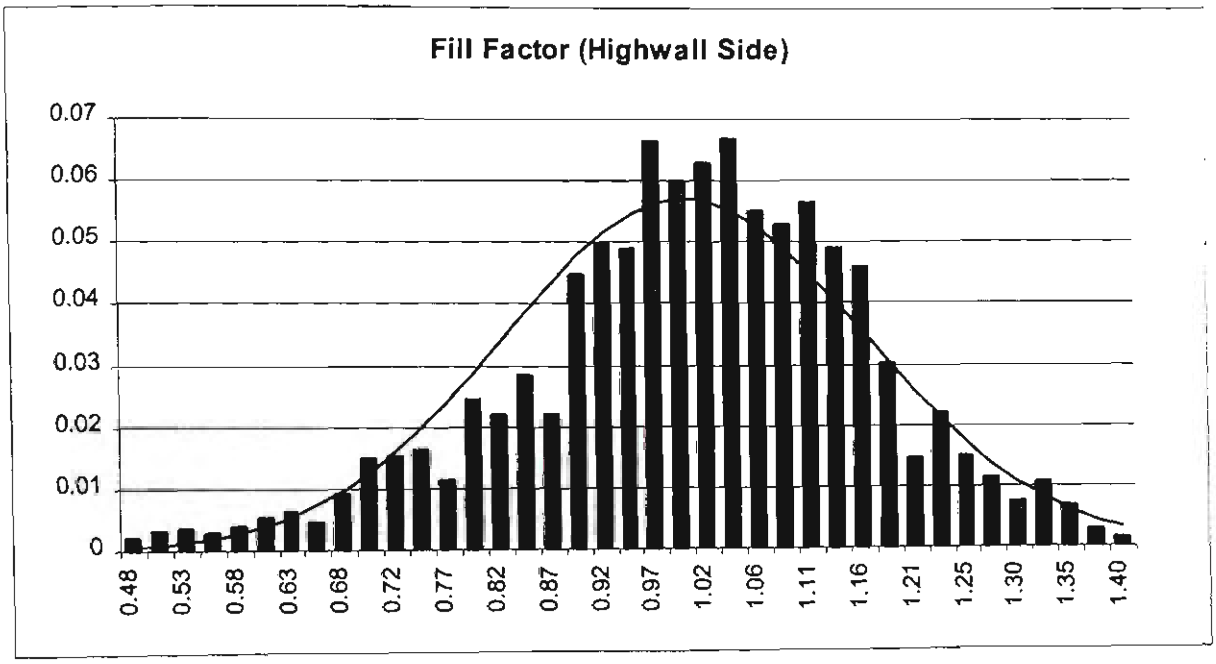
Kolmogorov-Smirnov Test

Test Statistic = 0.0482
Corresponding p-value < 0.01

Histogram Summary

Histogram Range	= 50 to 161
Number of Intervals	= 40

Filling Factor (Highwall Side Stripping)



Data Summary

No. of Data Points	= 45777
Min Data Value	= 0.139
Max Data Value	= 1.40
Sample Mean	= 0.968599
Sample Std Dev	= 0.19207

Best Fit Results

Function	*Sq Error*
Normal	0.00336
Beta	0.00365
Triangular	0.00479
Gamma	0.00739
Erlang	0.00758
Lognormal	0.0108
Uniform	0.0195
Exponential	0.0318
Weibull	0.117

Distribution Summary

Distribution: Normal
Expression: NORM(0.968, 0.192)
Square Error: 0.003361

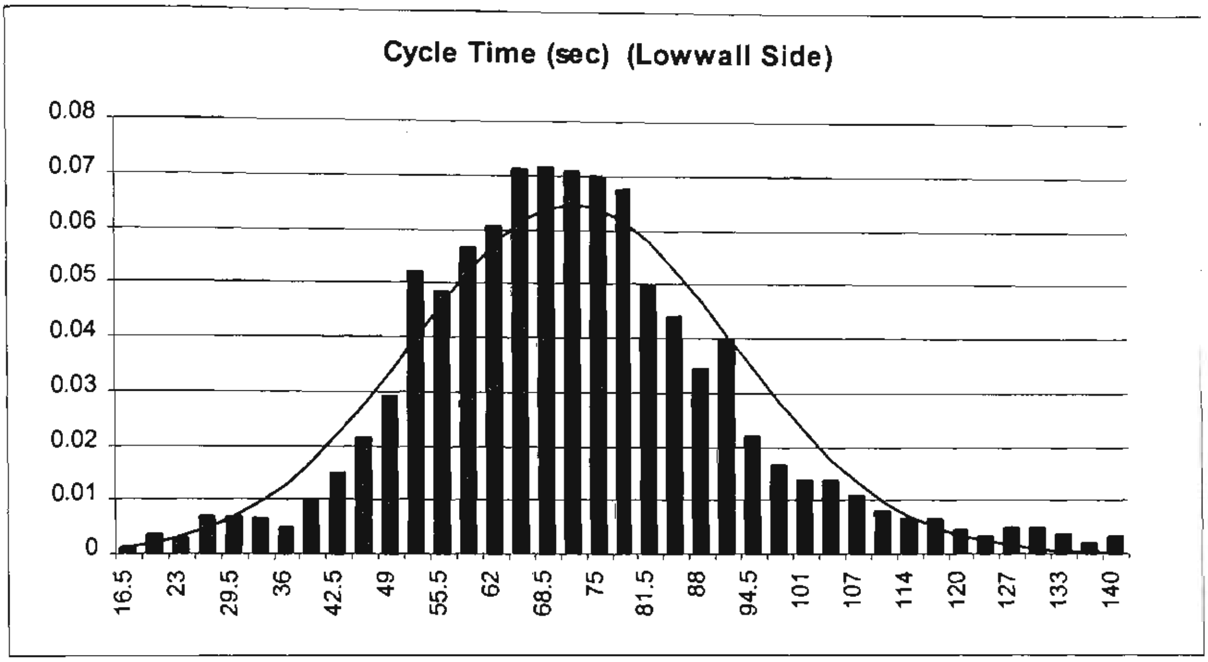
Chi Square Test
Number of intervals = 37
Degrees of freedom = 34
Test Statistic = 752
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 0.0482
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 0.48 to 1.4
Number of Intervals = 40

Cycle Time (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47738
Min Data Value	= 10
Max Data Value	= 140
Sample Mean	= 70.3
Sample Std Dev	= 20

Best Fit Results

Function	*Sq Error*
Normal	0.00241
Beta	0.00278
Erlang	0.0029
Gamma	0.00332
Lognormal	0.00683
Weibull	0.00776
Triangular	0.00823
Uniform	0.0267
Exponential	0.0366

Distribution Summary

Distribution:	Normal
Expression:	NORM(70.3, 20)
Square Error:	0.002414

Chi Square Test

Number of intervals = 36
Degrees of freedom = 33
Test Statistic = 1.3e+003
Corresponding p-value < 0.005

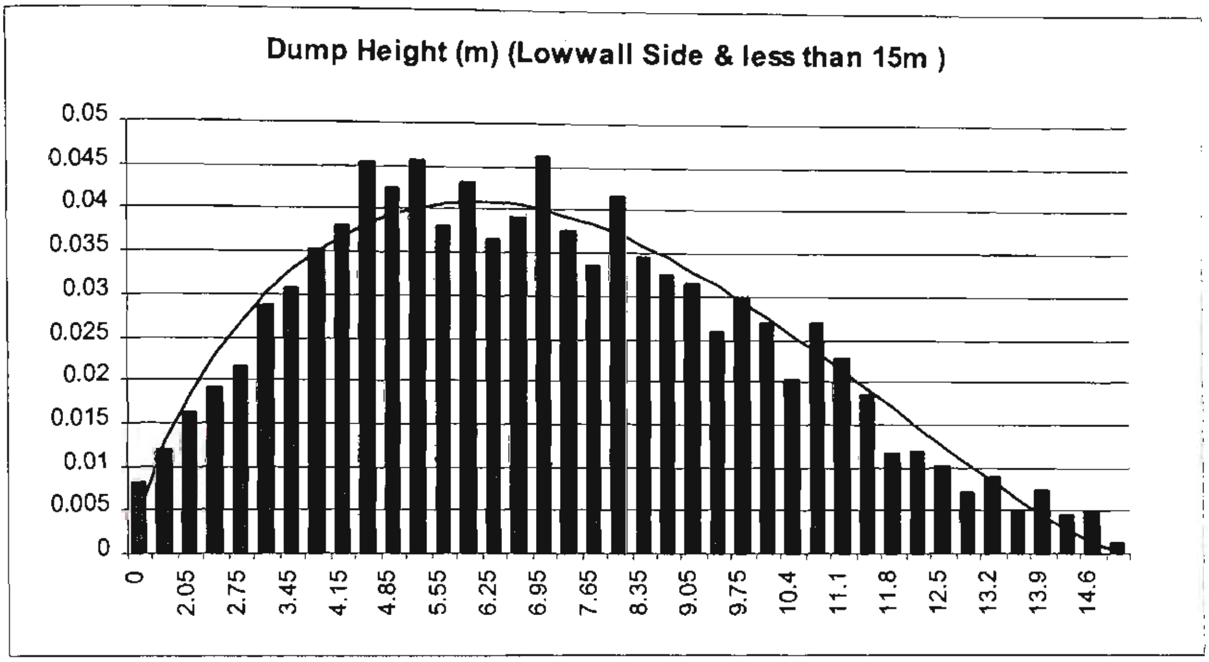
Kolmogorov-Smirnov Test

Test Statistic = 5.47e+181
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 10 to 140
Number of Intervals = 40

Dump Heights Less than 15 metre (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 22578
Min Data Value	= 1.01
Max Data Value	= 14.8
Sample Mean	= 6.84
Sample Std Dev	= 3.01

Distribution Summary

Distribution: Beta
Expression: 1 + 14 * BETA(1.77, 2.48)
Square Error: 0.000422

Best Fit Results

Function	*Sq Error*
Beta	0.000422
Erlang	0.00113
Gamma	0.00116
Normal	0.00129
Triangular	0.00232
Lognormal	0.00315
Uniform	0.00739
Exponential	0.0122
Weibull	0.103

Chi Square Test

Number of intervals = 38
Degrees of freedom = 35
Test Statistic = 87
Corresponding p-value < 0.005

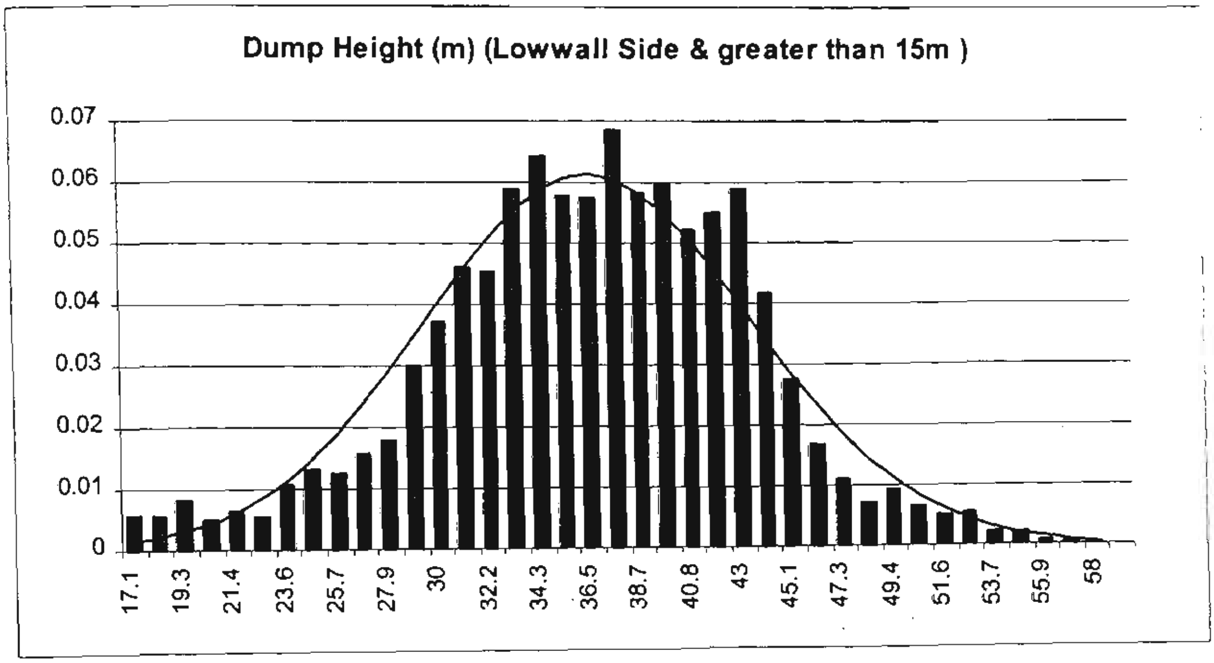
Kolmogorov-Smirnov Test

Test Statistic = 1.37e+131
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 1 to 15
Number of Intervals = 40

Dump Heights Greater than 15 metre (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 24549
Min Data Value	= 15
Max Data Value	= 57.6
Sample Mean	= 35.9
Sample Std Dev	= 6.99

Best Fit Results

Function	*Sq Error*
Normal	0.00127
Beta	0.00206
Triangular	0.00492
Gamma	0.00651
Erlang	0.00723
Lognormal	0.0123
Uniform	0.0211
Exponential	0.0312
Weibull	0.0993

Distribution Summary

Distribution: Normal
Expression: NORM(35.9, 6.99)
Square Error: 0.001271

Chi Square Test

Number of intervals = 35
Degrees of freedom = 32
Test Statistic = 460
Corresponding p-value < 0.005

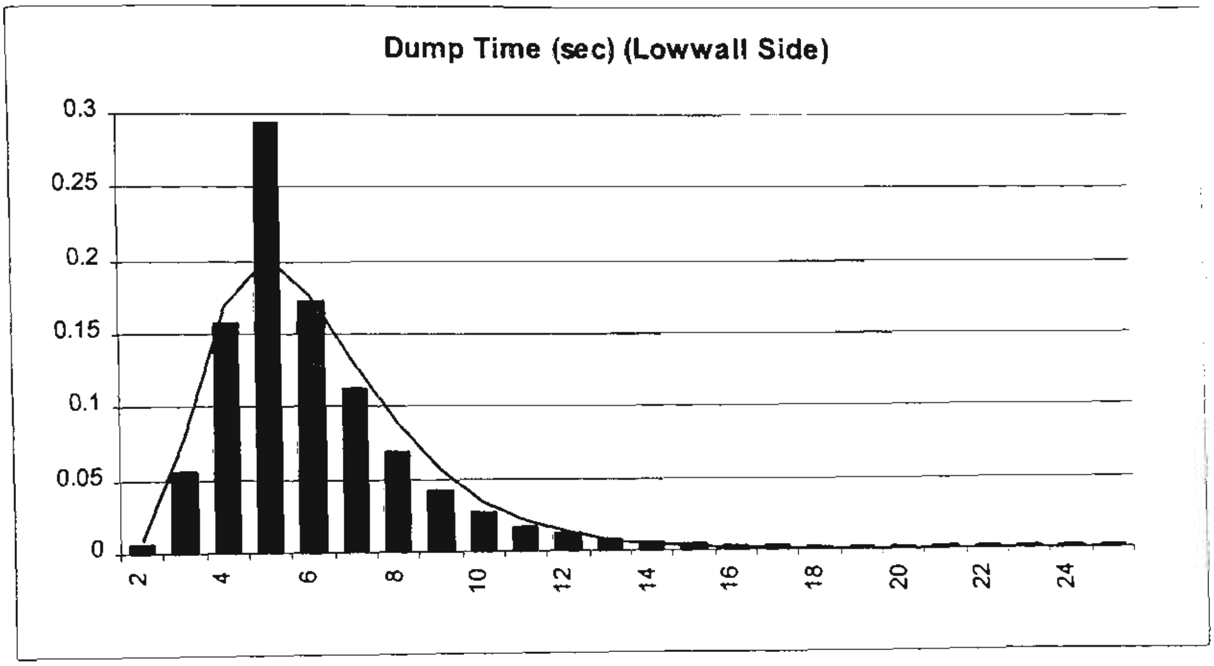
Kolmogorov-Smirnov Test

Test Statistic = 6.24e+251
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 15 to 58
Number of Intervals = 40

Dump Time (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47701
Min Data Value	= 1
Max Data Value	= 25
Sample Mean	= 6.2
Sample Std Dev	= 2.78

Best Fit Results

Function	*Sq Error*
Lognormal	0.0102
Erlang	0.0161
Gamma	0.0165
Poisson	0.0267
Beta	0.0307
Normal	0.0392
Triangular	0.0803
Exponential	0.112
Uniform	0.124
Weibull	0.193

Distribution Summary

Distribution: Lognormal
Expression: 0.5 + LOGN(5.66, 2.4)
Square Error: 0.010178

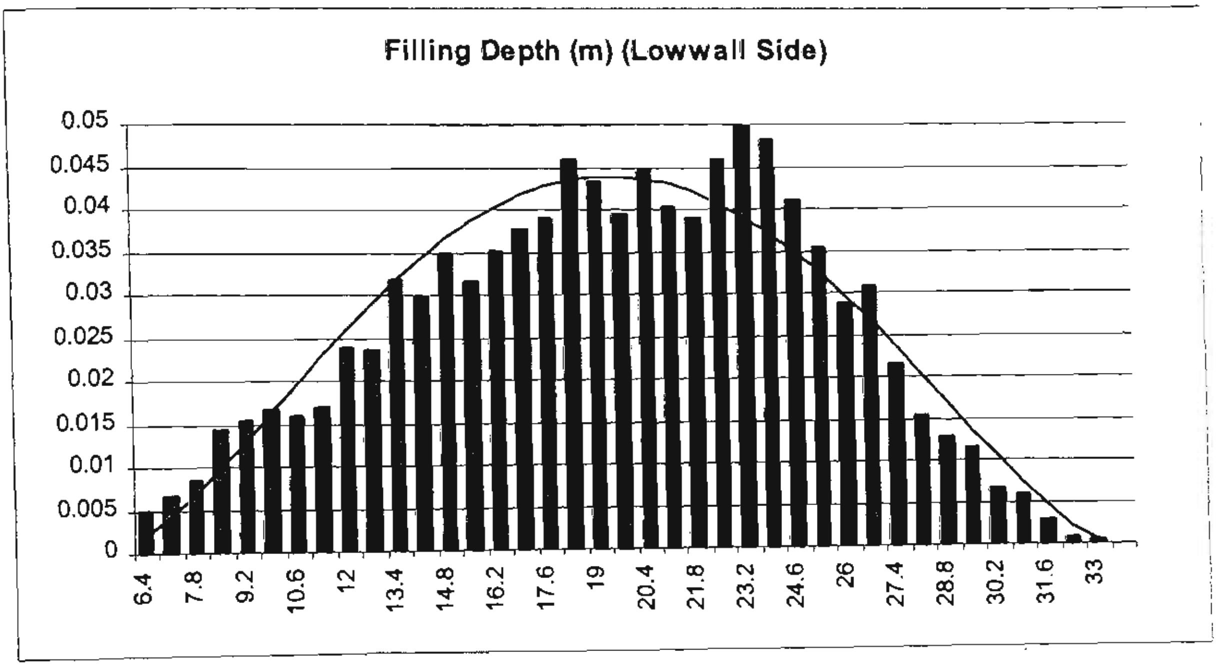
Chi Square Test

Number of intervals = 16
Degrees of freedom = 13
Test Statistic = 841
Corresponding p-value < 0.005

Histogram Summary

Histogram Range = 0.5 to 25
Number of Intervals = 25

Filling Depth (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47213
Min Data Value	= 5.05
Max Data Value	= 32.6
Sample Mean	= 19
Sample Std Dev	= 5.63

Best Fit Results

Function	*Sq Error*
Beta	0.000704
Normal	0.00113
Triangular	0.00114
Gamma	0.00358
Erlang	0.00377
Lognormal	0.0062
Uniform	0.00921
Exponential	0.0189
Weibull	0.0584

Distribution Summary

Distribution: Beta
Expression: 5 + 28 * BETA(2.6, 2.58)
Square Error: 0.000704

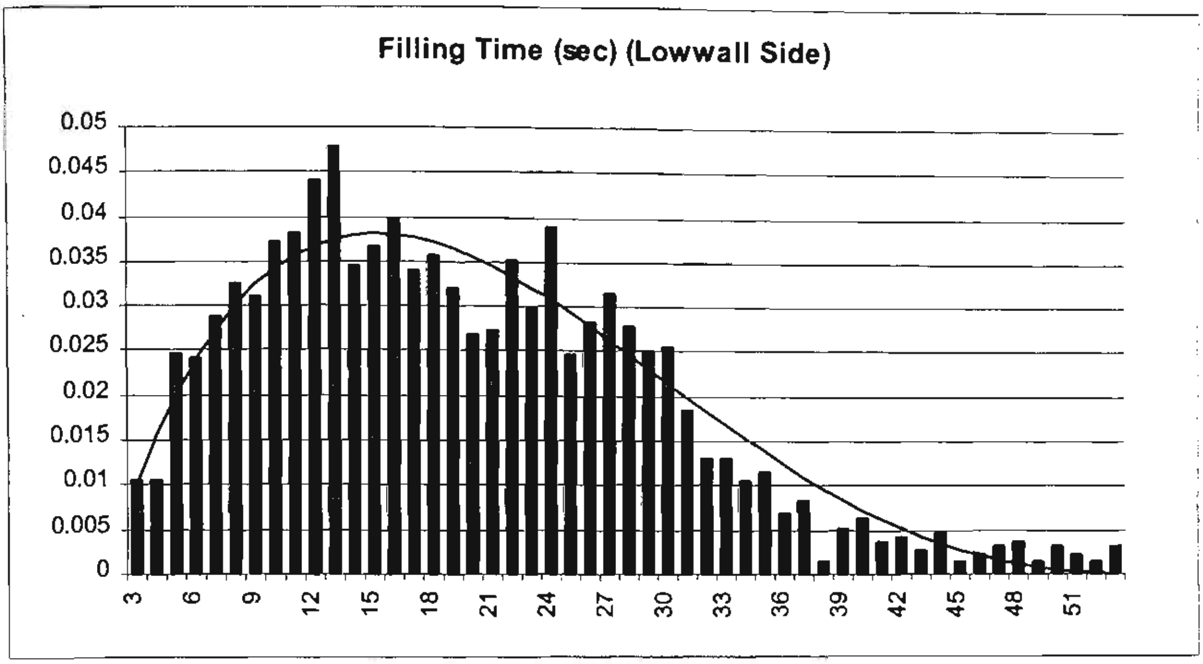
Chi Square Test
Number of intervals = 38
Degrees of freedom = 35
Test Statistic = 274
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 5.11e+064
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 5 to 33
Number of Intervals = 40

Filling Time (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 21905
Min Data Value	= 2
Max Data Value	= 55
Sample Mean	= 19.6
Sample Std Dev	= 10.2

Best Fit Results

Function	*Sq Error*
Beta	0.000797
Gamma	0.00105
Erlang	0.00116
Normal	0.00201
Weibull	0.00229
Lognormal	0.00231
Triangular	0.00232
Exponential	0.0106
Uniform	0.011
Poisson	0.0275

Distribution Summary

Distribution: Beta
Expression: 1.5 + 53.5 * BETA(1.91, 3.67)
Square Error: 0.000797

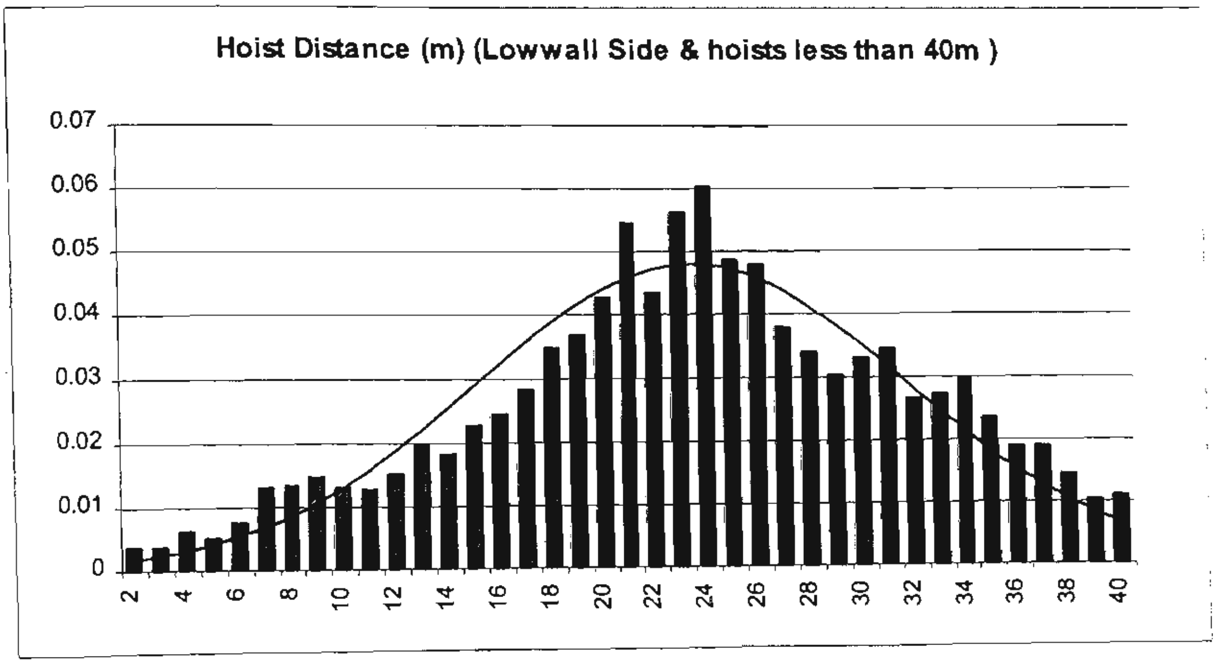
Chi Square Test

Number of intervals = 42
Degrees of freedom = 39
Test Statistic = 152
Corresponding p-value < 0.005

Histogram Summary

Histogram Range = 1.5 to 55
Number of Intervals = 55

Hoist Distances Less than 40 metre (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 23835
Min Data Value	= 0.21
Max Data Value	= 40
Sample Mean	= 22.9
Sample Std Dev	= 8.31

Best Fit Results

Function	*Sq Error*
Normal	0.00105
Triangular	0.00148
Beta	0.00164
Gamma	0.00357
Erlang	0.00383
Weibull	0.0053
Lognormal	0.00631
Uniform	0.00951
Exponential	0.0213

Distribution Summary

Distribution: Normal
Expression: NORM(22.9, 8.31)
Square Error: 0.001047

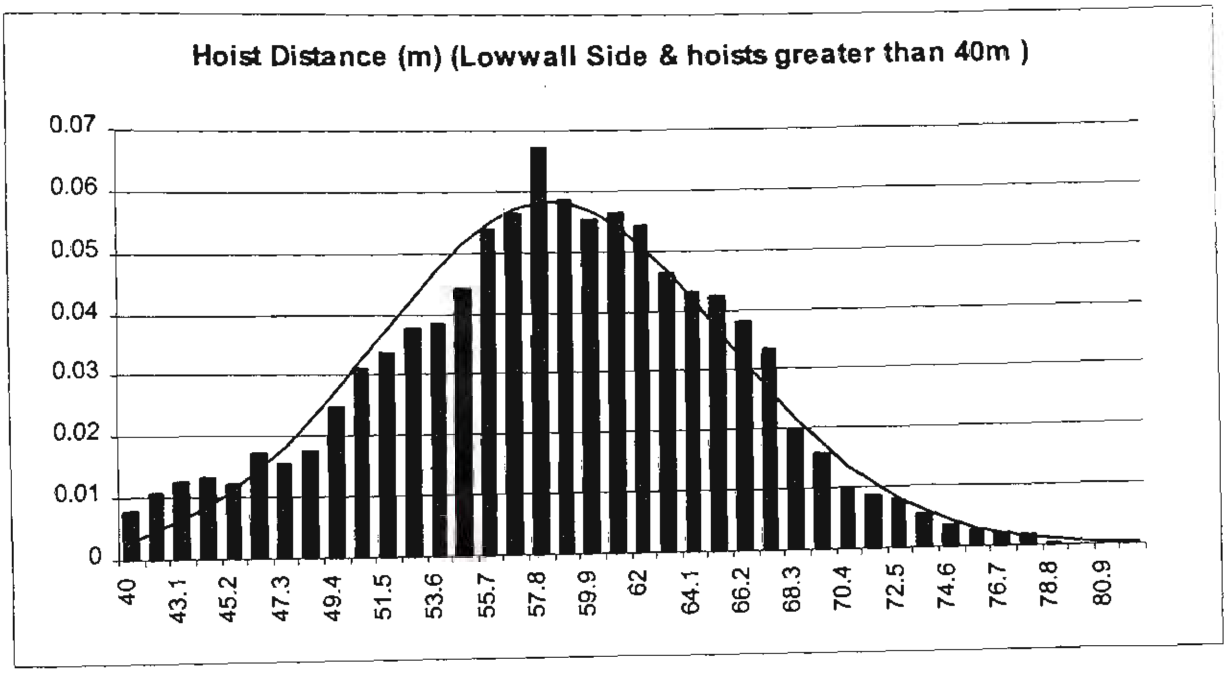
Chi Square Test
Number of intervals = 38
Degrees of freedom = 35
Test Statistic = 193
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 0.401
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 0 to 40
Number of Intervals = 40

Hoist Distances Graeter than 40 metre (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 24021
Min Data Value	= 40
Max Data Value	= 81.3
Sample Mean	= 57.7
Sample Std Dev	= 7.18

Best Fit Results

Function	*Sq Error*
Normal	0.000603
Beta	0.00157
Weibull	0.00241
Triangular	0.0028
Erlang	0.005
Gamma	0.00511
Lognormal	0.00998
Uniform	0.0165
Exponential	0.0235

Distribution Summary

Distribution: Normal
Expression: NORM(57.7, 7.17)
Square Error: 0.000603

Chi Square Test

Number of intervals = 35
Degrees of freedom = 32
Test Statistic = 146
Corresponding p-value < 0.005

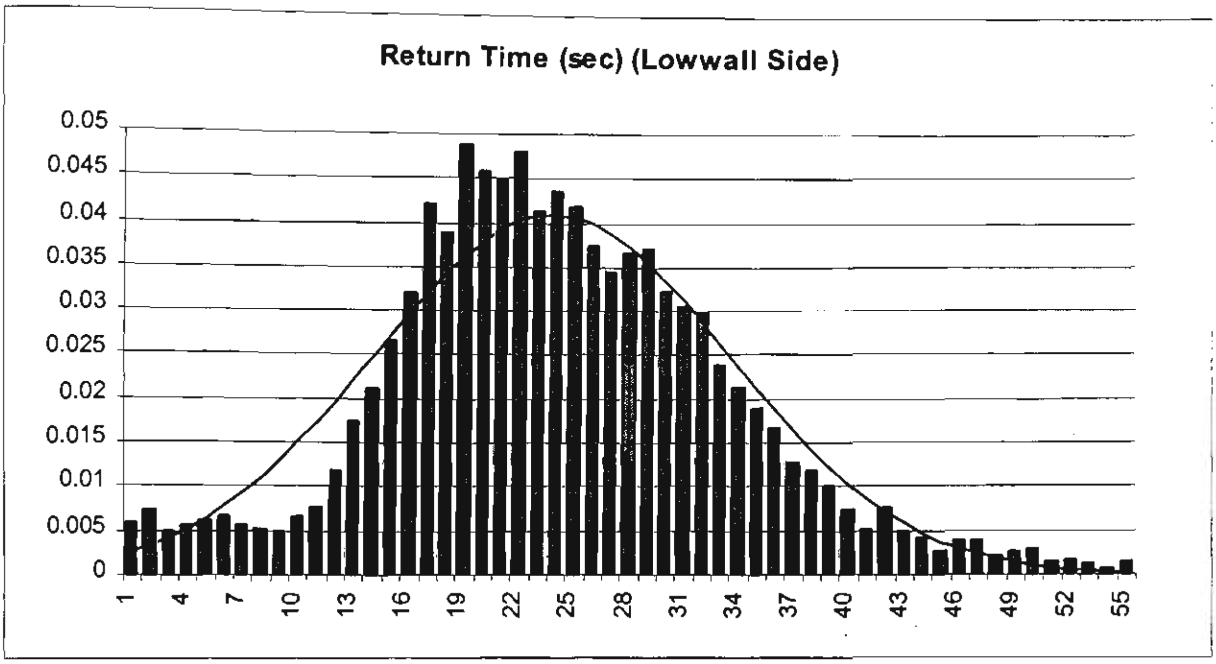
Kolmogorov-Smirnov Test

Test Statistic = 0.576
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 40 to 82
Number of Intervals = 40

Return Time (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47707
Min Data Value	= 0
Max Data Value	= 55
Sample Mean	= 23.7
Sample Std Dev	= 9.75

Best Fit Results

Function	*Sq Error*
Normal	0.00112
Beta	0.00206
Triangular	0.0032
Erlang	0.00421
Gamma	0.00438
Weibull	0.00792
Lognormal	0.00887
Poisson	0.0111
Uniform	0.0132
Exponential	0.0192

Distribution Summary

Distribution:	Normal
Expression:	NORM(23.7, 9.75)
Square Error:	0.001116

Chi Square Test

Number of intervals = 52
Degrees of freedom = 49
Test Statistic = 901
Corresponding p-value < 0.005

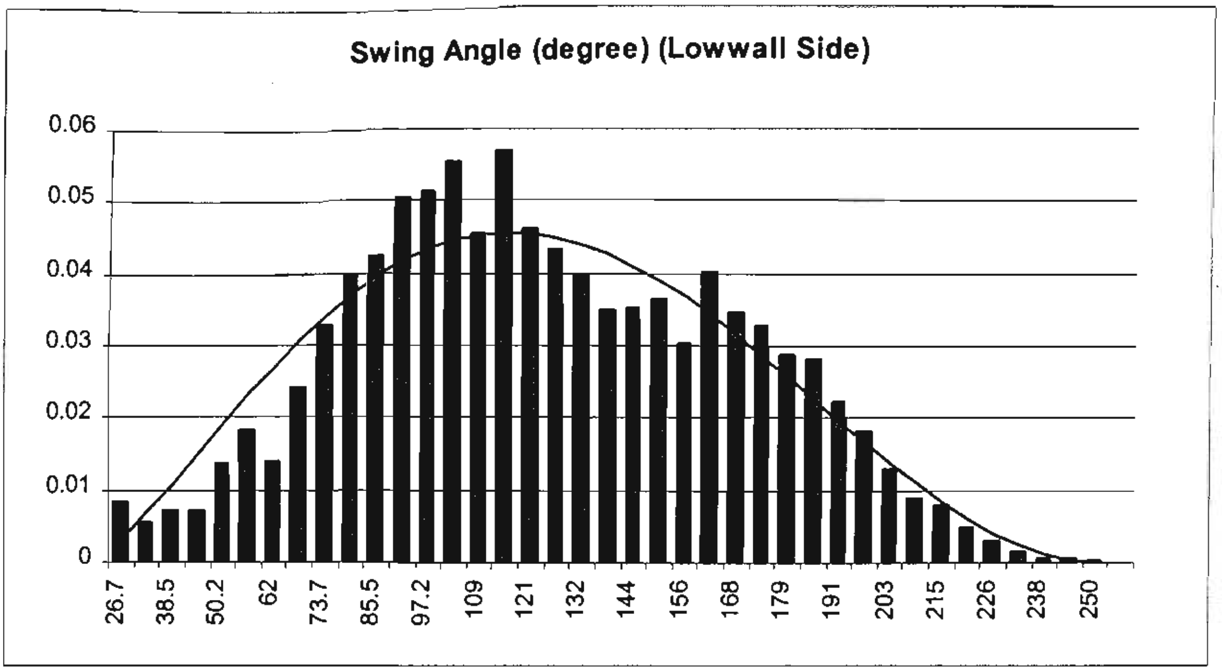
Kolmogorov-Smirnov Test

Test Statistic = 0.576
Corresponding p-value < 0.01

Histogram Summary

Histogram Range	= -0.5 to 55
Number of Intervals	= 61

Swing Angle (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47470
Min Data Value	= 15
Max Data Value	= 247
Sample Mean	= 120
Sample Std Dev	= 45

Best Fit Results

Function	*Sq Error*
Beta	0.00126
Triangular	0.00137
Normal	0.00148
Erlang	0.00395
Gamma	0.00395
Uniform	0.0117
Lognormal	0.0128
Exponential	0.0193
Weibull	0.0359

Distribution Summary

Distribution: Beta
Expression: 15 + 235 * BETA(2.56, 3.17)
Square Error: 0.001265

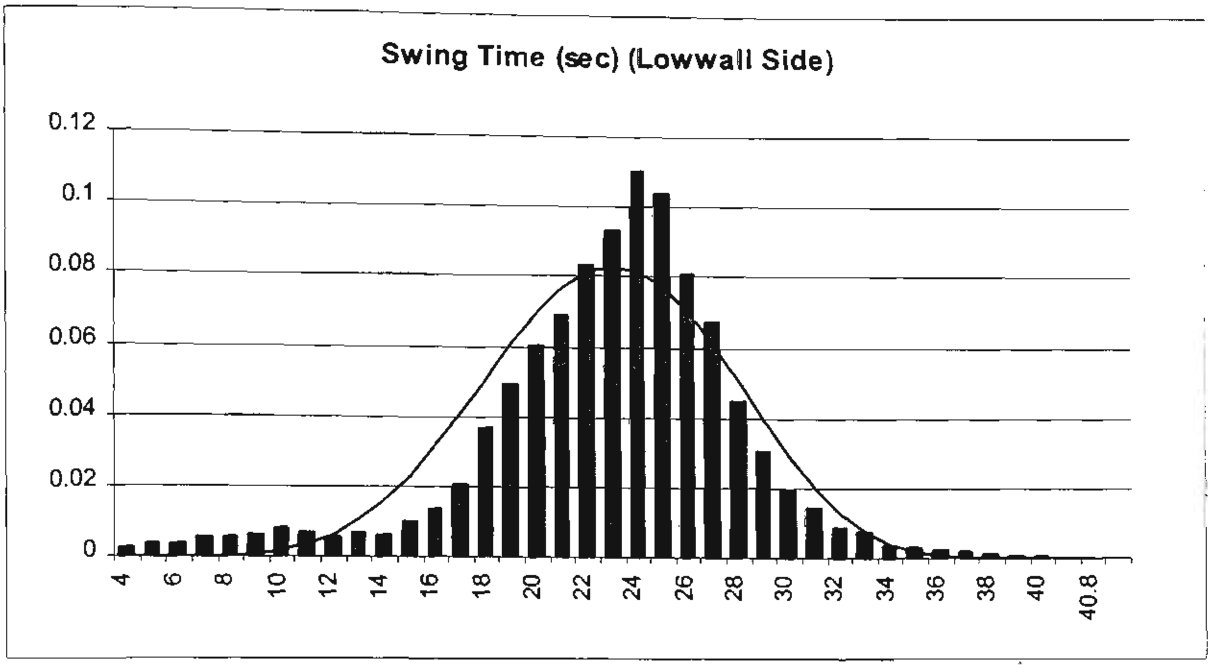
Chi Square Test
Number of intervals = 37
Degrees of freedom = 34
Test Statistic = 996
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 1.26e+169
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 15 to 250
Number of Intervals = 40

Swing Time (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47775
Min Data Value	= 3
Max Data Value	= 40
Sample Mean	= 22.9
Sample Std Dev	= 5.17

Best Fit Results

Function	*Sq Error*
Beta	0.00381
Poisson	0.00466
Normal	0.00487
Erlang	0.0132
Gamma	0.0133
Weibull	0.0146
Triangular	0.0177
Lognormal	0.0202
Uniform	0.0398
Exponential	0.0534

Distribution Summary

Distribution: Beta
Expression: 2.5 + 37.5 * BETA(8.24, 6.98)
Square Error: 0.003813

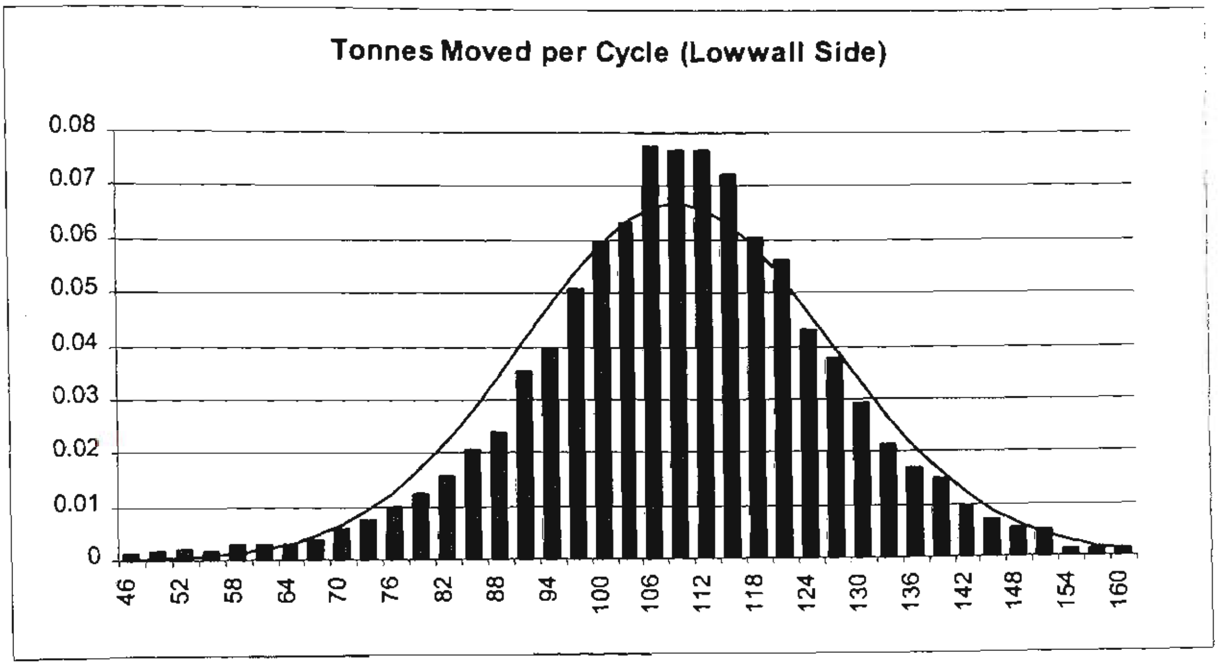
Chi Square Test
Number of intervals = 26
Degrees of freedom = 23
Test Statistic = 4e+003
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test
Test Statistic = 1.26e+169
Corresponding p-value < 0.01

Histogram Summary

Histogram Range = 2.5 to 40
Number of Intervals = 40

Tonnes Moved per Cycle (Lowwall Side Stripping)



Data Summary

No. of Data Points	= 47666
Min Data Value	= 40
Max Data Value	= 160
Sample Mean	= 107
Sample Std Dev	= 17.9

Best Fit Results

Function	*Sq Error*
Normal	0.00105
Beta	0.00201
Erlang	0.00419
Gamma	0.00426
Lognormal	0.00824
Triangular	0.00912
Weibull	0.0174
Uniform	0.0257
Exponential	0.038

Distribution Summary

Distribution:	Normal
Expression:	NORM(107, 17.9)
Square Error:	0.001054

Chi Square Test

Number of intervals = 34
Degrees of freedom = 31
Test Statistic = 511
Corresponding p-value < 0.005

Kolmogorov-Smirnov Test

Test Statistic = 0.0345
Corresponding p-value < 0.01

Histogram Summary

Histogram Range	= 40 to 160
Number of Intervals	= 40

APPENDIX E

"PIT OPTIMISATION OF THE VALIDATION CASE STUDY"

The objective of optimising the pit design is to maximise the productivity of the dragline operation and similarly the coal uncovering rates. In most of the situations the mine engineer only has control over the operating methods and minor changes to a small number of pit design parameters such as pit width or dragline working level. Changing these factors will alter the coal uncovering rates via changes in the rehandle, cycle time and walk time components.

To reduce rehandle in low wall passes, the dragline working level was kept at the minimum possible. The level of the low wall pad is firstly determined, by the amount of waste from first pass and a minimum level required to provide sufficient spoil room for waste from all three passes. To determine the optimum pit width, the strip width parameter was changed within a practical range of 40 to 70m. Repeating the dragline simulation, the productivity and different mining parameters were then calculated for each case. The impact of changes in the pit width on various operating parameters is shown in Figures E.1 through E.5.

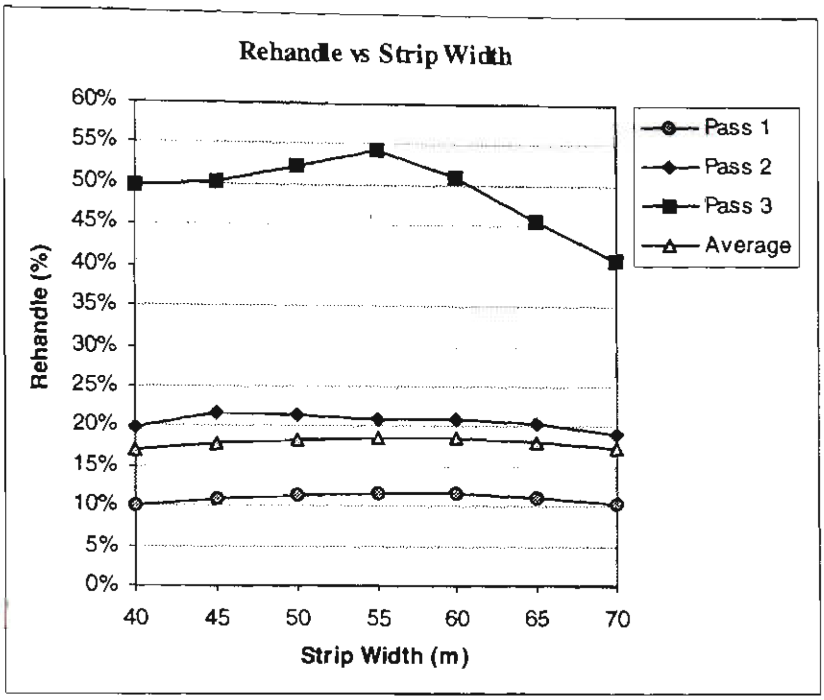


Figure E.1- Effect of the strip width on rehandle.

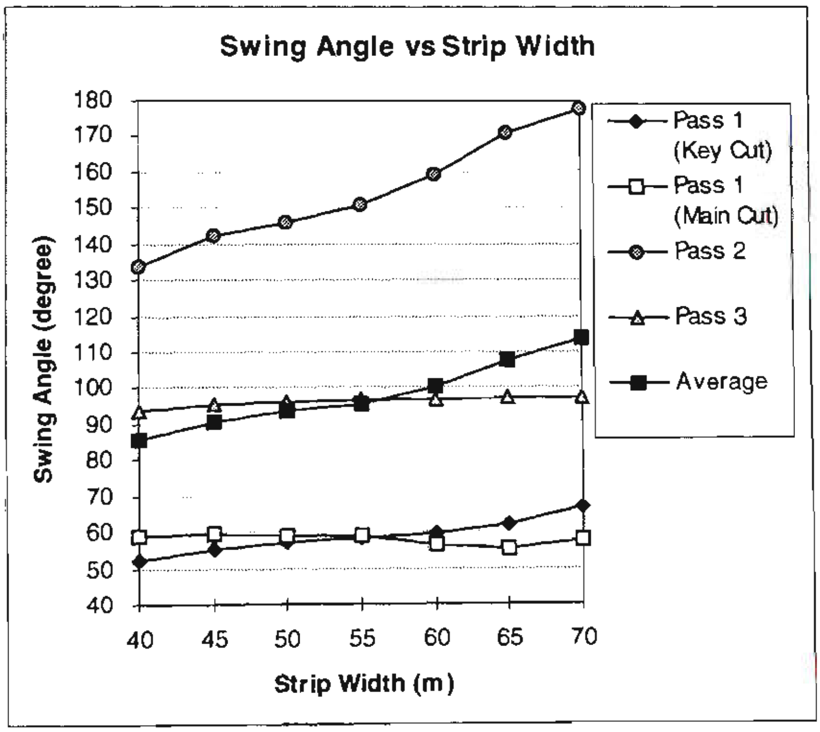


Figure E.2- Effect of the strip width on swing angle.

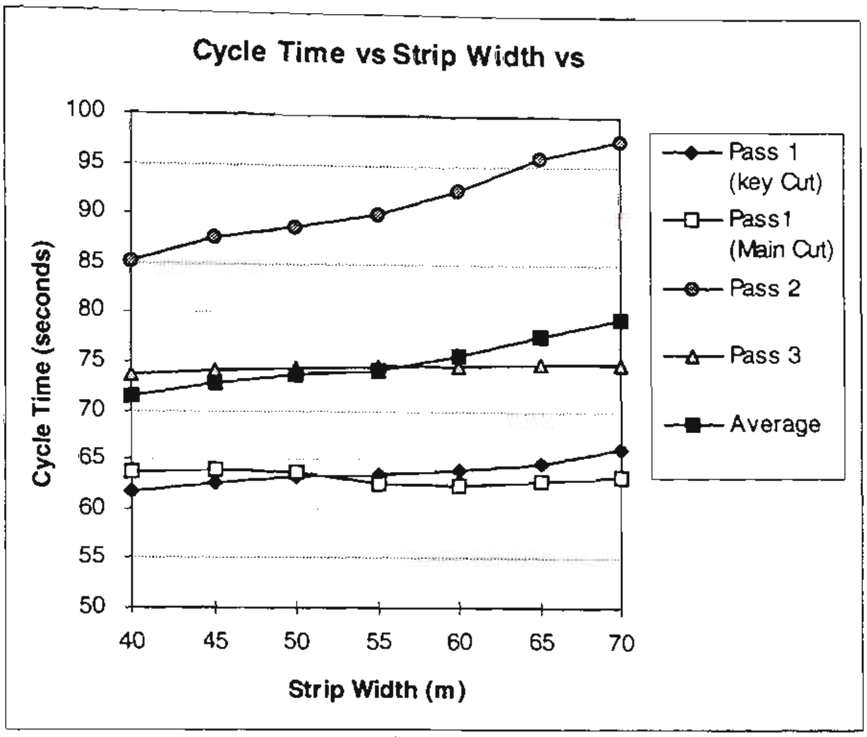


Figure E.3- Effect of the strip width on cycle time.

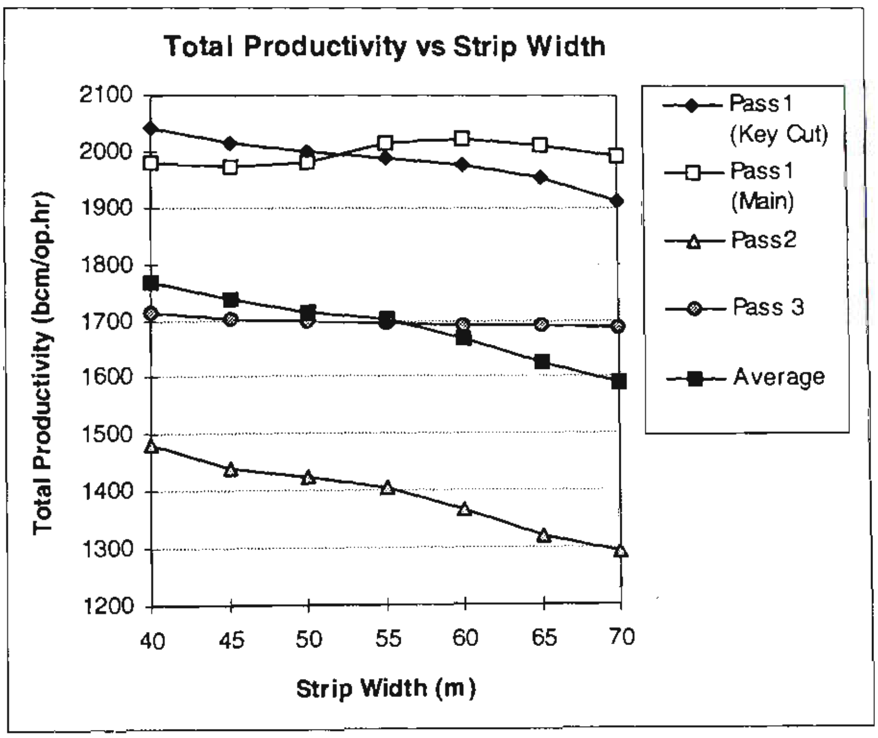


Figure E.4- Effect of the strip width on total productivity.

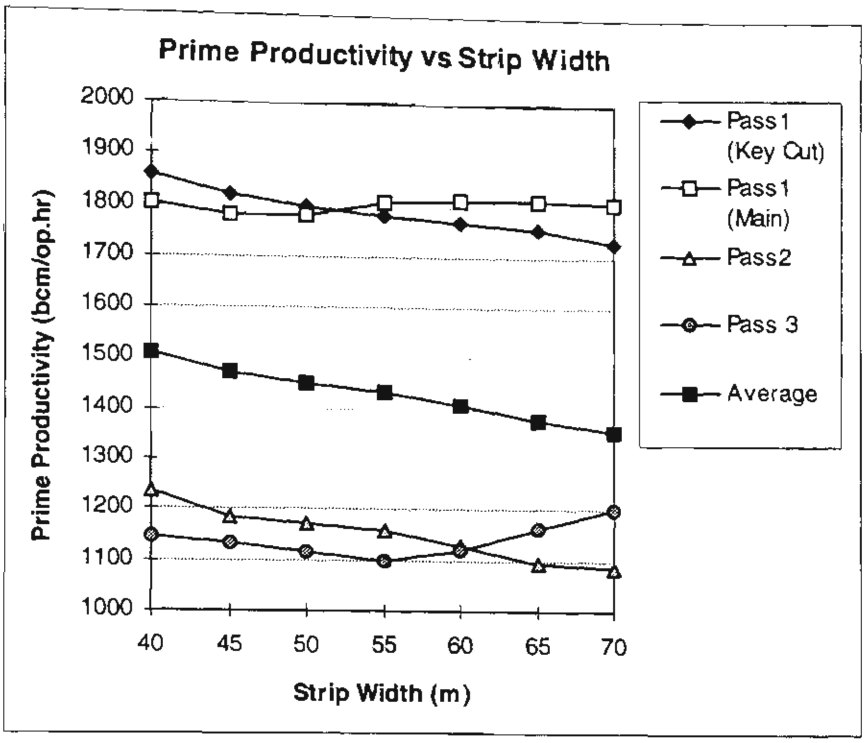


Figure E.5- Effect of the strip width on prime productivity.

The results show that although rehandle is decreased for wider strips, the total and prime productivity is decreased due to the longer swing angles and hence cycle times for those strips. This is especially true for the second pass where the longest swings are required. This pass comprises almost 40 percent of the total volume and 45 percent of total time spent to uncover coal seams for a strip width of 60m. As the strip width increases the proportion of volume and hence time spent in the second pass also increases. Figures E.6 and E.7 show the effect of strip width on the proportion of total volume and time for all the dragline passes.

From a combination of these factors it can be concluded that the narrower strips are more productive in this case. However, due to the coal mining constraints, a strip width of 50 metres was determined as an optimum strip width.

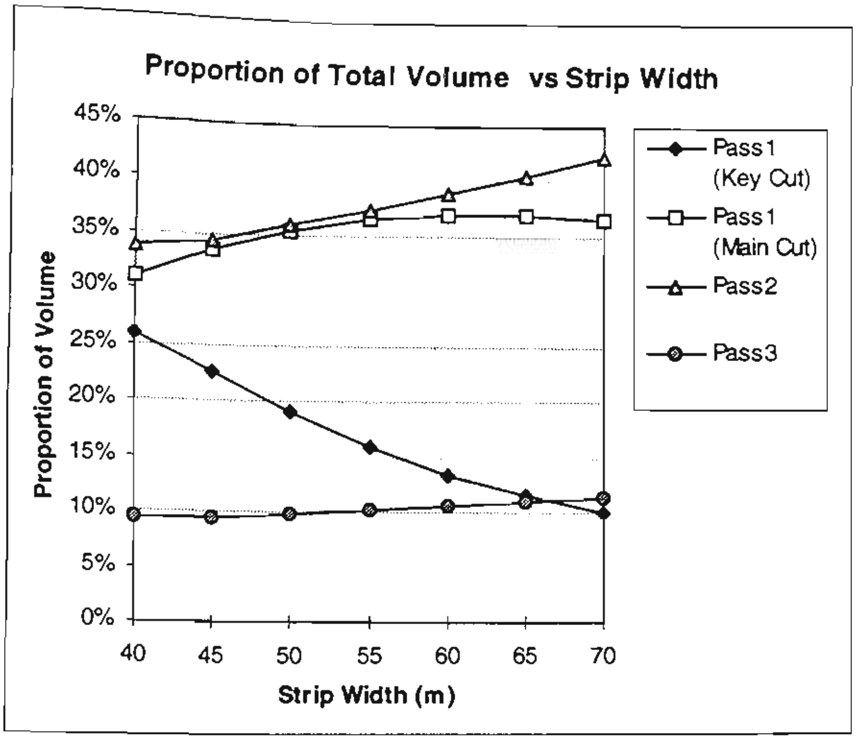


Figure E.6- Effect of the strip width on proportion of total volume.

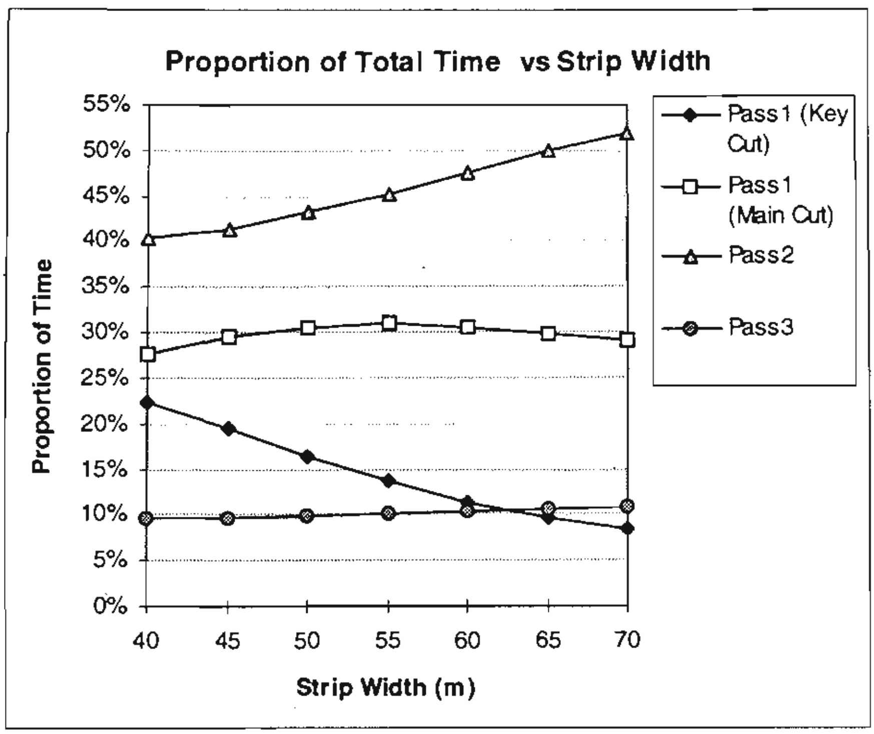


Figure E.7- Effect of the strip width on time spent in each pass.

Once the optimised pit layout for each method is identified, a great deal of information related to the operating parameters and productivity can be derived using the dragline simulation model. This detailed information may be used in analysing machine performance, scheduling and cost estimation procedures. The results of the dragline simulation and the productivity estimation for the optimum strip width (50m) are summarised in Table E.1. In this case a strip by strip approach has been used for the productivity calculation.

Table E.1- Results of simulation for 50m wide strips

	Pass 1		Pass 2	Pass 3	Average
CYCLE PARAMETERS	**Key Cut**	**Main Cut**			***(total)***
Swing angle (degree)	79.5	64.3	159.5	96.9	100.4
Hoist distance (m)	0.0	0.0	40.0	35.0	
Volume of prime (bcm)	35773.5	98447.5	103461.9	28133.5	*(265816.4)*
Volume of rehandle (bcm)	0.0	0.0	20692.3	11544.2	*(22803.5)*
Rehandle (including ramp rehandle)%	**0.0%**	**10.0%**	**20.9%**	**51.0%**	**18.6%**
CYCLE COMPONENTS (seconds)					
Swing time Loaded	18.2	16.5	28.7	20.9	22.3
Hoist time Loaded	0.0	0.0	17.4	12.2	8.1
Dump time	8.0	8.0	6.0	6.0	6.8
Hoist Pay time	0.0	0.0	12.1	8.4	5.6
Return time	18.4	16.8	27.8	20.0	21.8
Drag to fill time	14.0	14.0	18.0	18.0	16.0
Theoretical cycle time	58.6	55.3	80.5	64.9	65.8
Operator adjustment factor	1.15	1.15	1.15	1.15	1.15
Adjusted cycle time	67.4	63.6	92.6	74.6	75.7
HOURLY PRODUCTION					
Cycles per dig hour	53.4	56.6	38.9	48.2	49.2
Bucket capacity (bcm)	48.0	48.0	48.0	48.0	48.0
Material swell factor	1.30	1.30	1.30	1.30	1.30
Bucket fill factor	0.95	0.95	0.95	0.95	0.95
Prime volume moved per cycle (bcm)	35.1	35.1	35.1	35.1	35.1
Prime volume moved per dig hour (bcm/hr)	**1874**	**1987**	**1364**	**1692**	**1646**
% of Total strip volume	13.5%	37.0%	38.9%	10.6%	*(1.00)*
% of Total time	11.4%	30.6%	47.6%	10.4%	*(1.00)*
Weighted average dig rate (bcm/dig hr)			**1646**		

Examples of the information derived from the geological model and the dragline simulations regarding the dragline working depths and coal thicknesses for each pass are shown in Figures E.8 and E.9. It can be seen that as the sections progress towards the end of the pit, the first pass depth increased. From section 32 (approximately 1 km far from the southern ramp) a pre-stripping operation is needed due to insufficient spoil room. Dragline lowwall pad level is controlled by the first pass depth and this level affects the rehandle for both lowwall passes. Figure E.10 illustrates how rehandle is changed along the strip.

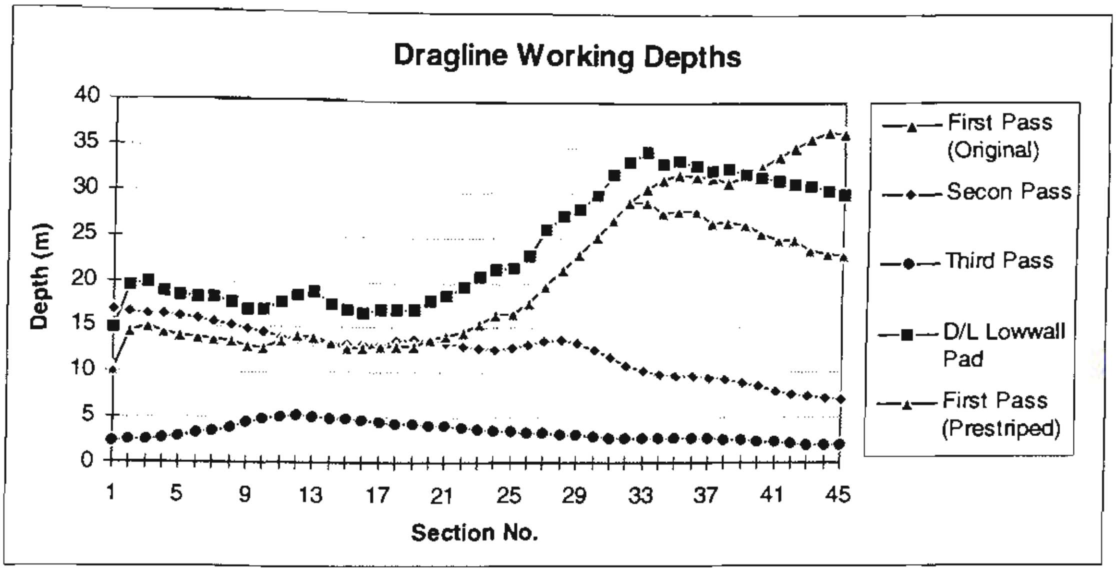


Figure E.8- Dragline working depths in each section.

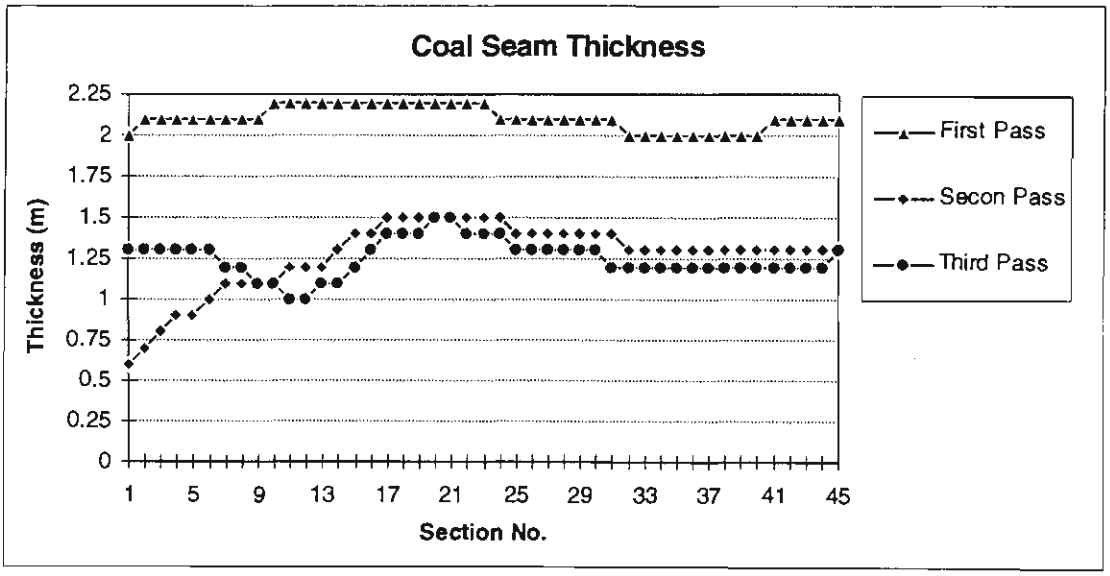


Figure E.9- Coal seam thickness in each section.

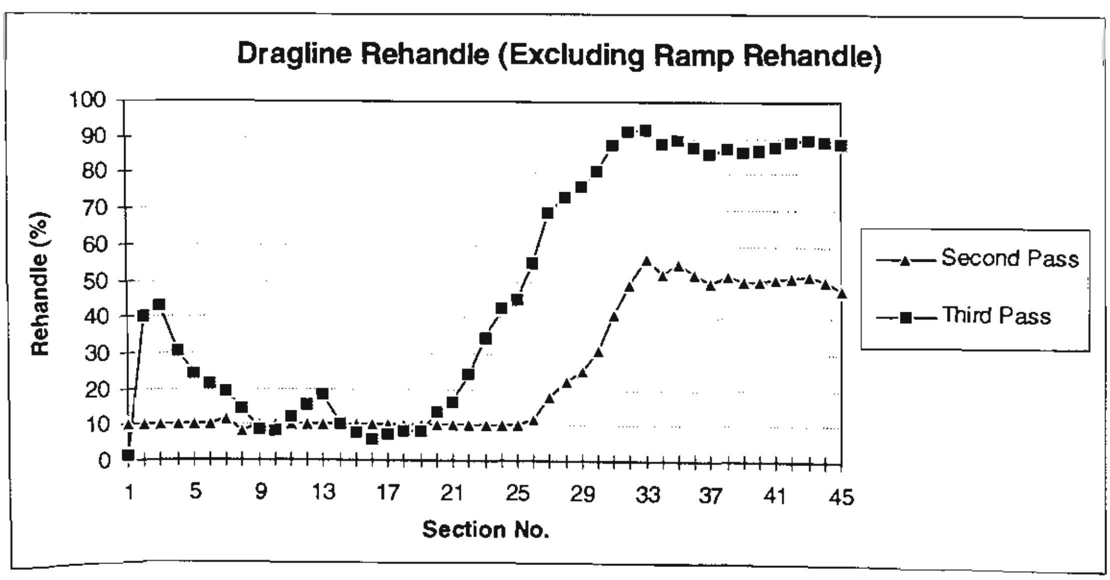


Figure E.10- Dragline rehandle in each section.