EIS 447

AA052395

Drayton Coal Mine : mine development proposal

DRAYTON CO-VENTURE

DRAYTON COAL MINE

MINE DEVELOPMENT PROPOSAL

Thiess Bros. Pty. Limited
The Shell Company of Australia Limited

4289001 2

THIESS BROS. PTY. LIMITED

AND

THE SHELL COMPANY OF AUSTRALIA LIMITED

DRAYTON CO-VENTURE

DRAYTON COAL MINE

MINE DEVELOPMENT PROPOSAL

JUNE 1978

CONTENTS

FIGURES

TABLES

APPENDICES

SUMMARY

This development proposal outlines the conclusions of a detailed feasibility study completed in January 1978 which investigated the extraction, preparation and transportation of coalfrom the proposed Drayton Coal Mining Lease. The open cut mining proposal, representing the optimum economic development in both resource utilisation and commercial terms, is summarised below:

- *Location*

  The resource is located approximately 12 km. south of the town of Muswellbrook and adjacent to the existing Bayswater No. 2 Open Cut Coal Mine.

- *Coal Resource*

  Within the lease area coal reserves are:

Measured	96.8 million tonnes
Indicated	42.9 million tonnes
Inferred	79.0 million tonnes
TOTAL	**218.7 million tonnes**

- *Open Cut Coal Reserves*

  Coal reserves available to open cut mining are as follows:

In-situ coal	57,858,000 tonnes
In-situ overburden	225,249,000 m^3
Raw Coal to be mined	52,943,000 tonnes
Overburden to be removed	
- Shovel	119,533,000 m^3
- Dragline, prime	109,510,000 m^3
- Dragline, rehandle	44,247,000 m^3
- **TOTAL**	**273,290,000 m^3**

  Ratios (m^3/tonne)

  - $\frac{\text{In-situ Overburden}}{\text{In-situ coal}}$ 3.89
  - $\frac{\text{Total Overburden to be removed}}{\text{Raw coal mined}}$ (excl. rehandle) 4.33

- *Coal Quality*

  The coal is primarily a high quality steaming coal with low ash, high specific energy and medium sulphur content.

- *Mining Method*

  The multi seam resource will be recovered using a combination of an electric walking dragline of 44.3 m^3 bucket capacity plus several shovel/dump truck fleet combinations employing both electric and diesel hydraulic shovels.

- *Mine Life and Planned Production*

  The proposed development will have an assured life of 22 years although additional reserves are available to extend the mine life beyond the year 2000. Raw coal production levels will be attained in the following key stages:

  - stripping overburden by electric shovel — 1.0 million tonnes per annum
  - after commissioning electric dragline — 2.9 million tonnes per annum
  - peak production rate — 3.5 million tonnes per annum

- *Environment*

  The mine is located in an area with a history of open cut coal mining and cleared land cattle grazing. A Draft Environmental Impact Study in accordance with State Government Statutory requirements is being prepared in association with the mining proposal. Consultants, Dames and Moore, have been engaged to carry out this work on behalf of the Co-Venture. A preliminary review of environmental factors has been completed by Dames and Moore.

- *Marketing*

  A total of 52.9 million tonnes of raw coal is to be mined. In accordance with marketing requirements, coal handling and preparation plant designs have been developed for up to three product types. Two alternative designs cater for:-

	Indicative Annual Production (millions of tonnes)
- **STEAM COAL PRODUCT**	
Steaming coal products comprising a "high" ash (10% - 15%) and a "low" ash (less than 10%) type.	2.9
- **STEAM COAL/COKING COAL PRODUCT**	
Steaming coal products of "high" and "low" ash, as above, plus	2.2
A hard/soft coking coal product.	0.6

- *Start of Development*

  Mine development will commence in the financial year 1979 - 80 with peak coal production being achieved in 1990. This production schedule is dependent on the electric dragline being commissioned during 1983. Conceptual design of the coal handling plant and mine establishment complex has been completed by Crooks, Michell, Peacock, Stewart Pty. Ltd. (C.M. P.S.), Engineering Consultants, in association with the co-venture study team. Design of these facilities will be refined after testing of further bulk samples.

  The target development schedule is presented in Figure 22.

1. INTRODUCTION

Interest in the Drayton Mine area, formerly known as the Balmoral area, was established in 1951, when Thiess Bros. Pty. Ltd., lodged exploration applications in accordance with the provisions of the New South Wales mining legislation. Since that time, exploration programmes conducted under the auspices of the Joint Coal Board, the New South Wales State Electricity Commission and Thiess Bros. Pty. Ltd. have been carried out. During 1977, The Shell Company of Australia entered into a co-venture agreement with Thiess and acquired a 44.5% interest in the proposed Drayton Mine.
The Drayton Co-Venture participants have purchased the surface rights to 874.8 hectares and are negotiating for additional land purchases within the proposed lease area. In February 1977 an authorisation was obtained for additional exploration activities which includes an excavation to enable bulk coal samples to be extracted.

Since the lodging of the initial applications, Thiess has carried out a number of investigations aimed at increasing the detailed knowledge of the area in respect of:

- the geology of the coal measures,
- the reserves available for open cut and underground mining methods,
- the quality of the saleable coal product which could be offered for export,
- applicable mining method alternatives as well as the associated coal handling, benefication and transportation options.

Following the signing of the Thiess/Shell Co-Venture agreement in 1977, it was decided to proceed to a detailed study of the Drayton Area. The first stage of this work was the preparation of the Drayton Geological Report in the last quarter of 1977 and the second stage was the completion of a definitive Mining Feasibility Study in January 1978. An excavation to allow extraction of bulk samples of the Balmoral seam commenced in February 1978 and will be completed in July 1978.

1.1 LOCATION

The Drayton Mine area is located in the Upper Hunter Valley district of New South Wales, approximately 10 kilometres south-east of the town of Muswellbrook, 125 kilometres from the city of Newcastle, and 290 kilometres north of Sydney. (Refer Figure 1) The closest major settlement is Muswellbrook, a well established expanding rural town with a population of approximately 10,000 people.

Transport services between Muswellbrook and Sydney are via a major highway, a rail link and air services to Scone (24 kilometres north of Muswellbrook), and Singleton (40 kilometres south of Muswellbrook). The Drayton area is situated close to the main Northern Line of the

New South Wales railway system. A site close to Antiene Station, a distance of 8 kilometres by road from the mine area, has been chosen for the construction of a rail loading facility. The rail distance from the proposed Antiene siding to the Port of Newcastle is 114 kilometres.

Access to the site is made from the east via an all-weather unsealed road, which adjoins the New England Highway, 11 kilometres south of Muswellbrook. The Drayton Mine area is traversed NW-SE by a bitumen sealed stockroute road (Refer Figure 3).

1.2 TOPOGRAPHY AND CLIMATE

The Drayton area is characterised by a central N-S to NW - SE trending ridge along which elevations above sea level generally range from 250 to 300 metres (rising to a maximum 330 metres), and from which a number of watercourses drain. Elevations above sea level along the eastern boundary of the Drayton Mine area average approximately 180 metres, falling to a minimum 160 metres on the ash dam. Most of the Drayton area is cleared or semi-cleared and used for general grazing purposes. A small area near the proposed initial mining area is subject to occasional cultivation and a few irregular scrub timber stands, mainly on the steeper slopes, are scattered through the area. The area experiences a warm temperate climate with hot maximum summer temperatures (mean maximum for December 29.1^{o} C.), and cold to mild winter temperatures, (mean minimum for July 1.8^{o} C.). Annual rainfall averages approximately 600 mm. with heavier falls generally occurring during the summer months November - March.

2. GEOLOGY

Regionally, the Drayton Mine area lies in the Muswellbrook-Balmoral inlier of the Lower Permian, and is situated in the extreme NE of the Permian-Triassic Sydney Basin, some 10 kilometres west of the Hunter Thrust System, which defines the NE boundary of the basin.

The Lower Permian Greta Coal Measures in the Muswellbrook-Balmoral area, crop-out along the approximately north-south trending crest of the Muswellbrook Anticline, a major gently southwards plunging structure which traverses the Drayton Mine area. The Greta Coal Measures in the area are subdivided into two formations:-

- The upper Rowan Formation, approximately 115 metres thick, consists of sandstones, siltstones, shales and coal seams. Figure 5 illustrates the two respresentative stratigraphic columns of the Greta Coal Measures in the Drayton Area. The major coal seams occuring include the Brougham, Grasstrees, Puxtrees and Balmoral series. The Puxtrees series includes the Thiess coal split.

- The lower Skeletar Formation, approximately 100 metres thick, consists of tuffaceous and pelitic shales and clays, cherts, and rhyolites. The formation is overlain by the Rowan group.

Many of the Upper coal seams in the Muswellbrook Anticline area have been impaired by the effects of faulting and intrusive igneous sills and dykes. The most significant fault affecting initial mine planning at Drayton is a NNE trending, normal, strike slip fault, located approximately 300 metres west of the Muswellbrook Anticline axis. The throw of the fault ranges from 10 to 20 metres with a downthrow to the West.

The effect of igneous activity in the Drayton area is confined to intrusive sills and associated neighbouring coal seam alteration. Intermediate to basic intrusive sills occur in the far northern, and north western and south eastern corners of the area. However the Balmoral coal member which contains the major part of the coal reserves at Drayton is completely unaffected by igneous activity within the limits of the proposed mining area.

2.1 COAL RESOURCES

Prospecting operations in the area date back to 1952. Exploratory drilling to date totals 311 holes with approximately 19,500 metres of drilling including some 13,700 metres of coring. Three shafts have been sunk to obtain bulk coal samples for testing. The mineable coal seams present, together with their average thicknesses, are listed from top to bottom as follows:-

Series	Seam	Identification Code of Mineable Section	Average Thickness (metres)
Brougham	Brougham Upper Split	R2	1.6
	Brougham Lower Split	R3	1.6
Grasstrees	Grasstrees Middle Split	G4	1.4
	Grasstrees Lower Split	G5	0.7
Puxtrees	Thiess Seam	T2	2.3
	Puxtrees Main Split	P2	1.9
Balmoral	Balmoral Savoy Split	B1	1.8
	Balmoral Main Split	B3	5.8

In the southwest part of the mine area (the West Pit), the Balmoral Main Split divides forming the Balmoral Main Split Upper (B31) and Balmoral Main Split Lower (B32) seams with average thicknesses of 3.3 metres and 2.7 metres respectively. The Balmoral Savoy Split also divides in the same area forming the Savoy Split Upper (B11) and the Savoy Split Lower (B12) with average thicknesses of 1.2 metres and 0.8 metres respectively.

Total measured, indicated and inferred geological coal reserves within the Mining Lease Boundary are summarised in Table 1. The selected open-cut mining areas are contained within the measured reserve areas of the East and West Block, and the mining reserves are derived from the major coal seam horizons as follows:-

Series	% of Mining Reserves
Brougham	9
Grasstrees	6
Thiess	11
Puxtrees	9
Balmoral	65

Laboratory testing of the Drayton coal seams has been conducted by a number of organisations, including:-

- Joint Coal Board
- Commonwealth Scientific Industrial Research Organisation (CSIRO)
- Australian Coal Industry Research Laboratories (ACIRL)
- Yawata Steel Mills (Japan)

A summary of the results from these testing programmes is given in Table 11. These allow the following conclusions to be drawn in respect to the coal resource potential of the Drayton Area:

- Coal seams other than the Thiess seam are high quality steaming coal with low ash, high specific energy and medium sulphur contents.
- The Thiess seam is a high volatile coking coal.
- The petrographic composition of each of the seams overlying the Balmoral series indicates that they may possess some soft coking properties. This point is to be subject to further investigation.

Regionally, the Upper Hunter Valley is an important coal resource area and many opencast and underground mining operations have been developed in the last 20 years. Figure 2 is a map showing the distribution of the mining activity already established in the neighbourhood of the Drayton Mine Area.

3. MINING PROPOSAL

Three areas, the North West, the West, and the East, have been identified as open cut mining blocks. These mining blocks have the following general characteristics:-

- Each contains multiple seams with the East and West blocks containing up to seven mineable seams including some with multiple splits.
- The thinner seams occur in the upper section of the measures, e.g. the Grasstrees series,
- The thickest coal seam, the Balmoral, is the lowest in the sequence,
- Seam dips range between 0° - 14°.

A flexible, selective mining method is essential to ensure maximum extraction of reserves and to allow the production of saleable run-of mine (raw) coal. A number of alternative mining methods were studied and it was concluded that the Drayton area should be mined as an open cut with three separate pits employing a combined shovel/truck and dragline operation for overburden removal. These are hereinafter referred to as the West Pit, the North West Pit, and the East Pit, with the latter two Pits being contiguous.

3.1 SUMMARY OF EXTRACTION SEQUENCE

The extraction sequence to be followed will be:-

- Establishment of a shovel and truck overburden stripping operation in the North West Pit.
- A second shovel and truck stripping operation will then develop the West Pit box cut in preparation for the introduction of an electric walking dragline.
- The dragline will later be complemented by the shovel and truck operation exposing the upper seams as required.
- After completing the overburden stripping in the North West Pit, the shovel and truck combination will open up a box cut on the anticlinal axis in the East Pit, and then pre-strip down the eastern flank of the anticline in preparation for the dragline sidecasting operation.
- When the West Pit stripping is completed the dragline will move to the East Pit initially cross-dragging from the box cut down the west flank of the anticlinal axis and later working eastwards.

3.2 PIT LAYOUT

Pit layout was determined essentially by the structure of the deposit to be mined, the geometry of the major extraction equipment alternatives, and the method of operation. Strike cuts were preferred with strata dipping into the highwall where possible because they provide the longest uninterrupted working horizon, immediate access to the coal with the lowest overburden ration suitable for dragline overburden stripping and a reasonable assurance that no significant geotechnical problems would occur.

NORTH WEST PIT

Cuts were oriented as close as possible to the strike of the seam with the initial box cut being located in the lowest overburden to coal ratio area. The cut widths were determined by shovel and truck dimensions and method of operation. The main access ramp into the pit runs in a W-E direction and overburden excavated in the ramp forms part of the pre-strip overburden for the East Pit.

WEST PIT

The West Pit cuts were orientated parallel to the western Mining Lease boundary as this permitted:

- Maximum cut lengths;
- The ability to extend the north and south pit limits should further exploration prove this feasible;
- A central access ramp splitting the cuts into north and south;
- The opportunity to work the higher ratio south cuts in conjunction with the lower ratio north cuts;
- Easier scheduling of in-pit operations;
- Elimination of shovel and truck pre-stripping work during the first two years of dragline operation;
- Progressive rehabilitation to be carried out from the Mining Lease boundary.

EAST PIT

The East Pit cuts were orientated in a N-S direction as this permitted:

- Maximum cut lengths for ease of equipment scheduling;
- The option of extending south through the fault if reserves are proven to be economic;
- The option of extending the pit beyond the eastern limit, if extension proves economic.
- A central access ramp to the Balmoral seam;
- Minimum box cutting to ensure continuity of coal during East Pit development.

3.3 MAJOR EQUIPMENT SELECTION

Major equipment capacity and geometry was selected after detailed study of the alternative mining strip layouts, mining schedule and the economics applicable to alternative available equipment.

DRAGLINE SELECTION

After detailed examination of:-

- The average production rates and geometry, in particular dumping heights and operating radii, of available draglines;
- The proportion of overburden (and rehandled overburden) handled by the dragline compared to the total overburden to be mined;
- The average overburden/coal ratio in the mine area;
- The in-pit operations such as drilling and blasting, coal and parting removal that would also have to be scheduled along with the dragline operation.

It was determined that a dragline of $44m^3$ ($58yd^3$) bucket capacity would provide the lowest operating unit cost for a mine production in the range of 2 - 3 m.t.p.a.

STRIPPING SHOVEL SELECTION

To maintain a flexible, selective mining method several shovel/dump truck fleet combinations employing both electric and diesel hydraulic shovels are used to excavate the overburden from the following sources:

- All the overburden in the North West Pit;
- Overburden from the initial box cuts in the West and East Pits;
- Pre-strip overburden in advance of the dragline operation.

For the initial mining operation commencing in the North West Pit a $7.6m^3$ hydraulic excavator was selected because it was:

- Best suited to the desired early production rate,
- Easily transferrable to the West Pit to open up the initial box cut without the need for site electrical reticulation,
- Best suited to coal loading operations in the West Pit after the introduction of the dragline,
- Suitable for the overburden pre-stripping necessary in the West Pit after the first two years operation with the dragline.

A $16m^3$ ($21yd^3$) electric shovel was selected for introduction in the second year of operation the North West Pit, to allow increased coal output.

For operations in the East Pit the proportion of shovel/truck overburden increases and both the $7.6m^3$ hydraulic excavator and the $16m^3$ electric shovel are required to pre-strip overburden from the upper seams in advance of the dragline.

Any subsequent reference to specific manufacturers' equipment models is to assist readers to understand the nature and type of equipment, and in no way indicates any commitment to, or preference for, particular manufacturer's products.

3.4 MINING RESERVES

The determination of the overall pit layout and the choice of strike-orientated cuts allowed a preliminary mine plan to be prepared, from which reserves were calculated on a cut by cut basis. In calculating the mining reserves it was necessary to identify the seams which were mineable either individually or as a combination of seam splits, where both the coal and parting would be mined together. Two principal constraints were applied in the vertical selection procedure, namely coal quality, and the ability to mine a particular coal/parting interval with the equipment available. Specifically the constraints applied were:-

- A simple coal quality constraint whereby mining seam horizons are to have a maximum 15% raw coal ash.
- A minimum mineable seam thickness of 0.75 metre.
- If the parting between two mineable seams is less than 0.5 metre, the seams and parting are to be combined to become one mining horizon.
- If the parting between two mineable seams is greater than 0.5 metre thick, the seams are to be treated as separate mining horizons.

The vertical seam selection procedure allowed the mineable seam intervals to be identified and this information, together with the geological interpretation of the Drayton Area, was used to construct 24 E-W oriented cross sections, at regular intervals through the deposit. Ratio and reserve estimation was based on the determination of the quantity of the situ-coal and overburden between each pair of cross sections. In order to determine strip quantities for phasing of the operation, the mining strips and seam benching patterns adopted in the mining plan were superimposed on the cross sections.

The overburden and coal quantities determined made allowance for pit end wall batters and benches as shown in the mining plan. Volumes of cindered or burnt coal were included in overburden quantities. Although not included in the reserve base the volume of potentially recoverable cindered coal was assessed at approximately 1.3 million m^3. In estimating the quantities of mineable coal, the depth of coal weathering was considered. Examination of the overburden weathering distributions resulted in the exclusion of all coal from the mining reserves which was at a depth of less than 10 metres in the West Pit, 15 metres in the East Pit, and less than 20 metres in the North West Pit.

To allow for coal losses during mining operations, the in-situ coal reserves were depleted by a value determined according to the major excavating machine uncovering the coal seam. The following constraints were applied:-

- For seams stripped by the shovel/truck overburden operation (R2 and G4) 20 cms of the seam thickness will be lost.
- For seams stripped by a parting removal operation (R3 and G5) 15 cms. of the seam thickness will be lost.
- For seams or splits of seams uncovered by the dragline 25 cms. of the seam thickness will be lost.
- For all splits within the Balmoral seam 15 cms. of each split will be lost. This was not the case for parting removal by the dragline in the south cuts of the West Pit, where a loss of 25 cm. coal was adopted.

The resulting overburden and coal quantities are shown in Tables 2, 3A and 3B and these quantities were the basis of the production schedule in Table 4.

3.5 DRAGLINE MODE OF OPERATION

Because of the complexity of the deposit and the large variation in parting and interburden thickness the dragline mode of operation varies between pits. It is anticipated that the hardest overburden for drilling, blasting and excavating will be the Balmoral Sandstone which lies between the P2 and B1 seams. In both pits this material is handled by a conventional inline dig and side cast operation in preference to the more difficult chop-down method.

3.5.1 West Pit (Refer Figure 9)

The cuts are conveniently divided by a central access ramp, initially exitting through the highwall for the box cut and the first two dragline cuts, and then being routed through the low wall spoils for the remaining cuts. The mining method involves the dragline digging alternate north and south cuts starting from the central ramp. A cut width of 50 m. is best suited to the multiseam application whereas a wider cut gave a lower rehandle figure in the single seam situation.

In the south cuts the parting between the B1 and B31 coal varies between 4-5m. and a method of working was designed to enable the dragline to dig this parting from a low wall bench. The mode of operation for the south cuts is shown in Figures 21A and 21B and involves one traverse of the cut with the dragline sitting on the highwall side and two on the spoil bench.

In the north cuts the parting between the B1 and B31 coal varies between 3-4m. and is removed by an in-pit shovel/ truck operation. The method of dragline operation in the north cuts is basically a conventional bridging method. A ramp is formed at the north end of the cuts rising from the P2 to the G5 level. This gives an exit road for the dragline after completing stripping to the B1 seam and enables it to walk back along the G5 bench to start stripping operations in the next south cut. It also provides access for coal removal from the T2 and P2 seams when the progressive dragline operation cuts off the road along the highwall benches. After completing cut 10W the dragline is temporarily transferred to the East Pit until all coal and partings are removed. The same procedure applies for cuts 11W and 12W and then upon completion of the final cut, the machine is moved to the East Pit.

3.5.2 East Pit (Refer Figure 10)

The initial dragline work in the pit requires cross-dragging from the anticlinal box cut down the west side of the anticlinal axis.

To expose the steeply dipping Balmoral seam down the east flank of the anticline a conventional bridging method will be adopted. A 70m wide cut is used, this having the advantage of slowing down the rate of advance of the dragline and creating more working space in the cut, without increasing the percentage of overburden rehandled. The North West Pit low wall benches and ramps provide a convenient access route down to the coal in the initial dragline strips in the East Pit. With increasing depth of operations and backfilling of the final North West Pit void by the East Pit shovel/truck pre-strip material a central ramp through the spoil (designed on a 10% grade) provides continued access to the Balmoral coal. As in the West Pit the dragline commences stripping at the central ramp and completes its operation at the north and south pit limits. For the first five dragline cuts all overburden above the P2 level is removed by shovel and truck. From cut 6E to 18E a shovel/truck operation removes all overburden down to the G5 level and the dragline removes the remaining overburden to the Balmoral seam. (Refer Figures 20A to 20C)

- Commencing from the central access ramp and progressing north, the dragline digs from the G5 level to expose the T2 coal.

- After completing this operation the dragline walks back along the G5 bench and down the central highwall ramp to sit at the P2 bench level.
- At this horizon the dragline "chops" all the T2-P2 partings, digs a key cut in the P2 to B1 overburden, and sitting out on the spoil bridge digs the remaining overburden exposing the B1 coal.
- When reaching the end of the cut the machine then walks back along the P2 bench to the central ramp and the same procedure is repeated in the south cuts.

This mining method was adopted after detailed consideration of various alternative dragline operations. The advantages of this method of operation are that:-

- Scheduling of overburden preparation, and coal and parting removal is simplified,
- Overburden preparation can always be carried out well in advance of the dragline,
- Linear advance along the cut is slow, avoiding crowding of the coal extraction operations and allowing the opportunity to blend coal from the various seams.

3.5.3 Geotechnical Considerations

Highwall batters and low wall spoil angles were initially determined after observation of these parameters in the neighbouring Bayswater Mine where the same geological sequence occurs. A preliminary geotechnical investigation completed by the consultants, Dames and Moore, substantiated the assumptions made. Two geotechnical boreholes have been drilled and core samples tested and further geotechnical studies are being carried out in association with the completion of the bulk sample excavation.

For the purpose of assuring spoil pile stability in the multiseam operation in the East Pit, a 25 metre bench will be formed in the spoil at the dragline bridge level. This bench may be used as a haul road by having a ramp running (parallel to the cut) from the bench down to the Balmoral seam. The ramp would enable Balmoral coal to be hauled up and along the spoil bench, across the end wall bench and out through the highwall upper seam ramp to the Pit Head.

Tight scheduling in the West Pit restricts the dragline to working on a five day week basis. This situation applies until the multiseam operation commences in the East Pit when the dragline can then be scheduled for a seven day week basis. Dragline specifications and production estimates are shown in Appendices 1 and 2.

3.6 MINING OPERATIONS

3.6.1 Site Preparation

Before mining operations commence all necessary site preparation will be carried out. This will involve the stripping and stacking of all topsoil from areas disturbed by tips and associated facilities and initial mining areas. Topsoil dumps will be positioned so as to provide visual screens and noise barriers where these are considered necessary. Settling ponds will be formed where the natural topography drains surface water from the site, ensuring that any water leaving the site will be within acceptable quality limits.

Before commencing operation in the West Pit the stock route will be diverted along the Western Lease boundary to join back into the existing route east of the South Tip.

3.6.2 Overburden, Coal and Parting Preparation

Overburden drilling and blasting for the shovels and dragline is scheduled to remain well in advance of the excavating equipment. The mining method adopted has been designed to minimise "peak" drilling demands. Drill rigs have been selected on the basis of suitability of application. (Refer Appendix 4)

It is anticipated that explosives will be purchased direct from the ICI Explosives depot at Liddell which has bulk ANFO and Slurry trucks available for "down-the-hole" deliveries. Estimates of blasting patterns are shown in Appendix 4.

3.6.3 Shovel/Truck Overburden Removal

The initial shovel/dump truck operation will start in the North West Pit with a 7.6m^3 hydraulic excavator loading overburden into 85 ton rear dump trucks. Overburden from the box cut and following cuts will be trucked to the East Tip until progressive backfill of the working pit commences. Because of the increasing length of cuts and depth of overburden, the backfill void created will only take a proportion of the annual quantity of overburden removed and the remaining material will go to the tip.

Commencing year 2, the hydraulic excavator will be transferred to start the box cut in the West Pit and a 16m^3 (21yd^3) electric shovel loading into 120 ton rear dump trucks will be introduced in the North West Pit. Shovel working benches are designed for 30 metres in width which gives a 60 metre working area as each cut is excavated. End wall benches and backfill benches are designed to give progressive backfilling of the mined out areas. Figure 6 shows the working pit with highwall and backfill benches at the end of year 5.

The overburden from the box cut in the West Pit will be trucked to the South Tip along an access ramp out through the highwall. Further pre-stripping in this Pit will not be necessary until the dragline has completed two full years of operation and during this period the hydraulic shovel and associated trucks will be used on coal and parting removal. When pre-stripping becomes necessary, a hydraulic excavator will be introduced together with the necessary 85 ton dump trucks and the pre-stripped overburden will be trucked around the end of the dragline cuts to restore the spoil area behind the dragline.

Upon completion of the North West Pit the electric shovel will be transferred to the East Pit to dig a box cut down the anticlinal axis and then to pre-strip overburden down the eastern flank of the anticlinal axis in preparation for the dragline. The electric shovel will be supplemented by the $7.6m^3$ hydraulic shovel from the West Pit to keep the pre-stripping in advance of the dragline.

Shovel/truck overburden from the south cuts in the East Pit will be trucked over to backfill the worked out West Pit and to restore the area to final surface contours. Overburden from the north cuts will be trucked back into the North West Pit and the dragline spoils area to progressively restore from the Western Lease boundary.

Production estimates for the shovels and loaders are shown in Appendix 3 and the mine development plan is shown in Figure 7.

3.6.4 Coal and Parting Extraction

Due to the dip of the coal seams, hydraulic excavators of $4.5m^3$ and $7.6m^3$ (equivalent rock bucket) capacity were selected for excavating the coal and partings. These excavators will be used in both face shovel and backhoe configuration. They will be supplemented by a $9m^3$ front-end loader to assist in parting removal between the R2 and R3 seams and the G4 and G5 seams.

Rear dump trucks of 50 ton and 85 ton capacity will be used to truck the coal and partings with the 50 ton trucks equipped with coal bodies and the 85 ton trucks a combination of coal and rock bodies. Rear dump trucks were chosen in preference to bottom dump coal haulers for the following reasons:

- Gradients on some of the access ramps are considered too severe for bottom dump haulers.

- A flexible fleet will be necessary, as many of the trucks will be required to carry both coal and partings.
- In areas with steeply dipping pavements in-pit manouvreability will be improved.

Equipment selected for cleaning the top of the coal seam, prior to drilling, and around the loading excavators are rubber tyred dozers (e.g. Cat 824) except in the East Pit where a hydraulic shovel with backhoe attachment is used for coal loading due to the steep seam dips. Table 6 shows a list of all equipment employed at the mine when operating on full production.

3.6.5 Manning Levels

Manning levels were determined on a yearly basis. Table 8 shows the total labour build-up and annual variations while Table 9 shows a detailed listing for year 13 of the operation. A similar listing of mine site staff is shown in Table 10.

All wage rates have been determined in accordance with the awards under the Coal Industry Act 1946 - 1973 as follows:-

- The Coal Mining Industry (Miners) Award, 1973, New South Wales.
- The Coal Mining Industry (Engine Drivers and Firemen's) Award, 1973, New South Wales.
- The Coal Mining Industry (Mechanics) Award, 1973, New South Wales.
- The Coal Mining Industry (Deputies and Shotfirers) Award, 1973, New South Wales.
- The Collieries' Staff Award, 1973.
- The Amalgamated Engineering Union.

4. COAL QUALITY

Analysis of coal samples obtained during the various phases of exploration has been conducted by several laboratories (refer Table 11), and the general quality of the Drayton coal has been known for some time. However the selective mining criteria developed in the feasibility study required a detailed re-evaluation of quantity and quality of the coal in the selected mining areas of Drayton. For the purposes of the re-evaluation the three mining pit areas, the North West Pit, the West Pit and the East Pit were computer modelled and studied in detail.

NORTH WEST PIT

A satisfactory computer model involving three Balmoral seam plies (B1, B3 and B4) and two interburdens (B1-B3 and B3-B4) was produced. The effect of varying the cut off thickness of the interburden waste on the overall coal quality in the pit was examined and the following selective mining criteria was adopted.

50 cm	interburden cut-off thickness
5 cm	dilution by waste per mining ply
20 cm	top of coal loss
15 cm	coal loss at each parting

Application of this mining criteria to the North West Pit computer model generated the following coal quality predictions (as received basis).

	IN SITU	AS MINED	STANDARD DEVIATION
Ash (%)	9.7	10.17	0.73
Total Sulphur (%)	0.83	0.83	0.03
Specific Energy (MJ/Kg)	28.17	27.96	0.55

WEST PIT

As for the North West Pit a computer model was developed and the parting within the B1 ply of the Balmoral seam was identified as being of critical thickness. The effect of varying the B11 - B12 parting cut off thickness on overall pit quality was examined in association with the iso-coal contour plans, and the following selective mining criteria was adopted:

50 cm	B11 - B12 interburden cut-off thickness
5 cm	dilution by waste per mining ply
20 cm	coal loss for seams stripped by shovel/truck overburden operation (Brougham, Grasstrees)
25 cm	coal loss for seams stripped by the dragline (Thiess, Puxtrees, Balmoral)
15 cm	coal loss for seam splits within the Balmoral seam (B31 and B32) and at the B11 - B12 parting.

Application of this criteria to the West Pit computer model generated the following coal quality prediction (as received basis).

SEAM	IN SITU			AS MINED		
	Ash %	Specific Energy (MJ/Kg)	Sulphur %	Ash %	Specific Energy (MJ/Kg)	Sulphur %
R3	9.07	28.76	0.96	11.17	27.86	0.94
STD. DEV.				1.52	0.39	0.15
G5	7.35	29.83	0.75	11.21	28.15	0.73
STD. DEV.				0.62	0.62	0.04
T2	6.69	29.63	0.87	8.99	28.65	0.85
STD. DEV.				0.76	0.34	0.06
P2	6.44	29.87	0.83	8.49	29.01	0.81
STD. DEV.				0.69	0.35	0.05
B1	11.70	27.77	1.12	13.90	26.84	1.08
STD. DEV.				3.97	1.40	0.37
B31	10.50	28.06	1.04	11.84	27.49	1.02
STD. DEV.				1.90	0.85	0.31
B32	12.54	27.14	0.80	13.86	26.58	0.79
STD. DEV.				1.76	0.77	0.15
AVERAGE	10.54	28.10	0.95	12.34	27.34	0.93
STD. DEV.				2.14	0.86	0.23

EAST PIT

As for the other pits a computer model was developed. No critical parting was identified in the East Pit although close to the anticlinal axis the B4 ply of the Balmoral seam is mineable as part of the B3 horizon. Only the first nine cuts of the East Pit, which coincides with fifteen years of operation on the current production schedule were modelled because the denisty of data outside this area is too scarce to realistically form the basis of a computer model.

The selective mining criteria used for evaluating the East Pit is consistent with that used in the West Pit and North West Pit i.e.

- 5 cm dilution by waste per mining ply
- 20 cm coal loss for seams stripped by shovel/truck overburden operation (Brougham, Grasstrees)
- 25 cm coal loss for seams stripped by the dragline (Thiess, Puxtrees, Balmoral)
- 15 cm coal loss for B3 seam ply.

Application of this criteria to the East Pit computer model generated the following coal quality predictions (as received basis.)

SEAM	IN SITU			AS MINED		
	Ash %	Specific Energy (MJ/Kg)	Sulphur %	Ash %	Specific Energy (MJ/Kg)	Sulphur %
R2	9.15	28.75	0.80	10.99	27.97	0.78
STD. DEV.				1.93	0.58	0.08
R3	8.21	29.00	0.81	10.13	28.19	0.79
STD. DEV.				1.52	0.62	0.12
G4	11.38	28.36	0.74	14.14	27.14	0.72
STD. DEV.				1.61	0.50	0.04
T2	7.23	29.68	0.90	9.00	28.92	0.89
STD. DEV.				1.90	0.83	0.12
P2	9.31	28.71	0.85	10.87	27.93	0.84
STD. DEV.				2.61	1.01	0.08
B1	8.04	29.20	0.97	9.99	28.37	0.95
STD. DEV.				2.45	0.91	0.11
B3	9.88	28.27	0.91	10.39	28.05	0.91
STD. DEV.				1.95	0.87	0.08
AVERAGE	9.36	28.60	0.89	10.47	28.11	0.88
STD. DEV.				2.00	0.84	0.09

4.1 PRODUCT QUALITY

The detailed analysis of the three major quality parameters in the mining areas of Drayton permitted the average ash, sulphur and calorific value results to be determined for the first fifteen years of operation. The quality predictions are (as received basis.)

IN SITU

PIT	ASH %	SPECIFIC ENERGY (MJ/Kg)	SULPHUR %
NORTH WEST	9.70	28.17	0.83
WEST	10.54	28.10	0.95
EAST	9.36	28.60	0.89
AVERAGE	9.76	28.40	0.90

(12211 Btu/lb = 28.40 MJ/Kg = 6784 Kcal/Kg)

AS MINED

Pit	Ash %	Std. Dev.	Specific Energy (MJ/Kg)	Std. Dev.	Sulphur %	Std. Dev.
North West	10.17	0.73	27.96	0.55	0.83	0.03
West	12.34	2.14	27.34	0.86	0.93	0.23
East	10.47	2.00	28.11	0.84	0.88	0.09
Average	11.02	1.91	27.85	0.81	0.89	0.13

(11975 Btu/lb = 27.85 MJ/Kg = 6653 Kcal/Kg)

An indicative product quality, including additional raw coal quality parameters as determined in the Drayton Geological Report is as follows:

Total Moisture (As Received)	7 – 9%
Inherent Moisture (Air Dried)	3 – 4.5%
Ash Content (As Received)	9 – 12%
Volatile Matter (As Received)	33 – 35%

CALORIFIC VALUE		MJ/KG	KCAL/KG	BTU/LB
Gross	(As Received)	27.7 – 28.5	6610 – 6800	11900 – 12240
	(Air Dried)	28.8 – 29.7	6885 – 7083	12400 – 12750
	(Dry)	30.1 – 30.9	7185 – 7390	12930 – 13300
Net	(As Received)	26.4 – 27.2	6305 – 6500	11350 – 11700

SULPHUR CONTENT (AIR DRIED)

Total	0.7 – 1.00%
Pyritic	0.2 – 0.3%
Sulphate	Negl. – 0.003%
Organic	0.5 – 0.7%
Phosphorus (Air Dried)	0.03 – 0.08%
Chlorine (Air Dried)	0.03 – 0.04%
Hardgrove Grindability Index	48 – 52
Product Sizing	38 x 0 mm

ASH FUSION TEMPERATURES	Reducing Atmosphere	Oxidising Atmosphere
Initial Deformation	1250 – 1380^{o} C	1290 – 1420^{o}C
Spherical	1270 – 1400^{o} C	1310 – 1440^{o}C
Hemispherical	1300 – 1420^{o} C	1360 – 1480^{o}C
Flow	1350 – 1480^{o} C	1390 – 1520^{o}C

PROXIMATE ANALYSIS (AIR DRIED)

Inherent Moisture	3 – 4.5%
Ash	10 – 13%
Volatiles	35 – 37%
Fixed Carbon	48 – 50%
Total Sulphur	0.7 – 1.0%

ULTIMATE ANALYSIS (DRY ASH FREE)

Carbon	82 – 84%
Hydrogen	5.3 – 5.6%
Sulphur	0.8 – 1.1%
Nitrogen	1.6 – 1.8%
Oxygen	8.0 – 10.0%
Chlorine	0.03 – 0.04%
Phosphorus	0.04 – 0.1%

COAL CLASSIFICATION

ASTM	High Volatile A Bituminous
International	611 – 612
Australian	611(2) – 612 (1)

COKING COAL

Further work is required to assess, predict and design for the coking coal within the mining areas. For the purposes of this report the details of the ACIRL study have been used and only the Grasstrees and Thiess seams have been identified as coking coal. As production from seams with coking potential is not realised until year 5, sufficient time is available to investigate this important potential of the deposit. An indicative specification for washed coking coal, taken directly from the Drayton Geological Report is as follows:

Indicative Specification for Washed Coking Coal
(air-dried basis)

Inherent Moisture	(%)	3.5
Volatile Matter	(%)	38.0 – 40.0
Ash	(%)	5.0 – 5.5
Fixed Carbon	(%)	52.0 – 54.0
Sulphur	(%)	Max. 0.85
Phosphorus	(%)	Max. 0.07
B.S. Crucible Swell No.		4 – 5½
Gray King Coke Type		G – G2
Sizing (mm)		38 x 0

4.2 QUALITY CONTROL

The coal quality is influenced by three factors, these being:

- Inherent properties of the seam or seam split being mined, which would include all minor dirt bands within the seam.
- Extraneous material which may come from the roof or floor of the seam during mining operations.
- Partings mined with seam splits

In order to plan for variations in inherent properties of the seam being mined, coal quality sampling will be a continuing operation throughout the mine life. Cored boreholes will be drilled immediately behind the stripping shovel or dragline enabling detailed coal sampling to be carried out in advance of the working coal faces. These holes will be drilled at intervals suited to the detail of data required and the variation in coal quality over any given area. Results obtained will enable areas of high and low quality to be determined and will give the opportunity to plan blending procedures in the pit to give a consistent ROM product. Channel samples taken along the exposed highwall face of each working cut will give a good indication of the coal quality to be expected in the next cut.

Dilution of the coal mined by extraneous matter from the roof or floor of the seam can be controlled by good in-pit supervision which ensures that the top of the seam being mined is clean and free from overburden or parting material and that the coal loading equipment is working on the correct floor horizon of the seam. Most of the strata forming the base of the seam consists of dark grey to black shales which will make selection of the correct floor horizon difficult and necessitate close and effective supervision.

The only seam where partings are mined with seam splits is the Balmoral. The vertical selection criteria allows for partings to be mined with seam splits provided the ash does not exceed 15%. In practice, the quantity of partings mined with seam splits will be dependent on the required product ash specification and the coal washing facilities available.

Blending of the steaming coal product will be possible at the pit head where coal stacking facilities allow for the stockpiling of 80,000 tonnes of steam coal products on two stockpiles each of 40,000 tonnes capacity. A further 40,000 tonne stockpile, primarily designed for clean coking coal storage could also be available for stockpiling steam coal depending on the coking coal production rate and railing requirements. This will allow stockpiling of "high" and "low" ash coal separately and provides an opportunity for further blending. Figure 14 shows a flowsheet of the elements of the quality control system planned for Drayton.

5. COAL HANDLING, TRANSPORTATION AND DESPATCH FACILITIES

The design of the coal handling facilities involved study of:-

- Alternative locations.
- The type of coal products that were to be produced, e.g. unwashed steaming coal, washed coking coal, washed steaming coal.
- The integration of the pit head with the mining plan, both in terms of the scale of the plant required and the timing of its construction.
- The impact on the environment of such facilities.

5.1 LOCATION

The most suitable location for the coal handling facilities has been identified as an area to the North East of the East Pit, (Refer Figure 15), where the upper coal seams have been intruded and cindered, and the Balmoral series has thinned and deteriorated into numerous carbonaceaous shale bands. The area also has the advantage of being:-

- Close to the main New England Highway, with an established access already existing.
- Well situated for the siting of a conveyor link to the train load-out facilities at the Antiene site.
- Well sited with respect to the North West and East Pit areas, where 42 million of the 53 million ROM tonnes of coal will be mined.
- Well hidden from public view, by both the local topography and vegetation.
- Fairly level and hence requiring only minimal earthworks prior to construction of the facilities.

5.2 COAL PRODUCTS

The design of the coal handling facilities is such as to provide sufficient flexibility to handle three main product types:-

- Unwashed, sized, steaming coal with a "high" ash composition (10%-15% ash).
- Unwashed, sized, steaming coal with a "low ash" composition (less than 10% ash).
- Washed, sized, soft/hard coking coal.

As well as installing a heavy media cyclone plant for benefication of the metallurgical grade coal, it was decided to introduce a simple jig plant to handle low tonnages of dirty steam from the following sources:-

- Coal recovered from the low wall rib pillar, representing some 3% of the total ROM coal.
- Diluted coal from the roof and floor of the coal seams.

- Thin upper seams not considered in the estimated reserves, but mined where localised thickening occurs.
- Cindered coal as it is anticipated that, though not included in measured reserves, some proportion of the estimated 1.7 million tonnes of this coal would be recoverable.

The jig plant recovered coal would be blended in with the "low ash" raw steaming coal.

5.3 SCALE AND TIMING OF COAL HANDLING FACILITIES

Construction of the coal handling facilities involves:-

- A staged duplication of the basic steam coal handling plant, this being a ROM dump hopper and primary and secondary crushing and screening facilities. The timing of the stages would be 1980 and 1983, the latter coinciding with the start-up of the dragline and the subsequent large increase in coal output from the mine.
- Two unwashed steaming coal stockpiles and associated reclaim facilities. The capacity of these stockpiles was designed to hold some two weeks production when the mine is producing at peak capacity, these on-site quantities being designed to overcome shortfalls in production due to industrial disputes and excessively wet weather. Environmentally it was considered preferable to have the major stockpiling adjacent to the mine site rather than at the Antiene loadout station. Adequate stockpiling at the port would still be necessary.
- An overland conveyor (Refer Figure 16) linking the coal handling plant at the mine to the Antiene rail loadout station. For the planned level of output at Drayton the overland conveyor is clearly superior in both practical and economic terms to road transportation of the coal to Antiene.
- A second stage of development, due for commissioning in 1984, providing further stockpiling facilities to coincide with the mining of the coking coal and the associated dense media cyclone wash plant. The jig plant for handling dirty steam coal would follow in 1985, although this facility could be introduced earlier if sufficient quantities of dirty coal are produced. Separate stockpiles would be provided for unwashed coking coal and dirty steam coal with the capacity of each of the two unwashed coal stockpiles sufficient to allow a constant feed rate to the two beneficiation plants, even when large run-of-mine fluctuations of a particular coal type occur.

The second line of crushing and screening facilities, installed in 1983, would incorporate the necessary transfer and reclaim conveyors to handle the three types of coal coming from the pit, i.e. coking coal, clean steaming coal and "dirty" steaming coal, and deliver the coal type to its respective stockpile or benefication plant.

- A train loading facility (Refer Figure 17) with a capacity sufficient to handle future increases in the unit train size. A flood loading system was preferred for rapid train loading. Bin storage capacity at the siding is sufficient to load two unit trains, so should any minor hold up of the overland conveyor occur train loading operations would not be affected.

6. PIT HEAD ESTABLISHMENT AND PROVISION OF SERVICES

The pit head establishment, (Refer Figure 15 consisting of administration office, bath house, workshop, store and associated facilities will be located on the rise in the terrain to the west of the plant and stockpile area. The layout adopted provides ideally for area drainage, efficient traffic flow, and ease of overall supervision.

Where possible natural vegetation will be left untouched in the vicinity of the pit head and the area around the administration office and bath house will be landscaped.

6.1 ELECTRIC POWER SUPPLY AND RETICULATION

Power for the mine site will be purchased from the Upper Hunter County Council, with delivery from the UHCC terminating in a switch yard adjacent to the workshop. From this switch yard, 33 kv power will be run to the mining areas, for supply to transportable 33/6.6kv substations providing power to the dragline and electric power shovel. Within the switch yard, power will be stepped down from 33 kv to 11 kv for distribution to the various plant substations.

6.2 WATER SUPPLY AND TREATMENT

Mine water requirements will be supplied by a 750 ML mine area catchment dam accumulating run-off which would normally flow to the Liddell Ash Dam. The dam is located east of the East Pit and its 150 Ha. catchment area covers most of the East Pit. As mining advances eastwards, the catchment area is reduced and after twelve years of mining operations the dam is unable to meet the full water requirements for Drayton. At this stage, a supplementary supply of water will be required, either from groundwater sources or from the Hunter River. Water from the dam will be pumped to a head tank for subsequent distribution around the pit head site and a small treatment plant will be needed to ensure the tank water remains potable. For sewerage treatment a Passveer system will be provided for the mine, as this system is currently advocated by the New South Wales Public Works Department. Treated water will be used for spraying the revegetated area of the overburden dumps.

The waste water from the preparation plant thickener will be returned via settling ponds to the process water dam south of the pit head area and the clear water will be recycled to the wash plant.

7. ENVIRONMENTAL IMPACT

A formal Environmental Impact Statement is currently being prepared by the Consulting Company, Dames and Moore. Factors considered during the design of the mine and associated facilities were:-

- *Visual Impact*

  The site is well screened from the main New England Highway by the undulating topography and natural vegetation. The mining plan has been developed so that progressive restoration can be carried out, starting from the diverted stock route on the western boundary of the Lease Area, and steadily progressing eastwards as mining advances.

  The Pit Head facility is sited in a wooded area and much of this natural vegetation will be retained to act as a visual screen.

  The train loading facility and rail loop is sited adjacent to a public road which provides access to a local recreation area on the shores of Lake Liddell. It was considered that enclosed storage bunkers sited away from the road would be more acceptable than open coal stockpiles.

  Barriers of vegetation will be developed using tree and shrub species selected to provide a maximum screen effect.

- *Water Pollution*

  Water pumped directly from the mine, and surface run-off from the South tip and the rehabilitated area, will be stored in a holding and settling area in the SW of the mine area as shown in Figure 8. The major water flow will be towards this area and drains will be provided along the diverted stock road and on the southern mine boundary to bring the surface run-off to the holding zone. Water from this storage area will be used for road watering and spraying of revegetated areas or allowed, after settling out of solids, to flow into existing watercourses. Surface run-off to the east will flow into the mine area catchment dam.

  The preparation plant incorporates a water recycling system which includes a thickener and a process water dam. Design of detailed water management plan is in progress as part of the Environmental Impact Study preparation.

- *Dust Control*

  With large scale mining excavations, creation of fugitive dust by loading machinery and trucks is unavoidable. Water trucks will be utilised to reduce dust along the haul roads and in the vicinity of working plant. Open coal stockpiles will be provided with a water spray system for surface wetting down. To control dust emission during coal handling, dust suppression systems will be provided at conveyor transfer points, at crushing and screening operations and at the train load-out facilities. The use of an enclosed bin storage system at the train loadout facility will ensure that dust pollution in this critical environmental position will be well within acceptable limits.

- *Noise and Ground Vibrations*

  In mining operations noise is generated by overburden and coal blasting, by operating plant in and adjacent to the pit, by various unit operations in coal preparation plants including breakers, crushers, screens, motors, blowers, feeders and chutes, and by workshop operations.

  The Drayton Mine is located in such a sparsely settled area, sufficiently remote from any community settlement that noise nuisance to any places of residence will be avoided. However, all conventional noise abatement measures will be applied to keep noise levels to a minimum for the protection of mine employees. Such measures will consist of earth barriers, vegetation barriers and effective silencing systems on operating plant and trucks.

- *Rehabilitation*

  The mining plan makes provision for the continuous rehabilitation of permanent overburden tips and disturbed areas both during and after the completion of mining operations.

  Topsoil will be stripped and stockpiled in advance of the mining activities and will later be recovered for use during the restoration of tips and recontoured areas. All topsoil tips will be seeded and an early growth encouraged in order to minimise soil erosion. Overburden tips outside of the mining areas have been designed for permanent placement and all restored ground levels and contours will be formed to blend in with existing topographical features in the undisturbed areas. Progressive rehabilitation of the excavated areas will commence at the west (and most public) boundary and progress steadily eastwards during the life of the mine.

  The restoration plan (Figure 11) shows the proposed final surface contours after mining operations are complete. The plan allows for the provision of water courses to be reinstated in their original position where this is possible. Overburden from the shovel pre-strip work in the East Pit will be used to fill the final void in the West Pit.

The final void in the East Pit will be contoured to form a depression with slopes conforming to the standards recommended by the Soil Conservation Service of New South Wales, and the bottom allowed to partly flood to create a small lake.

8. COAL TRANSPORT

8.1 RAIL TRANSPORT

Train loading at the Antiene loadout station is planned for a maximum rate of 3000 t/hr. Trains operated by the NSW Transport Commission will transport the coal 114 km to the ship loading facilities at Port Waratah, Newcastle. The rail link includes a standard guard (4 foot 8½ inches) double track from Antiene to Maitland, and four tracks from Maitland to Newcastle. The Public Transport Commission has confirmed that ample coal track capacity exists, (stated by them to be 26 mtpa) and is more than sufficient to handle the additional output expected from the Drayton Coal Mine, i.e. up to 3.5 mtpa. An earlier study by the Northern Branch of the N.S.W. Combined Colliery Proprietors Association is of the opinion that the main line track in the Hunter Valley is capable of handling tonnages of at least 20 mtpa and believes that this opinion is shared by the overseas consultants to the P.T.C.

The current unit train consists of 20 CHS wagons with a nominal capacity of 76 tonnes and a tare of 24 tonnes; giving a total train capacity of 1520 tonnes. The Commission wish to operate their rolling stock 7 days per week, 24 hours per day, and so adequate manning and coal storage levels at the loadout station would have to be maintained. Although the timetable could vary during certain periods of the day, and taking into consideration the anticipated time to unload wagons at the Newcastle Coal Loader, it is estimated that the turnaround, Antiene to Antiene via Port Waratah, will be approximately eight to nine hours.

Future plans are, first, for unit trains of 24 CHS wagons x 2 locos by mid 1978, then a further increase to 42 CHS wagons x 3 locos having a total train capacity of 3192 tonnes. This unit train size is not expected to be in operation for at least 2 years being dependent on delivery of additional locomotives. The P.T.C. states that the 300 CTS wagons currently on order for delivery prior to July 1979, will free sufficient CHS wagons from other duties to provide rolling stock for 20 mtpa on the Northern line.

8.2 PORT FACILITIES

The Port of Newcastle, situated at the mouth of the Hunter River, exports coal from the Newcastle, Hunter Valley and Northern N.S.W. collieries. Two major ship loading facilities are available at the port:-

- Port Waratah Coal Services
- The Marine Services Board Carrington Basin facility.

Since the construction of the Port Waratah loader, both facilities have become part of an integrated coal handling complex. Figure 18 shows the location of the two sites at Newcastle.

The Carrington basin loader has two coal loading heads each of 1,000 tph nominal capacity, supported by a very limited stockpile within the coal loader itself but also by addititional facilities in the Golins yard and more recently by interconnection to the new stockyard which primarily supports the new Port Waratah channel coal loader. The new loader recently constructed is now functional and has two loader heads each of 2,000 tph nominal capacity, supported by stockyards of 1,000,000 tonnes nominal capacity with two 2,000 tph reclaimers. The significant number of distinct coal types handled reduces the stockyard capacity by some 250,000 tonnes, whilst the limited yard belt capacity of 2 x 2,000 tph means that the new loader and old loader cannot be operated concurrently at full capacity. The present nominal capacity of the coal loading facilities has been assessed by others to be:-

Basin Loader	7 mtpa
Port Waratah	14 mtpa

To achieve this, some modifications will be necessary to the new facilities. In particular a new reclaimer and additional yard belts will be required. The cost of these modifications would amount to some $7 million. At the present time, however, an effective capacity of some 15 million tonnes per annum total is available.

The M.S.B. have currently embarked on a dredging programme which will provide additional water depths in the channel leading to, and beyond, the new coal loader. Its present programme is based to give 13.1 m of water to accommodate 75/80,000 d.w.t. ships fully laden by the end of 1979 and to give 15.2 m of water to accommodate 120,000 d.w.t. ships by the end of 1981. This work commenced in mid October, 1977, and has a tender programme of 3 years and 8 months, i.e. a nominal completion date of June, 1981. Work to date is ahead of Schedule but has not entered the hard rock phase.

Figure 19 illustrates schematically the layout of Port Waratah and the following photograph shows the coal loading complex at Port Waratah.

AERIAL VIEW OF COAL LOADING COMPLEX, PORT WARATAH, NEWCASTLE

TABLE 1

DRAYTON COAL MINE

SUMMARY OF TOTAL IN SITU

COAL RESERVES

(tonnes x 10^6)

STATUS / SEAM	MEASURED		INDICATED	INFERRED
	EAST BLOCK	WEST BLOCK		
Brougham Upper	3.43	0.33	6.40	10.0
Brougham Lower	4.17	0.37	6.76	15.0
Grasstrees Middle	3.94	0.42	4.34	3.0
Grasstrees G5	0.81	0.46	0.43	1.0
Thiess	9.67	2.16	7.25	11.0
Puxtrees	7.00	1.73	3.88	11.0
Balmoral B1/B3	4.25	–	–	–
Balmoral B1	6.14	1.91	1.00	10.0
Balmoral B1–1	0.91	1.79	0.46	–
Balmoral B1–2	–	1.38	0.55	–
Balmoral B2	0.69	0.05	–	–
Balmoral B3	27.37	3.63	2.45	4.0
Balmoral B3–1	0.93	5.55	7.12	14.0
Balmoral B3–2	0.71	4.72	2.25	–
Balmoral B4	2.23	–	–	–
SUB TOTAL	72.25	24.50	42.89	79.0
OVERBURDEN (m^3)	346.17	76.09		
RATIO OVERBURDEN: COAL (m^3:tonne)	4.79	3.10		
TOTAL COAL	96.75			
TOTAL OVERBURDEN (m^3)	422.26			
RATIO OVERBURDEN: TOTAL COAL (m^3 : tonne)	4.36			
TOTAL ALL COAL	218.64			

TABLE 2

DRAYTON MINE : SUMMARY OF MINE QUANTITIES

	NW PIT	WEST PIT	EAST PIT	TOTAL
Total In Situ Coal ('000's tons)	**4222**	**12398**	**41238**	**57858**
Raw Coal Mined ('000's tons)				
Brougham	–	229	4367	4596
Grasstrees	–	404	2539	2943
Thiess	–	822	5230	6052
Puxtrees	–	1142	3573	4715
Balmoral	4073	8614	21950	34637
TOTAL	**4073**	**11211**	**37659**	**52943**
Total In Situ Overburden ('000's m^3)	**17290**	**38979**	**168980**	**225249**
Handled Overburden ('000's m^3)				
Shovel	17405	11851	90277	119533
Dragline	–	28043	81467	109510
Rehandle	–	8827	35420	44247
TOTAL	**17405**	**48721**	**207164**	**273290**
Ratios (m^3/ton)				
$\frac{\text{In Situ Overburden}}{\text{In Situ Coal}}$	4.10	3.14	4.10	3.89
$\frac{\text{Total Overburden Handled*}}{\text{Raw Coal Mined}}$	4.27	3.56	4.56	4.33

* Not including rehandle.

TABLE 3A

DRAYTON MINE : OPEN-CUT MINING RESERVES

WEST PIT

Strip	IN SITU OVERBURDEN ('000 m^3)			In Situ Coal	Ratio O/B: In Situ Coal	RAW COAL MINED ('000 tonnes)						O/B Moved	Ratio O/B: Raw Coal Mined
	Dragline	Shovel	Total	('000 tonnes)	m^3/tonne	Brougham	Grasstrees	Thiess	Puxtrees	Balmoral	Total	('000 m^3)	m^3/tonne
1	–	3667	3667	899	4.08	–	–	–	–	839	839	3712	4.42
2	2149	588	2737	871	3.14	–	–	–	–	784	784	2799	3.57
3	2020	539	2559	834	3.07	–	–	–	–	752	752	2630	3.50
4	2564	527	3091	920	3.36	–	–	–	47	791	838	3154	3.76
5	2532	446	2978	1014	2.94	–	–	92	99	727	918	3063	3.34
6	2095	617	2712	1036	2.62	5	23	55	95	760	938	2786	2.97
7	2444	885	3329	1157	2.88	82	42	85	124	709	1042	3417	3.28
8	2654	1499	4153	1334	3.11	109	93	117	156	720	1195	4260	3.56
9	2519	1280	3799	1104	3.44	4	57	103	137	696	997	3881	3.89
10	2181	1042	3223	948	3.40	–	48	101	134	579	862	3289	3.82
11	1666	988	2654	970	2.74	21	81	140	121	507	870	2722	3.13
12	1639	473	2112	731	2.89	8	33	90	117	411	659	2168	3.29
13	1640	325	1965	580	3.39	–	27	39	112	339	517	2013	3.89
TOTAL	26103	12876	38979	12398	3.14	229	404	822	1142	8614	11211	39894	3.56
NORTH WEST PIT													
TOTAL	–	17290	17290	4222	4.10	–	–	–	–	4073	4073	17405	4.27

TABLE 3 B

DRAYTON MINE : OPEN-CUT MINING RESERVES

EAST PIT

Strip	IN SITU OVERBURDEN ('000 m³)			In Situ Coal	Ratio O/B: In Situ Coal	RAW COAL MINED ('000 tonnes)						O/B Moved	Ratio O/B: Raw Coal Mined
	Dragline	Shovel	Total	('000 tonnes)	m^3/tonne	Brougham	Grasstrees	Thiess	Puxtrees	Balmoral	Total	('000 m^3)	m^3/tonne
Box Cut	4205	3352	7557	1687	4.48	–	–	–	–	1639	1639	7594	4.63
1	1009	1413	2422	557	4.35	–	8	–	–	517	525	2460	4.69
2	3581	4861	8442	2085	4.05	–	8	23	146	1777	1954	8543	4.37
3	3680	6874	10554	2600	4.06	40	103	214	257	1798	2412	10699	4.44
4	3744	7703	11447	3141	3.64	246	175	355	346	1774	2896	11637	4.02
5	3714	5499	9213	2907	3.17	244	156	273	213	1761	2647	9414	3.56
6	4881	3776	8657	2289	3.78	263	186	309	226	1110	2094	8808	4.21
7	4783	3954	8737	2268	3.85	260	157	330	232	1094	2073	8887	4.29
8	4802	4007	8809	2288	3.85	258	192	345	225	1078	2098	8964	4.27
9	4770	4006	8776	2278	3.85	247	186	359	231	1053	2076	8932	4.30
10	4819	4027	8846	2290	3.86	303	167	356	220	1041	2087	9003	4.31
11	4754	4190	8944	2251	3.97	286	170	351	221	1005	2033	9111	4.48
12	4684	4463	9147	2177	4.20	305	155	337	212	963	1972	9305	4.72
13	4591	4997	9588	2161	4.44	334	148	334	201	917	1934	9762	5.05
14	4542	5103	9645	2142	4.50	348	143	331	190	918	1930	9810	5.08
15	4490	5225	9715	2105	4.62	338	155	331	169	904	1897	9873	5.20
16	4428	5230	9658	2065	4.68	329	150	329	162	891	1861	9818	5.28
17	4381	5128	9509	1997	4.76	293	137	329	162	866	1787	9663	5.41
18	4317	4997	9314	1950	4.78	273	143	324	160	844	1744	9461	5.42
TOTAL	80175	88805	168980	41238	4.10	4367	2539	5230	3573	21950	37659	171744	4.56

PRODUCTION SCHEDULE

TABLE 4

YEAR	–1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	TOTAL
Production Weeks	10	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	44	15	–
Overburden Drilling (m)	6944	49862	113746	115838	155438	193063	243994	249096	267290	221591	231714	229482	221814	231004	229214	229214	229214	229214	229214	229214	229214	120978	10413	4266765
Prepared Overburden ('000's m^3)	250	2037	4959	5360	8197	8960	10383	10172	11501	10398	11237	11112	10125	10587	10497	10497	10497	10497	10497	10497	10497	5951	524	195232
Prepared Partings for Dragline ('000's m^3)*	–	–	–	–	481	414	331	203	–	–	–	–	472	940	940	940	940	940	940	940	940	940	–	10361
Total Overburden + Partings Prepared ('000's m^3)	**250**	2037	4959	5360	8678	9374	10714	10375	11501	10398	11237	11112	10597	11527	11437	11437	11437	11437	11437	11437	11437	6891	524	205593
Dragline Production Weeks/Hours	–	–	–	–	46/3900	46/4140	46/4140	46/4140	46/4140	46/4140	46/4140	46/4140	46/5888	46/5888	46/5888	46/5888	46/5888	46/5888	46/5888	46/5888	46/5888	46/5888	9/1152	–
Total D/line O/B + Partings Handled ('000's m^3)	–	–	–	–	6460	7270	7260	7450	7280	6711	7440	7648	8338	9560	9504	9502	9619	9553	9556	9607	9551	9566	1824	153757
Dragline Rehandle (%)	–	–	–	–	24	29	30	34	30	13	22	28	53	49	49	48	48	49	54	52	55	50	41	–
D/line O/B + Partings Removed ('000's m^3)	–	–	–	–	5219	5657	5569	5554	5588	5957	6080	5959	5455	6415	6377	6415	6507	6396	6192	6303	6180	6393	1294	109510
Shovel Overburden Removed ('000's m^3)	–	1573	4973	4933	3553	3553	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	5269	1065	–	98685
Total Overburden Removed ('000's m^3)	–	**1573**	**4973**	**4933**	**8772**	**9210**	**10838**	**10823**	**10857**	**11226**	**11349**	**11228**	**10724**	**11684**	**11646**	**11684**	**11776**	**11665**	**11461**	**11572**	**11449**	**7458**	**1294**	**208195 †**
Overburden/Partings Inventory ('000's m^3)	250	734	782	1270	1286	1565	1576	1263	2042	1354	1384	1408	1415	1404	1341	1240	1048	966	1085	1095	1226	752	–	–
Total Coal Uncovered ('000's tonnes)	–	**490**	**1327**	**873**	**2330**	**2433**	**3223**	**2898**	**2396**	**2385**	**3835**	**3837**	**2918**	**2885**	**2866**	**2795**	**2858**	**2596**	**2465**	**2616**	**2434**	**2043**	**440**	**52943**
Coal Production ('000's tonnes)	–	400	1200	1000	2250	2400	3200	2900	2400	2500	3500	3500	3000	3000	3000	2900	2900	2600	2500	2500	2500	2150	643	52943
In-Pit Coal Inventory ('000's tonnes)	–	90	217	90	170	203	226	224	220	105	440	777	695	580	446	341	299	295	260	376	310	203	–	–
Partings Uncovered ('000's m^3)	–	42	384	337	835	635	1029	888	1171	927	1418	1160	1236	1268	1165	1228	1075	1250	1235	1154	1218	1025	166	20846
Partings Removed ('000's m^3)	–	34	222	442	818	648	994	924	1133	944	1324	1096	1156	1286	1197	1343	1076	1210	1306	1144	1153	1131	265	20846
In-Pit Partings Inventory ('000's m^3)	–	8	170	65	82	69	104	68	106	89	183	247	327	309	277	162	161	201	130	140	205	99	–	–
Overall ROM ratio $\left[\frac{m^3\text{:O/B + partings}}{\text{tonnes:coal}}\right]$	–	3.30	4.04	6.04	4.12	4.05	3.68	4.04	5.02	5.10	3.33	3.23	4.10	4.49	4.47	4.62	4.50	4.97	5.15	4.86	5.20	4.46	3.36	4.33

* Partings consist of B1—B3 partings which the dragline removes in the south cuts, West Pit and the T2 to P2 partings in the East Pit.

† Coal losses are included in this total.

TABLE 5

DRAYTON MINE

SALEABLE COAL TONNEAGE

(000's tonnes)

Year	Steaming Coal Case		Steaming/Coking Coal Case		
	No Jig Plant	With Jig Plant	Coking Coal	Steaming Coal	Total Coal
1	400	400	–	400	400
2	1200	1200	–	1200	1200
3	1000	1000	–	1000	1000
4	2250	2250	–	2250	2250
5	2364	2400	102	2286	2388
6	3104	3200	266	2890	3156
7	2813	2900	323	2523	2846
8	2328	2400	372	1973	2345
9	2425	2500	293	2164	2457
10	3395	3500	352	3094	3446
11	3395	3500	341	3104	3445
12	2910	3000	458	2463	2921
13	"	"	618	2282	2900
14	"	"	640	2255	2895
15	2813	2900	637	2161	2798
16	"	"	603	2208	2811
17	2522	2600	554	1964	2518
18	2425	2500	540	1878	2418
19	"	"	560	1860	2420
20	"	"	539	1881	2420
21	2086	2150	491	1595	2086
22	624	643	105	520	625
TOTAL	**51537**	**52943**	**7794**	**43952**	**51746**

N.B.
- Washed coal recovery assumed as: 80% Grasstrees Seam, 90% Thiess Seam.
- Jig Plant commences middle of Year 5.

TABLE 6

DRAYTON MINE

MAXIMUM NUMBER OF SITE PLANT AND EQUIPMENT

1 x Bucyrus Erie 1370W Dragline
1 x Bucyrus Erie 295B Shovel
3 x O & K RH75 Shovel
2 x O & K RH40 Shovel
3 x Bucyrus Erie 45R Drilling Rigs
2 x Gardener Denver RDC30 Coal and Partings Drills
1 x Mayhew 1000 Exploration Drilling Rig
1 x 475 Michigan Front-End-Loader
1 x Cat 966 Front-End-Loader
35 x Rear Dump Trucks : Capacity Range 50 ton, 85 ton & 120 ton.
1 x Cat 640 Cable Carrier
2 x Cat 640 Water Truck
1 x Cat 623B Scraper
5 x Cat D9 Dozers
2 x Cat D8 Dozers
2 x Cat G14 Graders
1 x Cat 824 Rubber Tyred Dozers
3 x Explosives Truck
1 x 12 Ton - Table-Top Stores Truck
1 x Stores Van
2 x Grease/Lube/Fuel Truck
1 x 35 Ton Mobile Site Crane
1 x 6 Ton Mobile Site Crane
1 x Fork Lift (Workshop Area)
17 x Site Personnel Vehicles
1 x Bucyrus Erie 40H Backacting Shovel
1 x JCB 3D
6 x Mine Drainage Pumps : 3" – 6" Capacity
Miscellaneous Mobile Welding Unit(s)
Lighting Sets

TABLE 7

DRAYTON MINE

CHART SHOWING STRIPPING EQUIPMENT UTILISATION

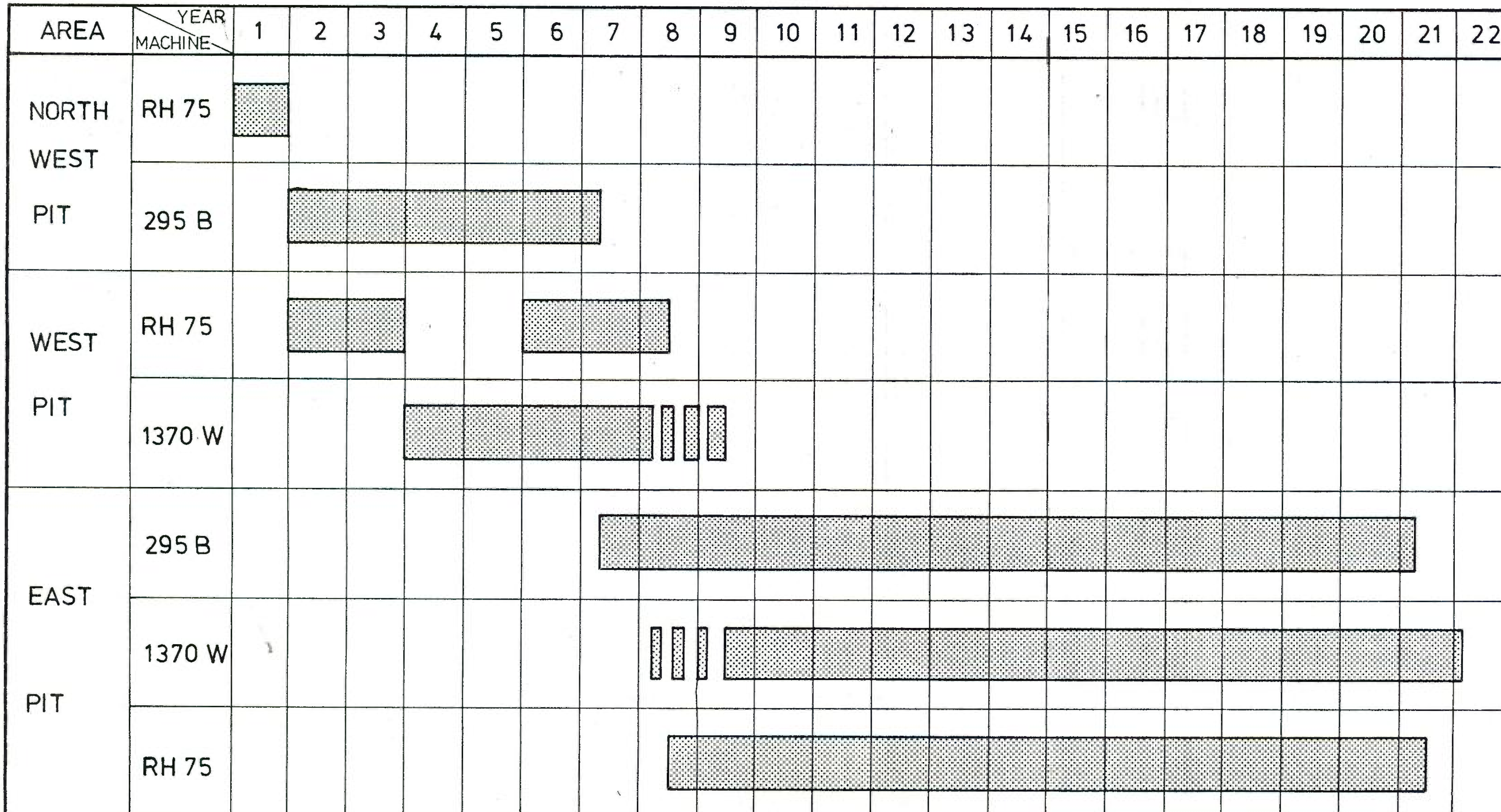


TABLE 8

DRAYTON MINE – MANNING SCHEDULE

Year	Mine Site Labour	Coal Handling/ Despatch	Washing Plant	Workshop Labour	Security Officers Stores Assistants Stores Van/Truck Drivers Bathhouse Attendants	Absentee Labour	Total Hourly Paid Labour	Salaried Staff	Total Mine Personnel
–1	15	–	–	70	10	2	37	23	60
1	76	20	–	59	17	8	180	35	215
2	150	27	–	70	19	16	282	47	329
3	161	"	–	77	20	"	301	"	348
4	157	33	–	100	22	"	328	48	376
5	161	"	8	109	"	"	349	"	397
6	204	"	"	124	"	20	411	"	459
7	198	"	"	"	"	"	405	"	453
8	194	"	"	"	"	"	401	"	449
9	190	"	"	"	"	"	397	"	445
10	208	"	"	"	"	"	415	"	463
11	204	"	"	"	"	"	411	"	459
12	200	"	16	128	"	"	419	"	467
13	198	"	"	"	"	"	417	"	465
14	"	"	"	"	"	"	"	"	"
15	"	"	"	"	"	"	"	"	"
16	"	"	"	"	"	"	"	"	"
17	"	"	"	"	"	"	"	"	"
18	"	"	"	"	"	"	"	"	"
19	"	"	"	"	"	"	"	"	"
20	"	"	"	"	"	"	"	"	"
21	183	"	"	116	"	"	390	"	438
22	98	"	"	70	"	10	249	"	297

TABLE 9

REPRESENTATIVE MANNING TABLE FOR FULL PRODUCTION OPERATION

	Day	A/noon	Night	Rotating Crew	Total Manning
MINE SITE					
Dragline Operator	2	2	2	2	
Dragline Greaser	1	1	1	1	
Driller	13	8	4	–	
Excavator Operator	11	11	4		
Rear Dump Operator	27	27	10	–	
Dozer Operator – Tracked	7	6	5	1	
Dozer Operator – Rubber Tyred	1	1			
Front-End-Loader Operator	1	1			
Scraper Operator	1				
Shotfirers	3				
Grader Operator	2	2	2		
Water Truck Operator	2	2	2		
Cable Carrier Operator	1				
Grease/Lube Truck Driver	2	2			
Tipmen and Labourers	12	7	4		
Excavator Groundsman	2	2	2		
TOTAL MINE SITE	88	72	36	4	200
PIT HEAD					
Watchman/Security	1	1	1	1	
Bathhouse Attendant	1	1	1	1	
CPP Foreman	1	1			
CPP Operators and Labourers	18	18	3		
Truck Driver (Wash Plant)	1	1			
CPP Front-End-Loader Operator	1	1			
ROM Spotter	2	2			
Crane/Fork Lift Drivers	3	3			
Stores Assistants	3	3	3	1	
Stores Truck/Van Drivers	2	2			
Workshop personnel including Fitters, Welders, Electricians, Greasers etc.	48	44	30		
TOTAL PIT HEAD	81	77	38	3	199
SUMMARY					
Staff	34	8	6	–	48
Wages Personnel	169	149	74	7	399
Absentee Allowance	9	8	3	–	20
TOTAL MANNING REQUIREMENTS	212	165	83	7	467

TABLE 10

MINE SITE STAFF

Staff	Day Shift	Afternoon Shift	Night Shift
Mine Manager	1		
Production/Pit Manager	1		
Senior Mining Engineer	1		
Drill and Blast Engineer	1		
Planning Engineer	1		
Mine Accountant	1		
Industrial Relations Officer	1		
Plant Manager	1		
Mechanical Engineer	1		
Electrical Engineer	1		
Shift Supervisor	1	1	1
Pit Foremen	3	3	2
CPP Supervisor	1	1	
Chemist	1		
Laboratory Assistant	2		
Office Manager	1		
Pay Clerk	1		
Cost Clerk	2		
Stores Supervisor	1		
Surveyor	1		
Survey Assistants	2		
Typist	2		
Receptionist/Telephonist	1		
Service Foremen	3	3	3
Plant Clerk	1		
Secretary	1		
TOTAL	**34**	**8**	**6**

TABLE 11
DRAYTON COAL MINE
SUMMARY OF PHYSICAL & CHEMICAL PROPERTIES

	BROUGHAM SEAM				GRASSTREES SEAM				THIESS SEAM						PUXTREES SEAM				BALMORAL SEAM				
LABORATORY TEST CODE	1 *	3	3	4	1 *	3	3	4	1 *	2(a)	2(b)	3	3	4	1 *	3	3	4	1 *	3	3	4	5
SAMPLE CONDITION	RAW	RAW	Cum. Floats at SG. 1·45	Cum. Floats at SG. 1·45 plus <30#	RAW	RAW	Cum. Floats at SG. 1·45	Cum. Floats at SG. 1·45 plus <30#	RAW	RAW	RAW	RAW	Cum. Floats at SG. 1·45	Cum. Floats at SG. 1·45 plus <32#	RAW	RAW	Cum. Floats at SG. 1·45	Cum. Floats at SG. 1·45 plus <30#	RAW	RAW	Cum. Floats at SG. 1·45	Cum. Floats at SG. 1·45 plus <30#	RAW
YIELD %				78·0				87·5						86·7				82·7				79·5	
PROXIMATE ANALYSIS (% air dry basis)																							
Inherent moisture %	7·0	5·0	3·7	2·3	7·0	3·5	3·2	2·3	7·0	1·5	4·02	3·7	4·1	2·4	7·0	4·2	3·7	2·7	7·0	3·4	4·2	2·8	3·5
Volatile matter %	35·3	35·0	36·9	38·0	34·8	37·1	39·0	38·8	36·1	40·9	38·86	39·0	39·1	39·3	35·5	34·4	36·4	35·1	32·6	35·1	35·2	35·3	36·3
Ash %	8·6	10·2	6·8	7·9	16·6	12·2	6·9	7·5	8·1	5·2	5·28	7·6	5·0	6·2	10·2	8·9	6·3	8·5	11·4	9·1	6·9	9·5	6·8
Fixed Carbon %	49·1	49·8	52·6	51·8	41·6	47·2	50·9	51·4	48·8	52·4	51·86	49·7	51·8	52·1	47·3	52·5	53·6	53·7	49·0	52·4	53·7	52·4	53·4
TOTAL SULPHUR %	0·79	0·72	0·73	0·73	0·77	0·87	0·73	0·76	0·91	0·83	1·04	0·83	0·80	0·86	0·84	0·89	0·75	0·72	0·95	0·86	0·77	0·88	0·90
Pyritic Sulphur %			0·02	0·01			0·01	0·01		0·02			0·03	0·01			0·06	0·00			0·03	0·02	
Sulphate Sulphur %			0·08	0·09			0·04	0·06		0·13			0·07	0·10			0·17	0·08			0·17	0·20	
Organic Sulphur %			0·63	0·63			0·68	0·69		0·68			0·70	0·75			0·52	0·64			0·57	0·66	
PHOSPHORUS %			0·034				0·016		0·01				0·007		0·02		0·015		0·05		0·091		0·08
CHLORINE %			0·03				0·03		0·03				0·03		0·04		0·02		0·04		0·04		0·03
ULTIMATE ANALYSIS (% dry ash free basis)																							
Carbon %			82·34				81·75		83·50		78·19		82·45		83·50		82·77		84·40		83·5		83·66
Hydrogen %			5·18				5·09		5·90		5·16		5·07		5·40		5·16		5·40		4·93		5·42
Nitrogen %			1·81				1·84		1·90		1·73		1·78		1·90		1·76		1·90		1·78		1·63
Sulphur %			0·82				0·81				1·07		0·88				0·83				0·87		0·99
Oxygen %			9·85				10·51		8·70		8·35		9·82		9·20		9·48		8·30		8·92		8·31
Ash %											5·5												
			100·00				100·00		100·00		100·00		100·00		100·00		100·00		100·00		100·00		100·00
SPECIFIC GRAVITY		1·33		1·39		1·34		1·29				1·30		1·31		1·31		1·32		1·31		1·33	1·34
SPECIFIC ENERGY MJ/kg.	28·66	28·19	31·45	32·06	26·08	28·99	31·10	32·17	29·19	32·28	31·33	30·76	31·45	32·85	28·18	29·67	31·14	31·95	27·75	30·02	31·36	31·23	30·52
B.S. CRUCIBLE SWELL No.			1				3			5½	6		4				1				1		2
GRAY KING COKE TYPE			E				F			G2			G				D				E		
HARDGROVE GRINDABILITY INDEX			45				44						44				46				54		46
ASH FUSION PROPERTIES (reducing atmos.)																							
Deformation Temp. °C.			1440	1350			1560	1450						1420			1460	1450			1270	1230	1415
Hemisphere Temp °C.			1490	>1560			1580	>1560						>1560			1520	1540	1350		1300	1320	1540
Flow Temp. °C.			1540	>1560			1590	>1560						>1560			1550	>1560			1340	1400	1570
ASH ANALYSIS																							
SiO_2 %			66·2				67·0				55·76		65·8				64·0				48·5		44·8
Al_2O_3 %			22·9				23·6				24·99		25·8				23·7				26·6		27·0
Fe_2O_3 %			3·68				2·40				10·38		4·08				5·72				7·12		7·21
TiO_2 %			1·32				1·40				1·69		1·57				1·34				0·99		1·95
Mn_3O_4 %			0·02				0·02				0·06		0·01				0·03				0·04		0·06
CaO %			1·17				0·94				1·68		0·54				1·00				6·68		7·76
MgO %			0·74				0·66				0·48		0·38				0·76				0·83		1·67
Na_2O %			0·38				0·36						0·20				0·25				0·19		0·16
K_2O %			0·70				0·62						0·32				0·59				0·13		0·13
P_2O_5 %			1·11				0·50				0·24		0·30				0·52				2·90		2·64
SO_3 %			0·62				0·50				0·34		0·20				0·35				4·00		5·21
Loss on ignition or undetermined %			1·16				2·00				4·38		0·80				1·74				2·02		
SILICA RATIO# %			92·0				94·0				82·0		93·0				90·0				77·0		73·0
GIESLER PLASTOMETER TEST-ASTM 1812-66																							
Initial Softening Temp. °C.			370				366				302		378				366				364		
Re-solidification Temp. °C.			444				452				454·5		468				444				444		
Fusion Temp. at 5 DD/Min.			416				406				374		422				408				412		
Maximum Fluidity Temp. °C.			419				428				432		436				419				416		
Maximum Fluidity DD/Min.			5·5				72				log 3·79		42				9·0				5·5		
Range Soften to Re-solidification °C			74				86				152·5		90				78				80		
Temp. Range at 1 DD/Min.			34				47				80·0		52				40				38		

LEGEND

1. Electricity Commission of N.S.W. (1964, 1966-67)
2(a). Thiess Seam Bulk Shaft Sample (J.C.B. Laboratory, 1965)
2(b). Thiess Seam Bulk Shaft Sample (Yawata Laboratory, 1965)
3. Combined Bulk Samples from 9 inch Cores (A.C.I.R.L. Laboratory, 1970)
4. Samples from "T" series Boreholes (J.C.B. Laboratory, 1971)
5. Samples from 130mm Core Drilling (Thiess & A.C.I.R.L. Laboratory, 1976)

Note: * % "as mined" basis

$\dfrac{100 \times SiO_2}{SiO_2 + Fe_2O_3 + CaO + MgO}$

D1/1/202-8

APPENDIX 1

EQUIPMENT SPECIFICATIONS

The following specifications are presented to indicate the types of equipment utilised on the site.

DRAGLINE

Bucyrus Erie 1370W electric walking dragline

	1370W
Boom length	99.06
Boom angle	38^{o}
Max. dumping radius	87.2 m
Max. digging depth	47.0 m
Max. dumping height	48.0 m
Max. suspended load	134,100 kg
Bucket capacity	44.35 m^3
Base diameter	17.6 m
Walking shoes	3 m x 18.9 m
Length of step	2.7 m
Walking speed	0.23 km/hr
M.G. set drives	Two 3000 hp Synchronous motors
Hoist motors (blown)	Four 1300 hp at 475V
Drag motors (blown)	Four 1045 hp at 475V
Swing motors (blown)	Four 640 hp at 475V
Propel motors (unblown)	Four 375 hp at 475V
Working weight	3045 tonnes

MINING FACE SHOVEL

Bucyrus Erie 295B electric shovel

Boom length	15.2 m
Boom angle	46^{o}
Max. cutting radius	19.2 m
Max cutting height	14.6 m

Radius of level floor at face	12.7 m
Max. dumping radius	16.2 m
Max. dumping height at max. dumping radius	9.2 m
Max. digging depth below ground level	2.6 m
Bucket capacity	16.0 m^3
Overall width	8.7 m
Induction motor	800 hp continuous, 2000 hp intermittent
Hoist motor (blown)	One 800 hp at 475V
Swing motor (blown)	Two 195 hp at 475V
Crowd motor (blown)	One 195 hp at 475V
Propel motor (blown)	One 375 hp at 475V
Working weight	549 tonnes

HYDRAULIC EXCAVATOR

Orenstein and Koppel (O & K) RH75–800

	Shovel Operation	Backhoe Operation
Max. cutting/dumping radius	10.5 m	15.0 m
– corresponding cutting/dumping ht	9.8 m	5.0 m
Max. cutting/dumping height	10.0 m	11.0 m
–corresponding cutting/dumping radius	7.4 m	11.5 m
Max. digging depth	2.9 m	8.6 m
Rated bucket capacity (rock type)	7.6 m^3	5.5 m^3
Engine	2 x water cooled Cummins diesel engine (NTA – 855 – C)	
Horsepower (DIN 6270)	2 x 348 hp at 2100 rpm	
Hydraulic horsepower	2 x 270 hp	
Operating weight	129 tonnes	

Orenstein and Koppel (O & K) RH40–700

	Shovel Operation
Max. cutting/dumping radius	9.0 m
– corresponding cutting/dumping height	3.0 m
Max. cutting/dumping height	8.7 m
– corresponding cutting/dumping radius	5.9 m
Max. digging depth	4.0 m
Rated bucket capacity (coal type)	6.0 m^3
Engine	1 x Deutz air-cooled diesel engine (BF 12 L413)
Horsepower (DIN 70020)	1 x 432 hp at 2300 rpm
Operating weight	82 tonnes

FRONT-END-LOADER

Michigan 475B Wheel Type Loader

Overall operating height	7.9 m
Dump height	4.1 m
Dumping reach	1.7 m
Rated bucket capacity (rock type)	9 m^3
Engine	Cummins VTA – 1710 – C 700 diesel 680 hp max., 617 hp flywheel at 2000 rpm
Operating weight	70700 kg.

OVERBURDEN BLAST HOLE DRILL

Bucyrus Erie 45R Rotary Drill

Drill pipe length (optional)	9.9 m – 16.7 m (single–double pass drilling)
Blast hole diameter	170 mm – 270 mm (6¾ in – $10^5/_8$ in.)
Engine	Cummins diesel NT–855–P–380, rated 320 hp at 2000 rpm
Pull down force	31,818 kg.
Operating weight	67,727 kg.

COAL AND PARTING BLAST HOLE DRILL

Gardner Denver RDC 30

Drill pipe length	7.6 m (single pass)
Blast hole diameter	159 mm – 200 mm (6¼ in. – $7^7/_8$ in.)
Engine	G.M. 8V–71–N diesel
Pull down force	13,600 kg.
Operating weight	29,384 kg.

TRUCKS

(a) *Haulpak 120B Electric Rear Dump*

Engine	G.M. DDAD 12V–149T
Rated brake horsepower	1200 hp at 1900–rpm
Net vehicle weight	79,063 kg.
Payload	108,863 kg.
Solid volume overburden (struck)	52.7 m^3

(b) Euclid R85 Rear Dump

Engine	Cummins VTA–1710C
Gross vehicle horsepower (SAE)	800 hp at 2100 rpm
Net vehicle weight	51,600 kg.
Payload	77,100 kg.
Solid volume overburden (struck)	39.8 m^3
Coal body capacity (side boards)	65 tonne

(c) Haulpak Model 50 Rear Dump

Engine	Cummins VTA–1710C
Rated brake horsepower	635 hp at 2100 rpm
Net vehicle weight	34,174 kg.
Payload	45,360 kg.
Solid volume overburden (struck)	23.55 m^3

SCRAPERS

CAT 623B Elevating Scraper	
Scraper bowl capacity (heaped)	16.8 m^3
Flywheel horsepower	330 kw at 1900 rpm
Net weight	31,736 kg.

APPENDIX 2

DRAGLINE PRODUCTION ESTIMATES

The dragline works on a three shift basis, five days a week for the first eight years of operation and then operates seven days a week when working in the multi-seam situation in the East Pit.

Shift hours per week on five days operation = 5 x 24 = 120 hours
Deduct ½ hour/shift for shift changeover
Allow machine availability of 80%
Working hours/week = (120) - (15 x ½) x 80% = 90 hours
Working weeks/year = 46
Operating hours per year = 4140 hours
Walking time is included in the 20% non working time.

When working on a seven day week operation, maximum operating hours per year were taken as 5888 hours.

The annual production for the dragline was determined by considering the various dragline operations and applying the corresponding output rate. Outputs were based on a 50 minute hour. Relevant information is as follows:

Maximum suspended load	...	134,100 kg
Bucket rating heavy duty	...	1,219 kg/m^3
Overburden bank density	...	2,470 kg/m^3
Overburden swell	...	30 %
Bucket Fillability	...	95 %
Bucket Factor	...	0.73
Bucket Capacity	...	44.35 m^3 (58 yd^3)
Solid volume handled by bucket	...	32.37 m^3

Dragline operations consist of:

WEST PIT

North Cuts

(a) Digging to Thiess seam - no rehandle.
(b) Digging to Puxtrees seam - rehandle key cut from operation (a).
(c) Digging to Balmoral seam by extended bench method with large rehandle involved.

South Cuts

(d) Chopping to Puxtrees Seam (chopping operations above the dragline seat level are inefficient and output is taken as 80% of similar digging operation).
(e) Digging 70% of Balmoral overburden from highwall side.
(f) Working from bench on spoils and cross-dragging remaining 30% of dig down to Balmoral Seam including some rehandle.

(g) Working from spoil bench and chopping parting between B1 and B3 plus remaining rehandle. (Take 90% of digging output).

Operation	Swing Angle	Overburden depth (m)	Theoretical cycles/hr	Cycles/ 50 min.hr	Output 50 min. hr m^3/hr
(a)	90°	10	77.77	64.81	2093
(b)	90°	10	77.77	64.81	2093
(c)	135°	20	63.94	53.25	1720
(d)	180°	10	54.31	45.26	1165
(e)	90°	20	77.77	64.81	2093
(f)	150°	20	60.38	50.32	1623
(g)	150°	30	60.38	50.32	1460

EAST PIT

Cuts 1E to 5E (70m)

(a) Digging key cut to Balmoral and extending dragline bridge.

(b) Sitting out on extended bench, cross-dragging and swinging round to dump at 150°.

Cuts 6E to 18E (50m)

(c) Digging from G5 to T2 with key cut spoils being dumped on P2 pavement and remainder of spoil being cast over lower highwall.

(d) Chopping parting T2 - P2. (Take 80% of digging rate).

(e) Digging key cut to Balmoral seam and extending bridge.

(f) Sitting on bridge and cross-dragging to form final spoil profile.

Operation	Swing Angle	Overburden depth (m)	Theoretical cycles/hr	Cycles/ 50 min.hr	Output 50 min.hr m^3/hr
(a)	90°	10	77.77	64.81	2093
(b)	150°	40	60.38	50.32	1623
(c)	90°	10	77.77	64.81	2093
(d)	180°	10	54.31	45.26	1165
(e)	90°	20	77.77	64.81	2093
(f)	150°	40	60.38	50.32	1623

NOTE: Although the maximum operating radius of the dragline is 87.2m an operating radius of 83m was used for the determination of Rehandle quantities.

APPENDIX 3

SHOVEL/LOADER PRODUCTION ESTIMATES

(a) *Bucyrus Erie 295B electric shovel (overburden removal).*

Rated bucket capacity	16.0 m^3
Bucket Factor	0.6
Solid volume handled by bucket	9.60 m^3
Average cycle time	30 secs.
Output per 50 min. hour	960 m^3/hr.

Shift hours/week : 3 x 8 x 5	= 120 hours
Deduct meal breaks : start up : shut down : 3 x 1 x 5	= 15 hours
Allow 80% mechanical availability	
Working hours per week	= 84 hours
Output per week	= 80700 m^3
Working hours per 44 week year	= 3696 hours
Output per 44 week year	= 3,551,000 m^3

(b) *O & K RH75 hydraulic excavator (overburden, coal and partings removal).*

Rated bucket capacity (rock)	7.6 m^3
Bucket factor	0.61
Solid volume handled by bucket	4.64 m^3
Average cycle time	30 secs.
Output per 50 min. hour	464 m^3/hr.

Shift hours/week (overburden) : 3 x 8 x 5	= 120 hours
Deduct meal breaks : start up : shut down : 3 x 1 x 5	= 15 hours
Allow 80% mechanical availability	
Working hours/week	= 84 hours
Output per week	= 38976 m^3
Working hours per 44 week year	= 3696 hours
Output per 44 week year	= 1,715,000 m^3

When used on coal and parting removal the machine is working at an average rate less than this estimated output.

(c) *O & K RH40 hydraulic excavator (coal removal)*

Rated bucket capacity (coal)		6.0 m^3
Bucket factor		0.61
Solid volume handled by bucket		3.66 m^3
Average cycle time		30 secs.
Output per 50 min. hour		366 m^3/hr.
Shift hours/week : 2 x 8 x 5	=	80 hours
Deduct meal breaks : start up : shut down : 2 x 1 x 5	=	10 hours
Allow 80% mechanical availability		
Working hours/week	=	56 hours
Output per week	=	20496 m^3
Working hours per 44 week year	=	2464 hours
Output per 44 week year	=	902,000 m^3

On coal excavation the machine is working at an average rate less than this estimated output.

(d) *Michigan 475B Front-End-Loader (coal and parting removal).*

Rated bucket capacity (rock)		9.0 m^3
Bucket factor		0.60
Solid volume handled by bucket		5.4 m^3
Average cycle time		40 secs.
Output/50 min. hour		405 m^3/hour
Shift hours/week : 2 x 8 x 5	=	80 hours
Deduct meal breaks : start up : shut down : 2 x 1 x 5	=	10 hours
Allow 75% mechanical availability		
Working hours/week	=	52½ hours
Output per week	=	21,260 m^3
Working hours per 44 week year	=	2310 hours
Output per 44 week year	=	935,000 m^3

APPENDIX 4

OVERBURDEN, COAL AND PARTING PREPARATION

Pit	Material	Location	Quantity ('000's m^3)	Pattern (m x m)	Drilling (m)	Loading Ratio m^3/kg.
NW	Overburden	Box Cut	1125	6 x 6	31250	2.95
	Overburden	Remaining	17701	7 x 7	361243	2.95
	Coal	Balmoral	2677	5 x 5	107080	7.60
WEST	Overburden	Box Cut	3080	6 x 6	85554	2.95
	Overburden	Shovel pre-strip	3928	6 x 7	93522	2.95
	Overburden	G5 to T2 & T2 to P2	8641	6 x 5	288033	2.95
	Overburden	P2 to B1	17462	7 x 8	311821	2.95
	Coal	B31 & B32	4711	3 x 3	523442	7.60
	Partings	B1 – B31	3872	3 x 4	322663	3.20
EAST	Overburden	All shovel Overburden	72052	6 x 7	1715521	2.95
	Overburden	G5 to T2	18099	6 x 7	430857	2.95
	Overburden	P2 to B1	53144	7 x 8	948964	2.95
	Parting	T2 to P2	8937	4.5 x 4.5	441328	3.20
	Parting	B1 – B3	6370	3 x 4	530832	3.20
	Coal	B3	14234	4.5 x 4.5	702904	7.60

Pit	Material	Average Thickness
NW	Balmoral coal	10 m
West	Overburden G5 to T2 and T2 to P2	8 m
West	Overburden P2 to B1	19 m
West	Partings B1–B3	4 m
West	Coal B31 and B32	2.5 m
East	Overburden G5 to T2	10 m
East	Overburden P2 to B1	21 m
East	Partings B1–B3	4 m
East	Partings T2–P2	5 m
East	Coal B3	6 m

DRILLING ESTIMATES

Overburden Drills (Bucyrus Erie 45R drilling 229 mm dia. holes)

Annual drilling requirements were determined throughout the mine life and drills were scheduled to operate on a sufficient number of shifts to meet these requirements.

The drills are utilised on a five day week basis, working two or three shifts per day.

Considering an 8 hour shift.

Allow: 1 hour for meal break, start-up and shut-down.
80% mechanical availability.
an average penetration rate of 25 metres/hour.

i.e. working hours per shift = 5.6 hours.
Working hours/week = 28 hours.

Operating on 44 weeks/year, working hours/year = 1232 hours.

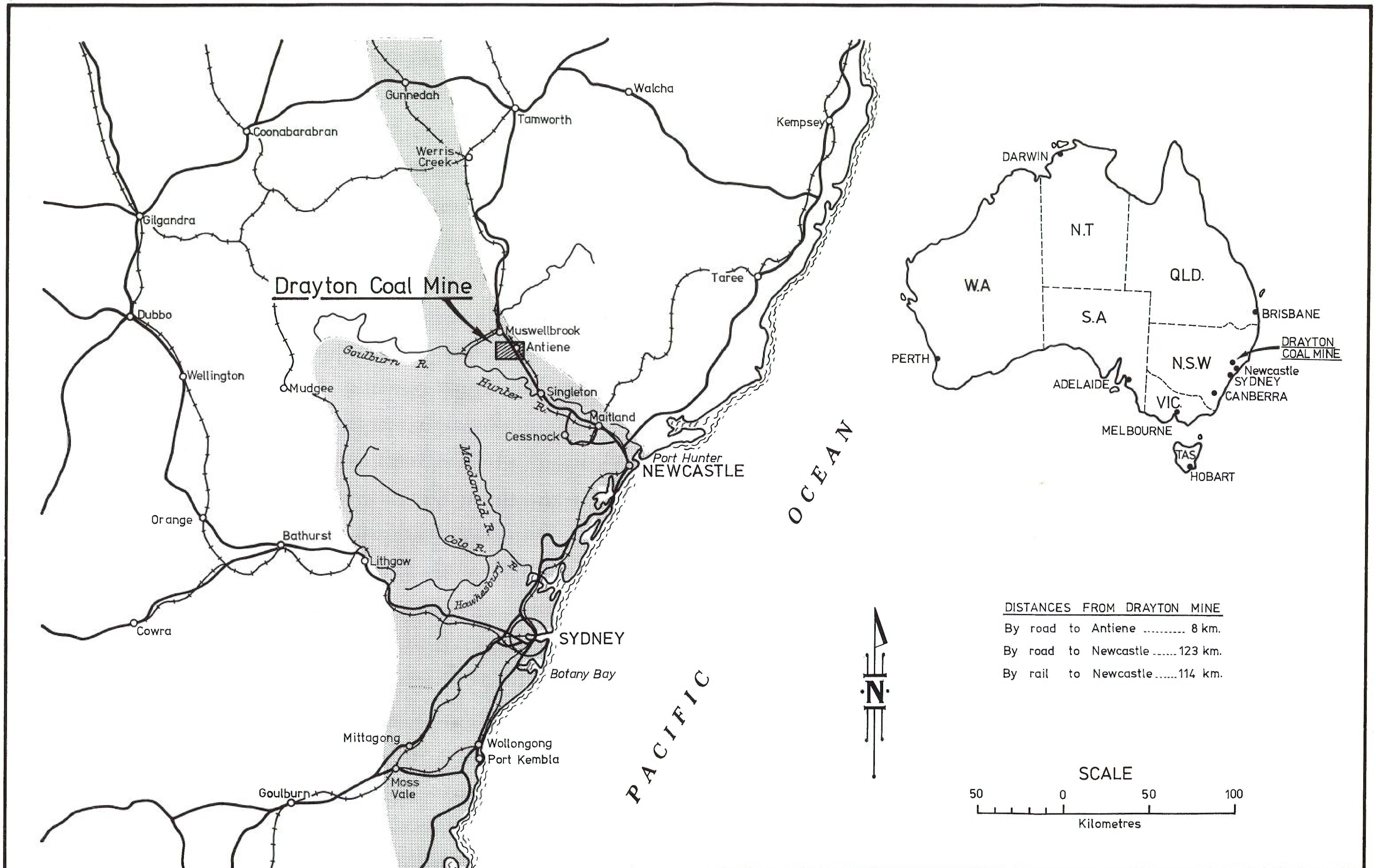
Drilling achieved per week for 28 hours = 700 metres.

For a double and triple shift operation the drilling achieved per week would be 1400 m and 2100 m respectively.

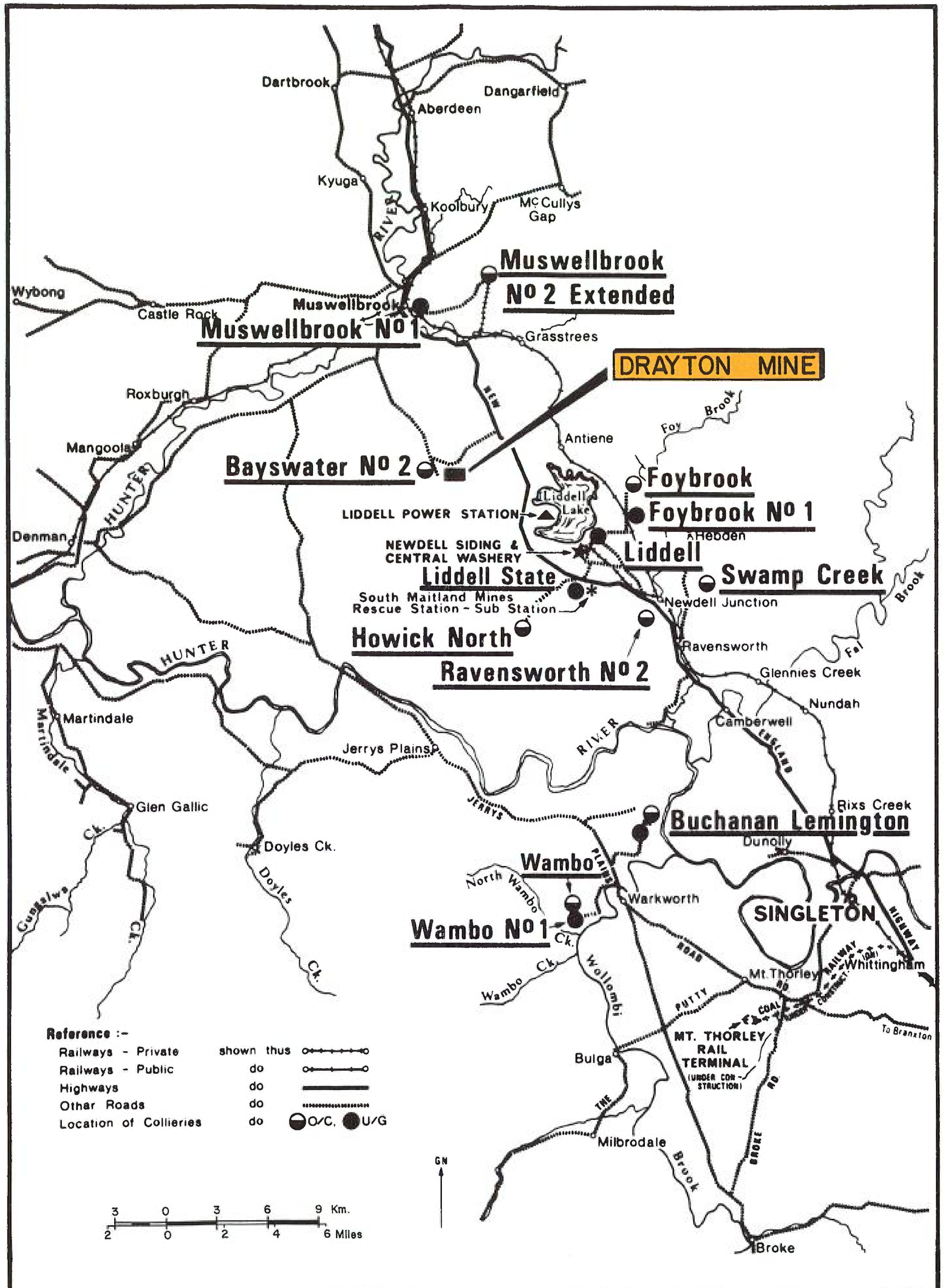
Coal and Parting Drills (Gardner Denver RDC 30 drilling 159 mm dia. holes)

Coal and Parting drills are worked on a single (8 hrs) or double (16 hrs) shift basis five days/week (i.e. 28 or 56 working hours/week).

Average penetration rates were taken as 35 metres/hour for drilling partings and 45 metres/hour for drilling coal. For a single and double shift operation in parting drilling this gives 980 and 1960 metres/week respectively and for a single and double shift operation in coal drilling this gives 1260 and 2520 metres/week respectively.

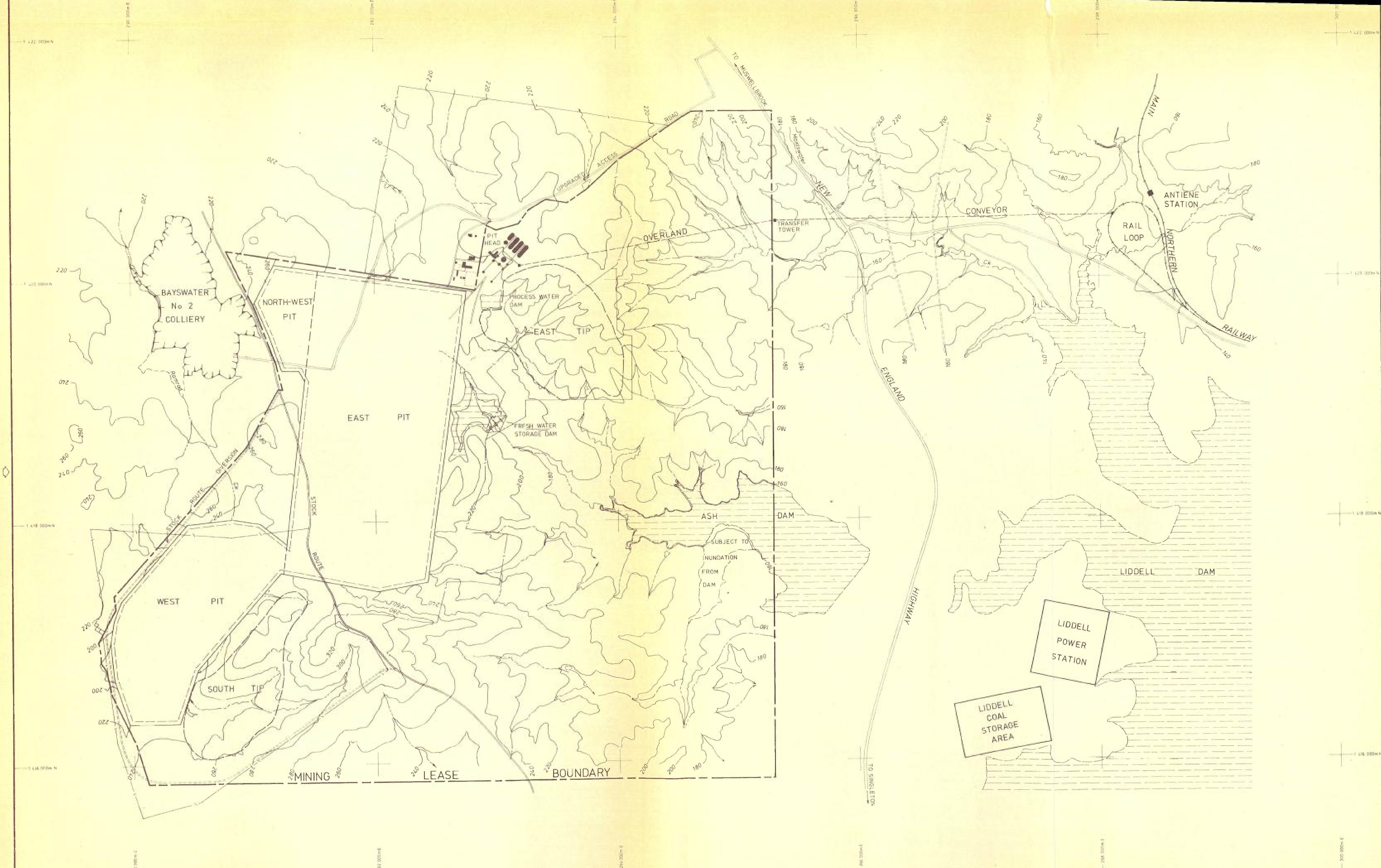


SCALE 1 : 2 000 000		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE LOCALITY MAP	FIGURE 1
DRAWN L.G.B	Nov. '76		DRAWING No D1/1/100-25
CHECKED PAR			
APPROVED			



Source of information Joint Coal Board of N.S.W.

SCALE 1:300 000	THIESS-SHELL DRAYTON CO-VENTURE	
DRAWN C.C. Nov. '77	HUNTER VALLEY COALFIELD	FIGURE 2
CHECKED KDS	NORTH WEST DISTRICT	DRAWING
APPROVED	LOCATION OF COLLIERIES	D1/1/100-32



SCALE 1:10 000

200 0 200 400 600 800

Metres

SCALE	1 10 000
DRAWING	DATE
DRAWN R B	7-7-78
CHECKED	10-7-78
APPROVED	
REVISED	

THIESS-SHELL DRAYTON CO-VENTURE

DRAYTON COAL MINE

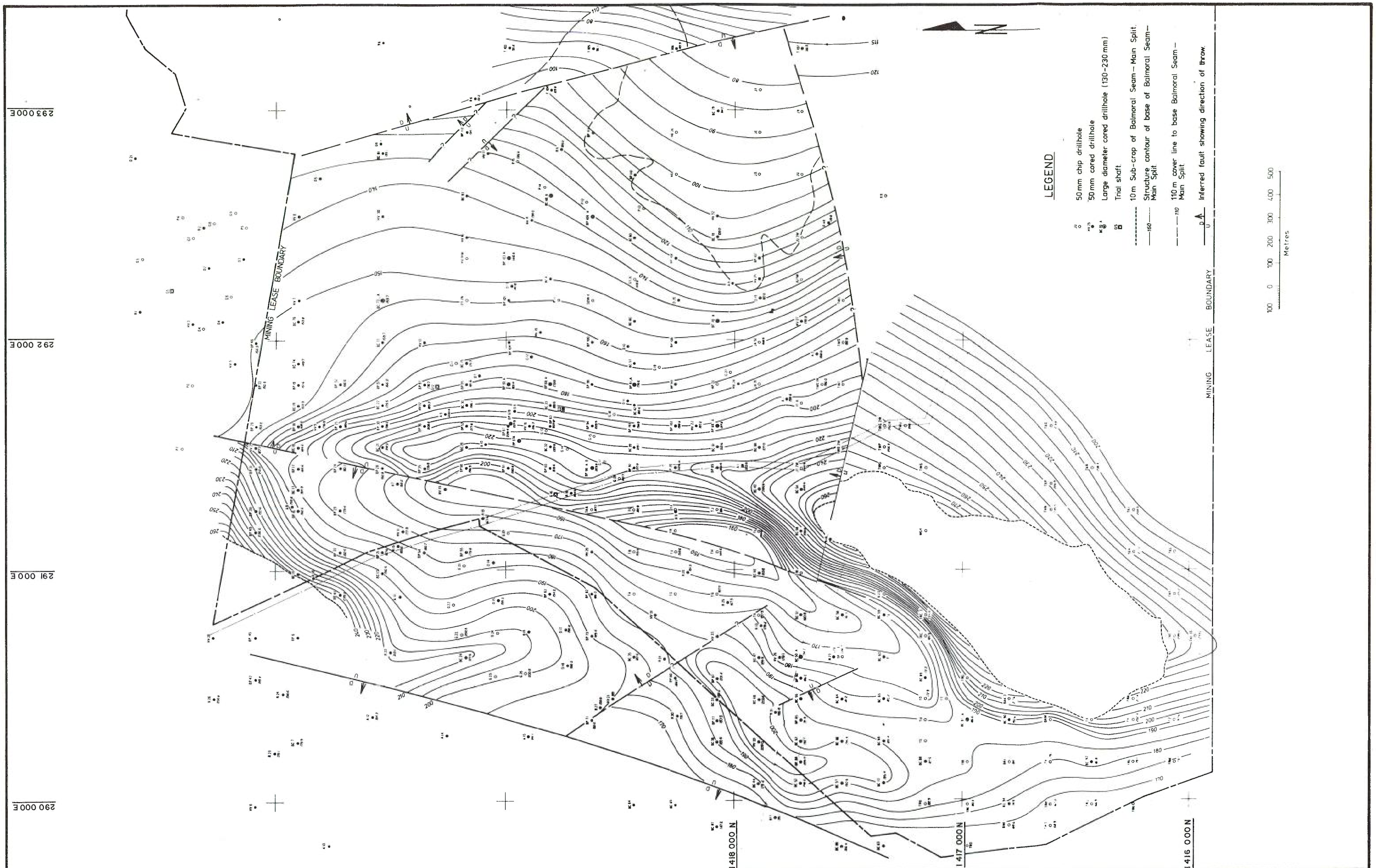
GENERAL ARRANGEMENT

FIGURE 3

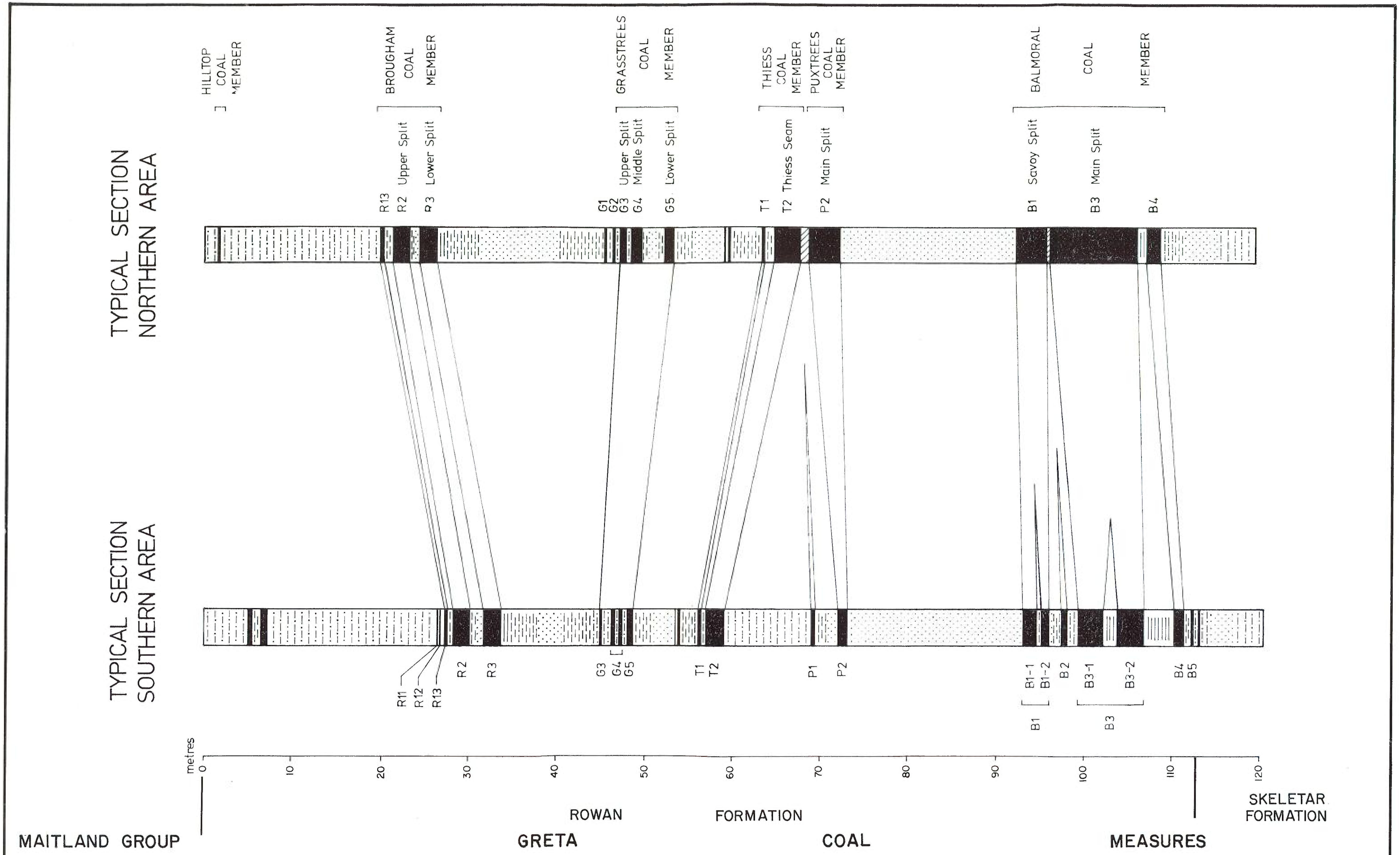
REVISION

DRAWING No. D1/1/300-24 R

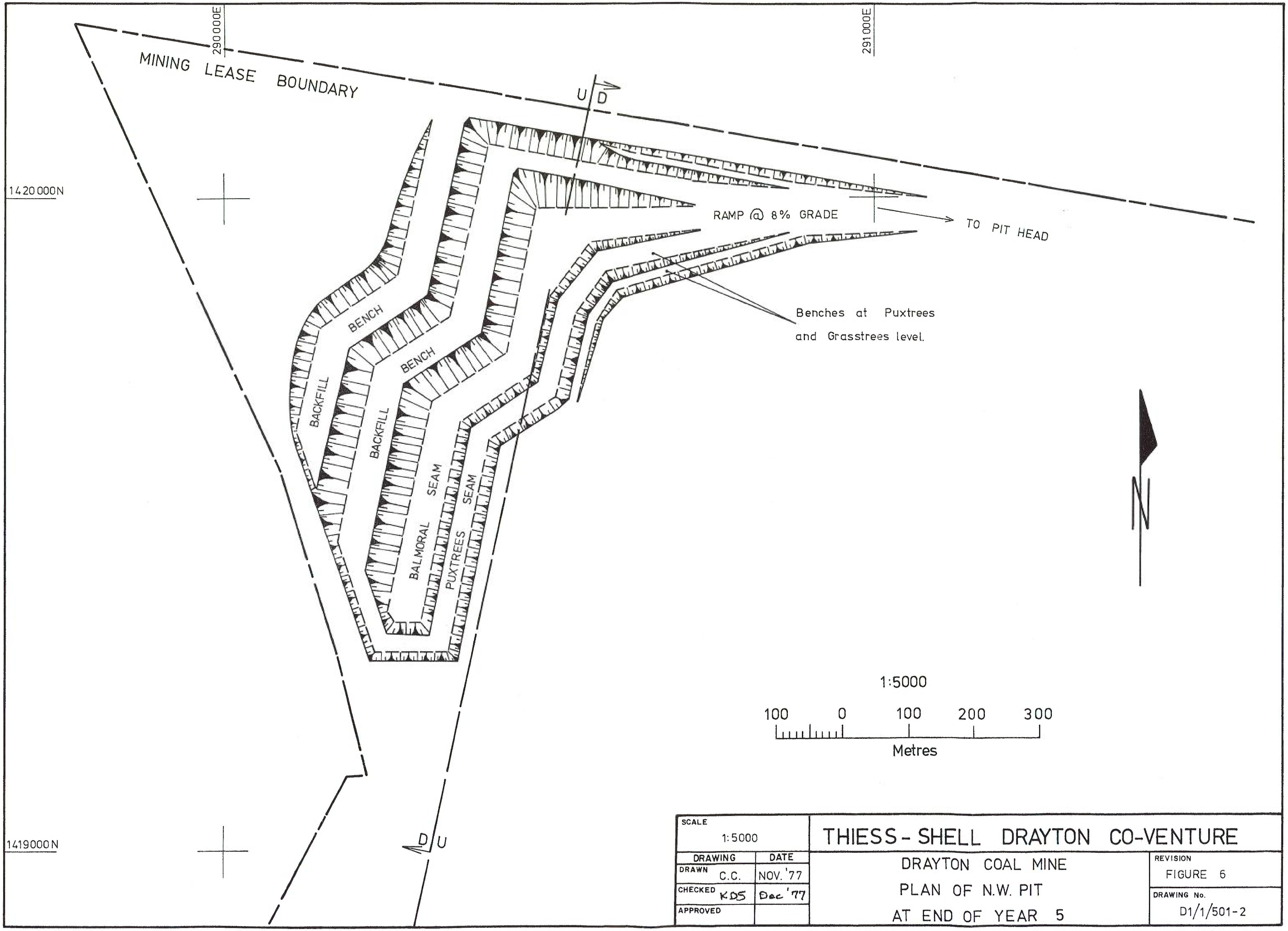
DRAWING No.	TITLE REFERENCE DRAWINGS	DATE	REV	BY	CKD	APP	ORIGINAL ISSUE DETAILS REVISIONS

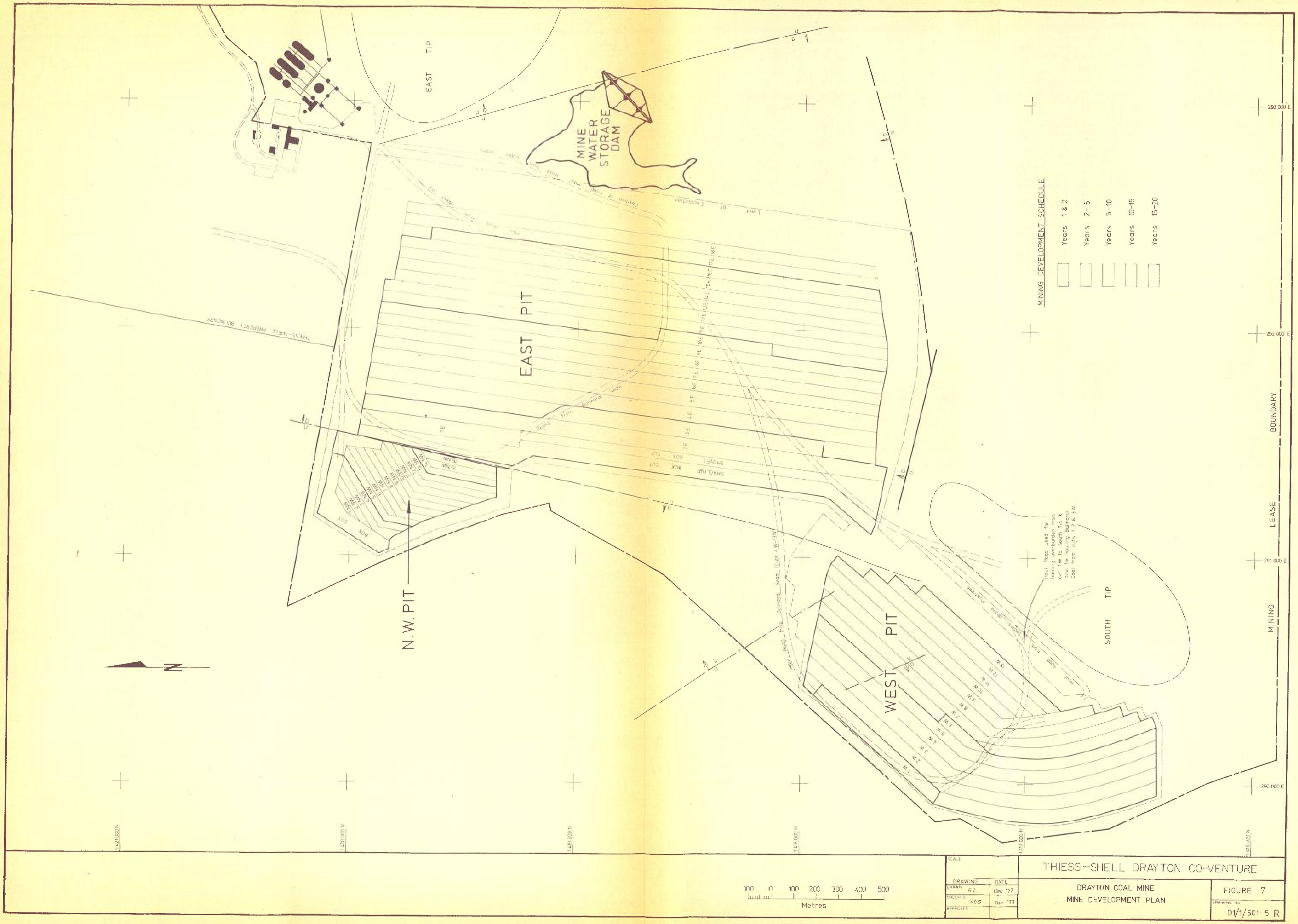


SCALE		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE BALMORAL SEAM-MAIN SPLIT (B3) GEOLOGICAL STRUCTURE	FIGURE 4
DRAWN G.M.	Nov.'77		
CHECKED KDS	Dec '77		
APPROVED			D1/1/201–118

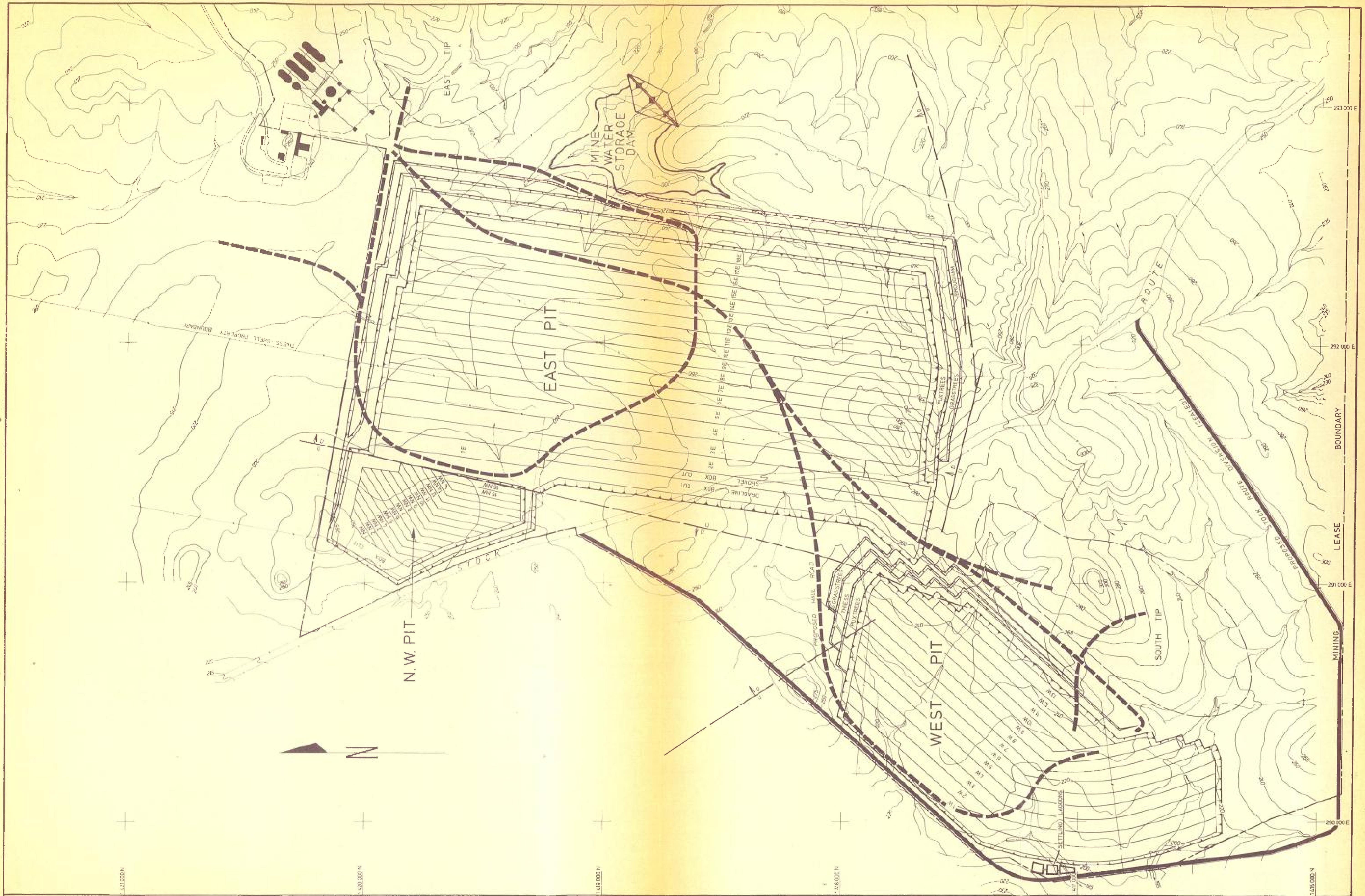


SCALE 1:400 Vert.		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE	FIGURE 5
DRAWN R.L.	Aug. '77	TYPICAL STRATIGRAPHIC SECTIONS	DRAWING No D1/1/201-112
CHECKED PAR	19·8·77		
APPROVED			


290 000E
291 000E
MINING LEASE BOUNDARY
U D
1420 000N
RAMP @ 8% GRADE
TO PIT HEAD
Benches at Puxtrees
and Grasstrees level.
BENCH
BENCH
BACKFILL
BACKFILL
BALMORAL SEAM
PUXTREES SEAM
N
1:5000
100 0 100 200 300
Metres
1419 000N
D U
SCALE
1:5000
THIESS - SHELL DRAYTON CO-VENTURE
DRAWING
DATE
DRAWN C.C.
NOV. '77
CHECKED KDS
Dec '77
APPROVED
DRAYTON COAL MINE
PLAN OF N.W. PIT
AT END OF YEAR 5
REVISION
FIGURE 6
DRAWING No.
D1/1/501-2



SCALE		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE	FIGURE 7
DRAWN R.L.	Dec '77	MINE DEVELOPMENT PLAN	DRAWING No.
CHECKED KDS	Dec '77		D1/1/501-5 R
APPROVED			

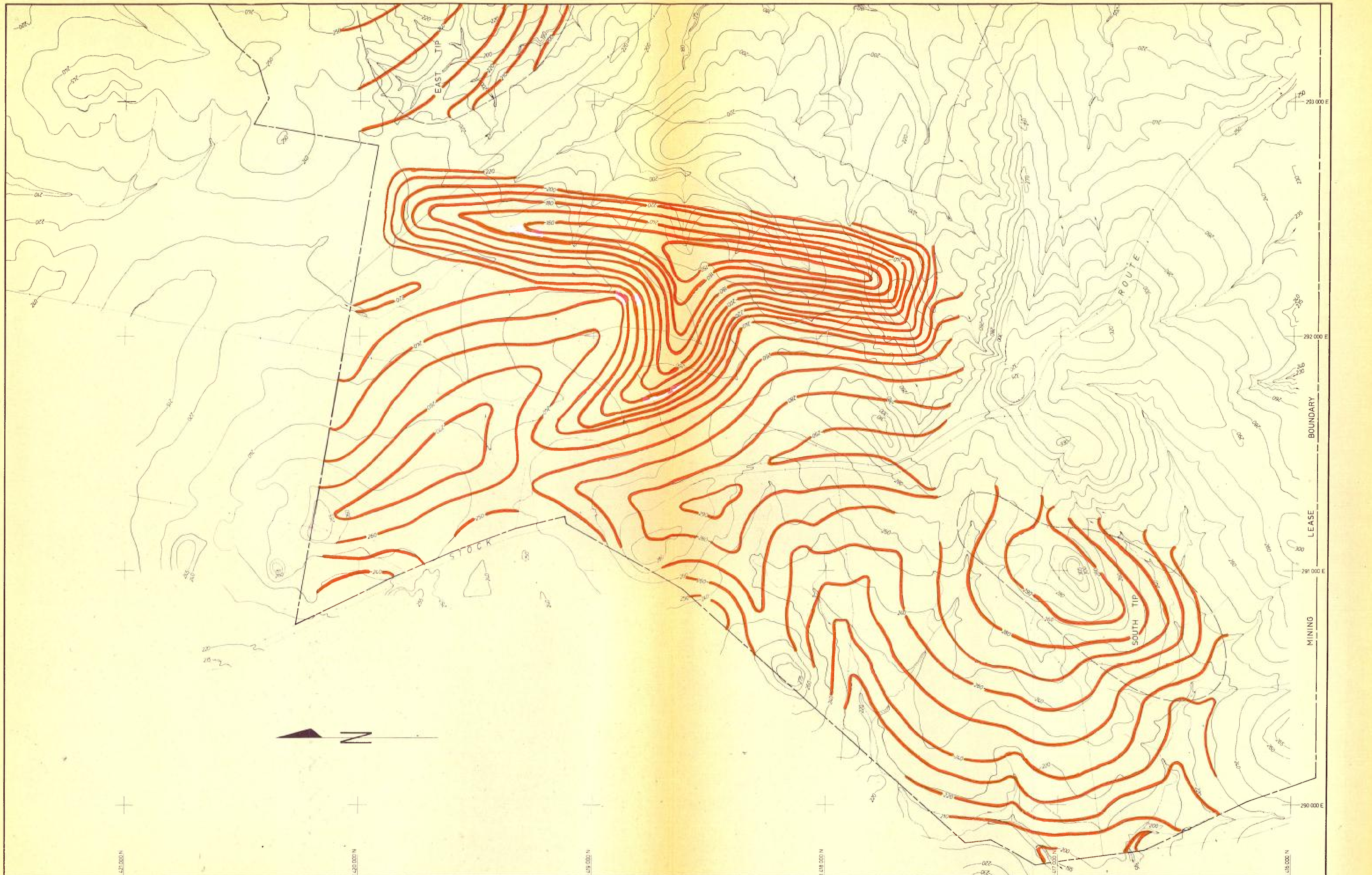


100 0 100 200 300 400 500
Metres

SCALE		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE OPENCUT MINING PLAN	FIGURE 8
DRAWN RL	Dec. '77		DRAWING No. D1/1/501-3 R
CHECKED KDS	Dec. '77		
APPROVED			

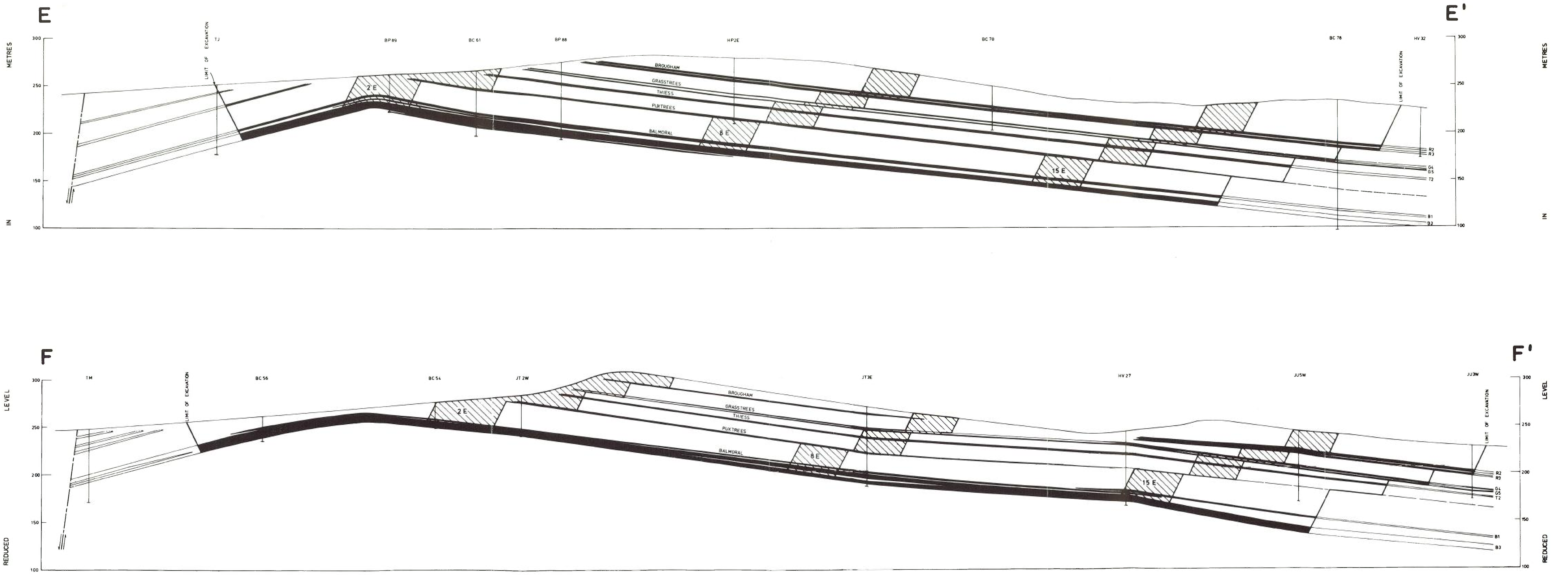

WEST PIT FIG 9

EAST PIT FIG 10



100 0 100 200 300 400 500
Metres

SCALE	THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING: DRAWN R.L. Dec '77; CHECKED KDS Dec '77; APPROVED	DRAYTON COAL MINE RESTORATION PLAN	FIGURE 11 DRAWING No. D1/1/501-6 R



LEGEND

Inferred fault showing direction of throw

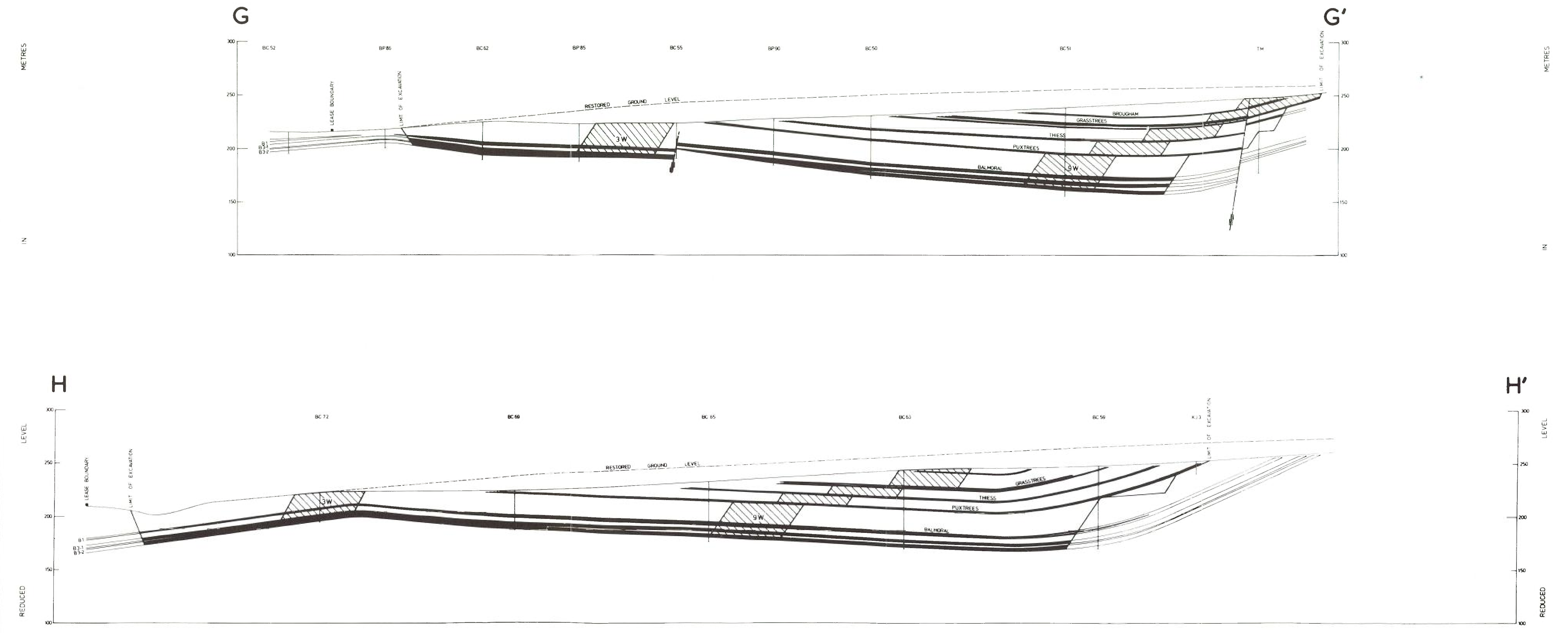
JT3E Exploratory drill hole

2E Working profile of strip cut

20 0 20 40 60 80

Metres

SCALE		THIESS-SHELL DRAYTON CO-VENTURE
DRAWING DRAWN C C	DATE OCT 77	DRAYTON COAL MINE EAST PIT CROSS SECTIONS E-E', F-F'
CHECKED		FIGURE 12
APPROVED		DRAWING No. 01/1/201-121



LEGEND

Inferred fault showing direction of throw

BC 65 Exploratory drill hole

9W Working profile of strip cut

20 0 20 40 60 80
Metres

DRAWING	DATE
	Oct 77

THIESS-SHELL DRAYTON CO-VENTURE

DRAYTON COAL MINE
WEST PIT
CROSS SECTIONS G-G', H-H'

FIGURE 13

DRAWING No 01/1/201-122

DRAYTON COAL MINE
ELEMENTS OF QUALITY CONTROL

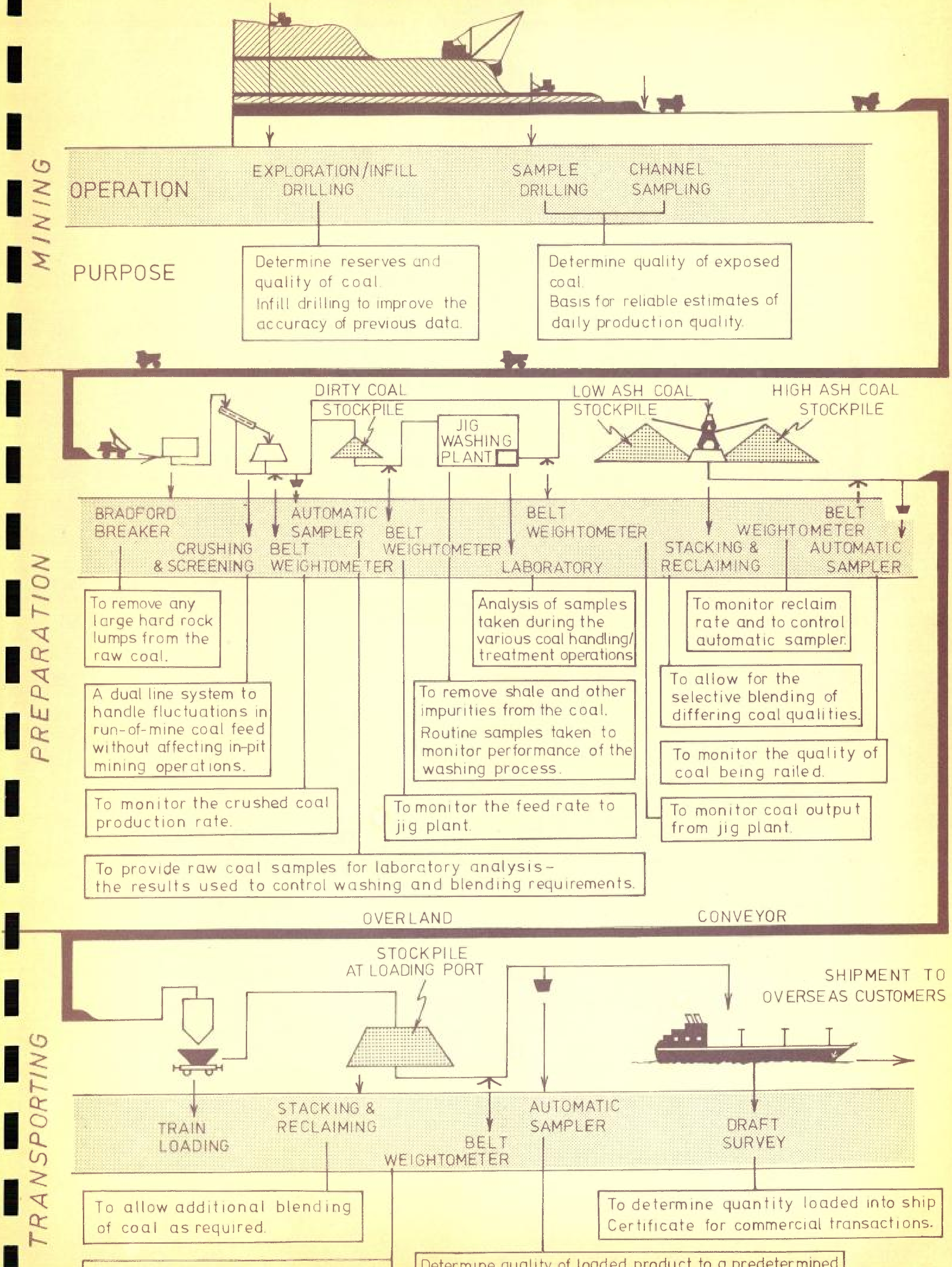
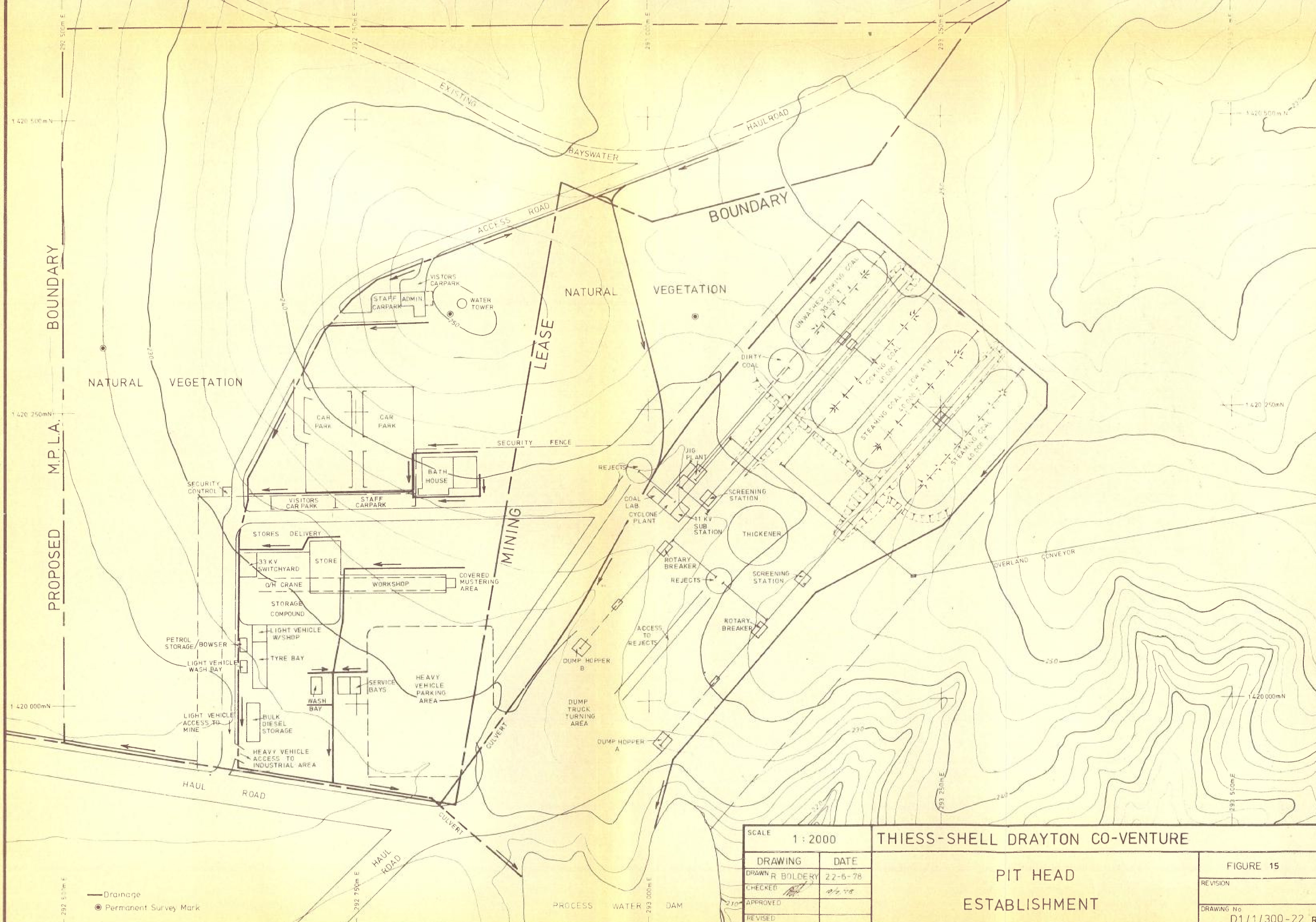
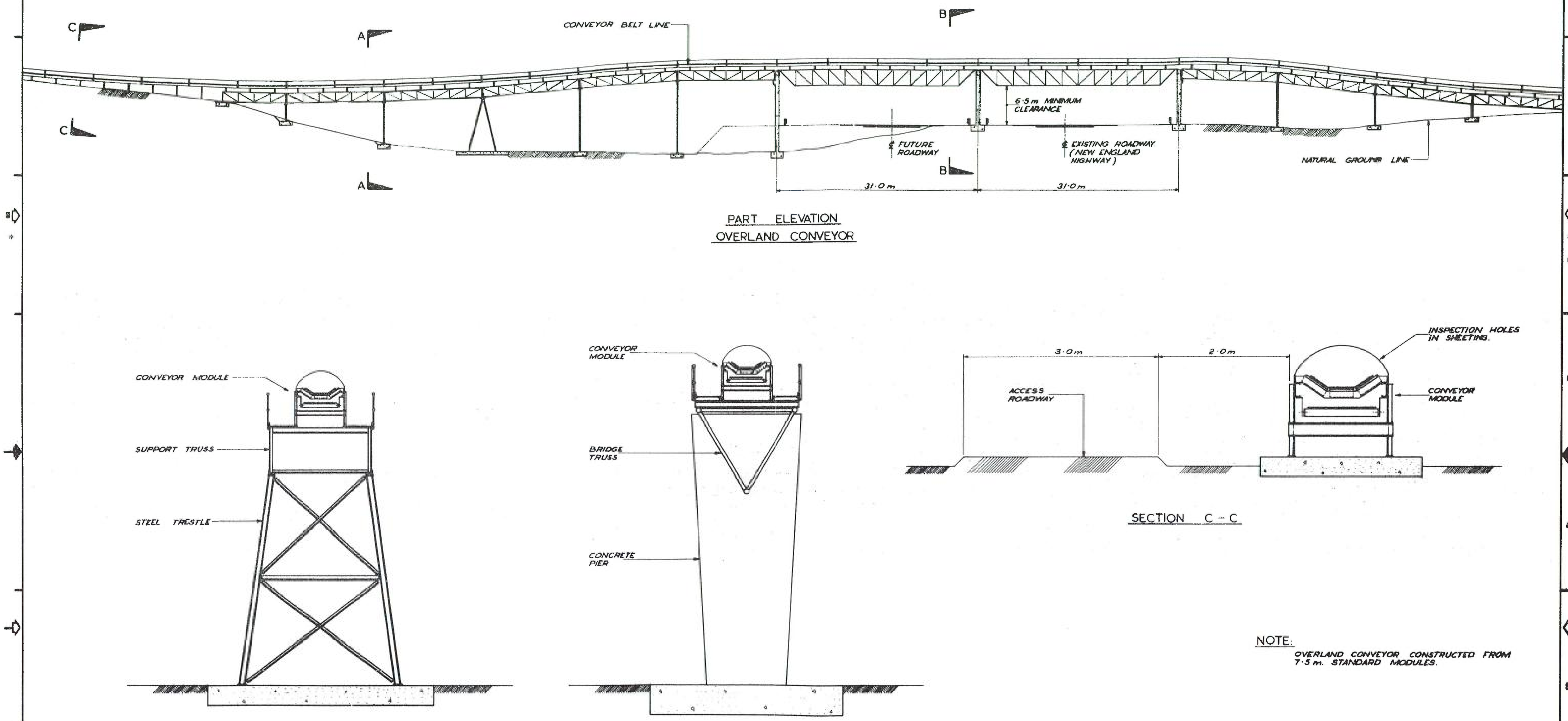


FIGURE 14


EXISTING
BAYSWATER
HAUL ROAD
ACCESS ROAD
BOUNDARY
PROPOSED M.P.L.A. BOUNDARY
MINING LEASE
NATURAL VEGETATION
NATURAL VEGETATION
VISTORS CARPARK
STAFF CARPARK
ADMIN
WATER TOWER
CAR PARK
CAR PARK
SECURITY FENCE
BATH HOUSE
SECURITY CONTROL
VISITORS CAR PARK
STAFF CARPARK
STORES DELIVERY
STORE
33 KV SWITCHYARD
O/H CRANE
WORKSHOP
COVERED MUSTERING AREA
STORAGE COMPOUND
LIGHT VEHICLE W/SHOP
PETROL STORAGE
BOWSER
TYRE BAY
LIGHT VEHICLE WASH BAY
WASH BAY
SERVICE BAYS
HEAVY VEHICLE PARKING AREA
LIGHT VEHICLE ACCESS TO MINE
BULK DIESEL STORAGE
HEAVY VEHICLE ACCESS TO INDUSTRIAL AREA
HAUL ROAD
HAUL ROAD
CULVERT
CULVERT
DUMP HOPPER B
DUMP TRUCK TURNING AREA
DUMP HOPPER A
ACCESS TO REJECTS
REJECTS
COAL LAB
CYCLONE PLANT
JIG PLANT
SCREENING STATION
11 KV SUB STATION
THICKENER
ROTARY BREAKER
REJECTS
SCREENING STATION
ROTARY BREAKER
DIRTY COAL
UNWASHED COKING COAL
COKING COAL
STEAMING COAL - LOW ASH
STEAMING COAL
OVERLAND CONVEYOR
PROCESS WATER DAM
250
240
230
220
1 420 500 m N
1 420 250 m N
1 420 000 m N
292 500 m E
292 750 m E
293 000 m E
293 250 m E
293 500 m E
Drainage
Permanent Survey Mark
SCALE 1 : 2000
THIESS-SHELL DRAYTON CO-VENTURE
DRAWING
DATE
DRAWN R BOLDERY 22-6-78
CHECKED
APPROVED
REVISED
PIT HEAD ESTABLISHMENT
FIGURE 15
REVISION
DRAWING No D1/1/300-22 R



PART ELEVATION

OVERLAND CONVEYOR

SECTION A – A

TYPICAL TRESTLE DESIGN

SECTION B – B

SECTION C – C

NOTE:

OVERLAND CONVEYOR CONSTRUCTED FROM 7·5 m. STANDARD MODULES.

FIGURE 16

GENERAL NOTES

D1/1/300-13

SCALE 1:25 1:50 1:250

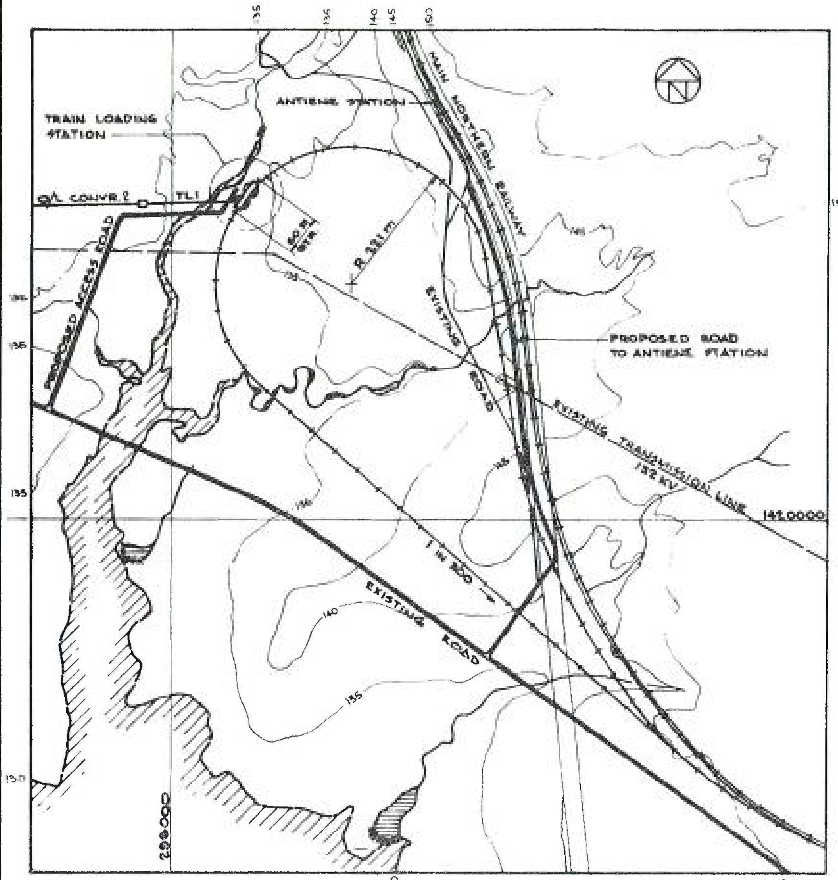
CHECKED A. Rofs 14-10-77

DRAWN P.A.D. 14-10-77

14-10-77

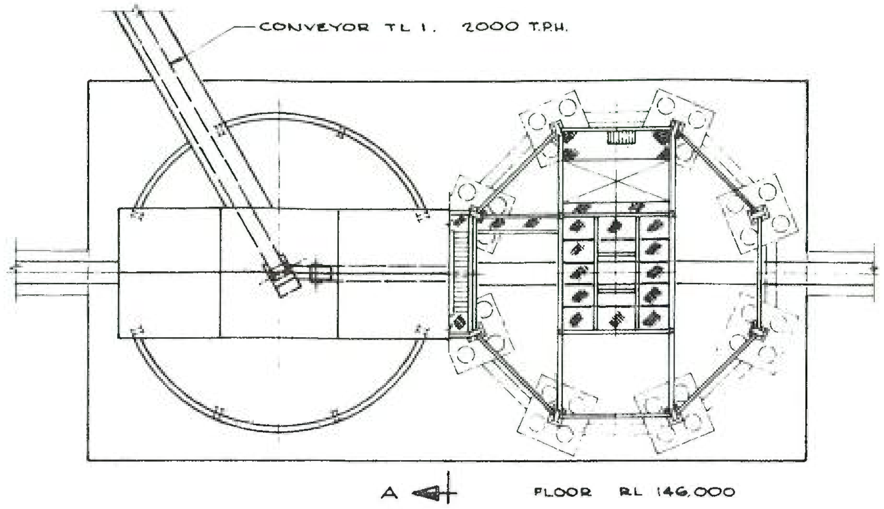
THIESS — SHELL DRAYTON CO-VENTURE
COAL MINE PROJECT
O. L. CONVEYOR, TRESTLES AND BRIDGE

CONTRACT DWG. NUMBER Q201-04

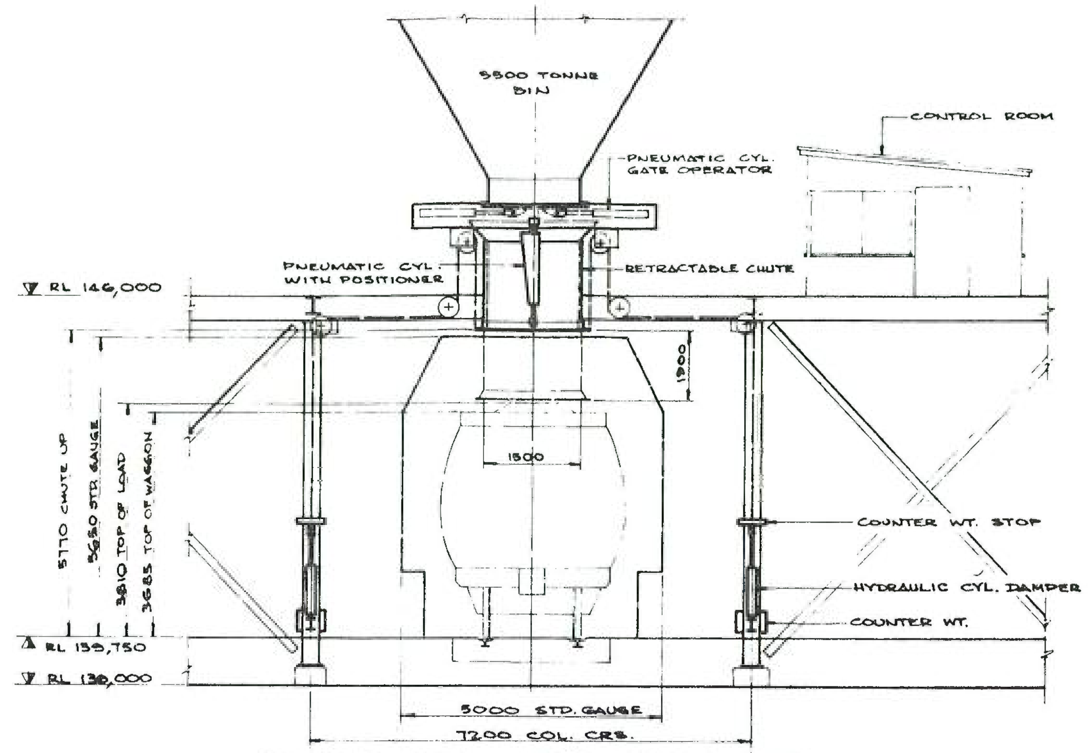


LOCATION PLAN ANTIENE RAIL LOOP

SCALE 1 : 5000

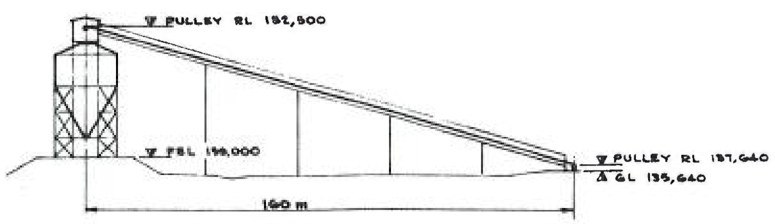


PLAN VIEW



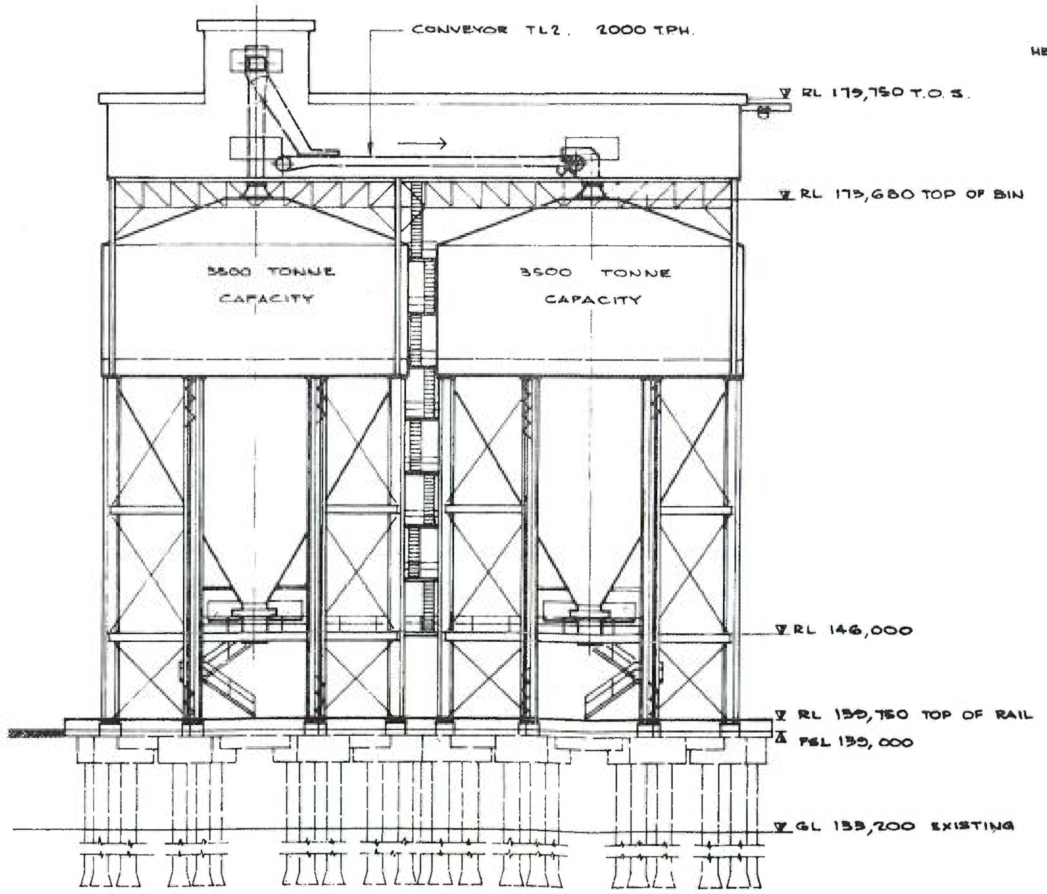
ENLARGED DETAIL AUTO LOADING CHUTE

SCALE 1:50

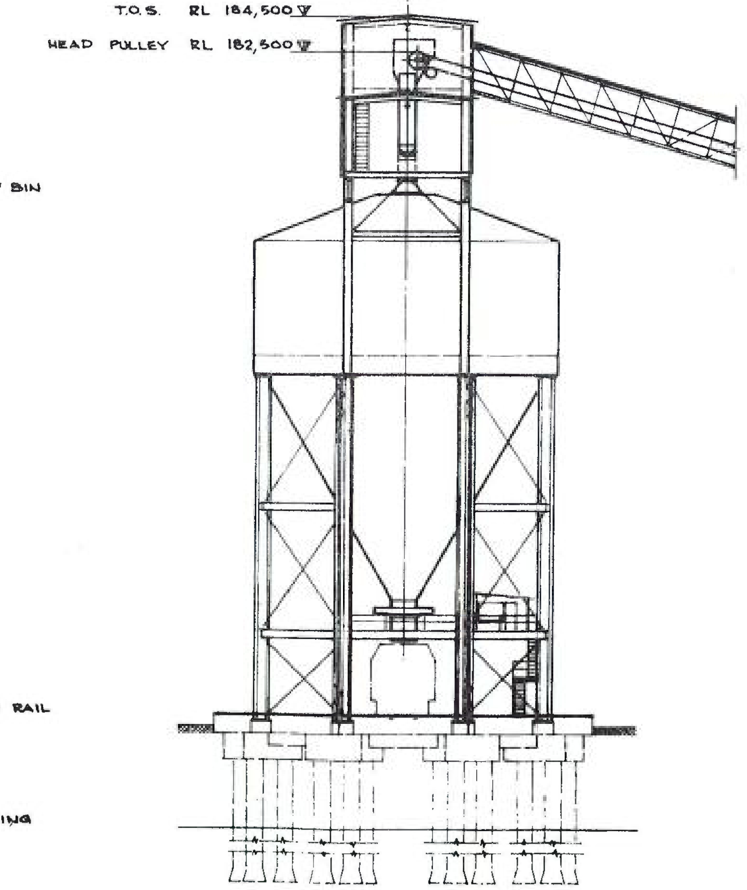


ELEVATION CONVEYOR TL1.

SCALE 1:1000



ELEVATION



SECTION AA

FIGURE 17

GENERAL NOTES

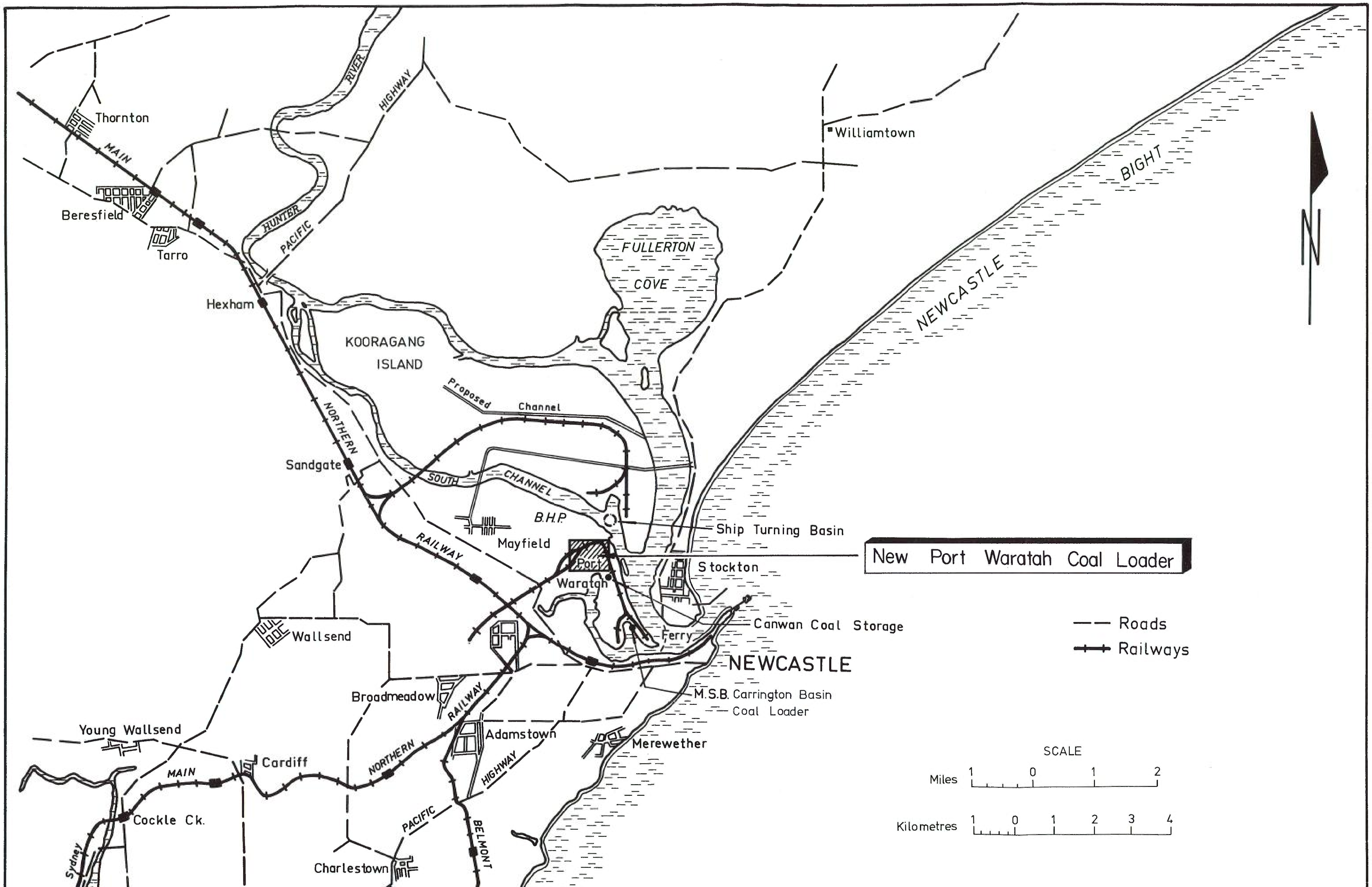
NOTES.
1. ALL LEVELS IN METRES.
2. ALL DIMENSIONS IN MILLIMETRES U.O.N.

D1/1/305-11

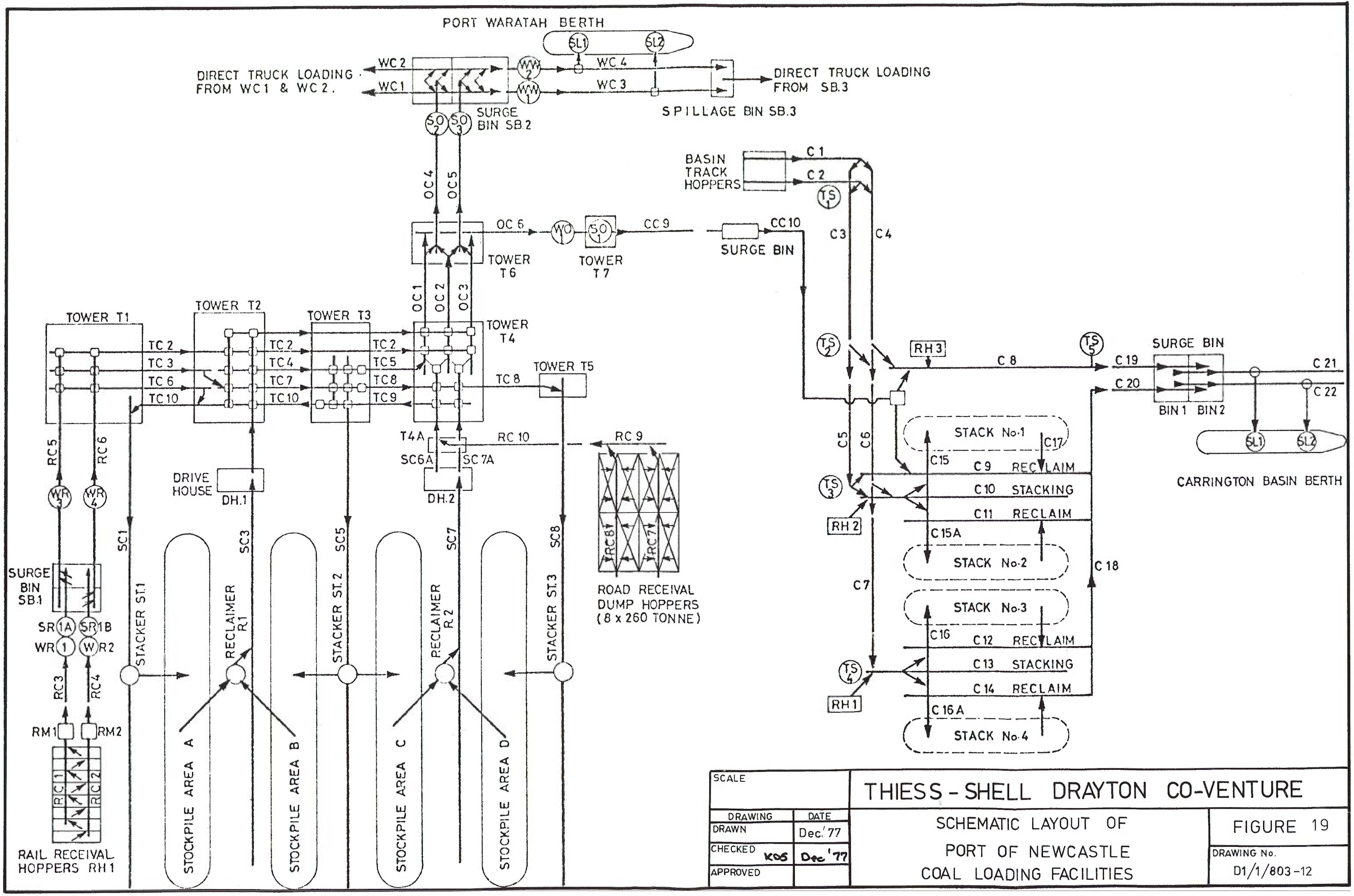
DRAWN	C F M.
SCALE	1 : 200 U.O.N.
CHECKED	A. Roth 14-4-77
DESIGN	P.A.D. 14-4-77
PROJECT	14-4-77
CLIENT	

THIESS-SHELL DRAYTON CO-VENTURE
COAL MINE PROJECT
TRAIN LOADING STATION

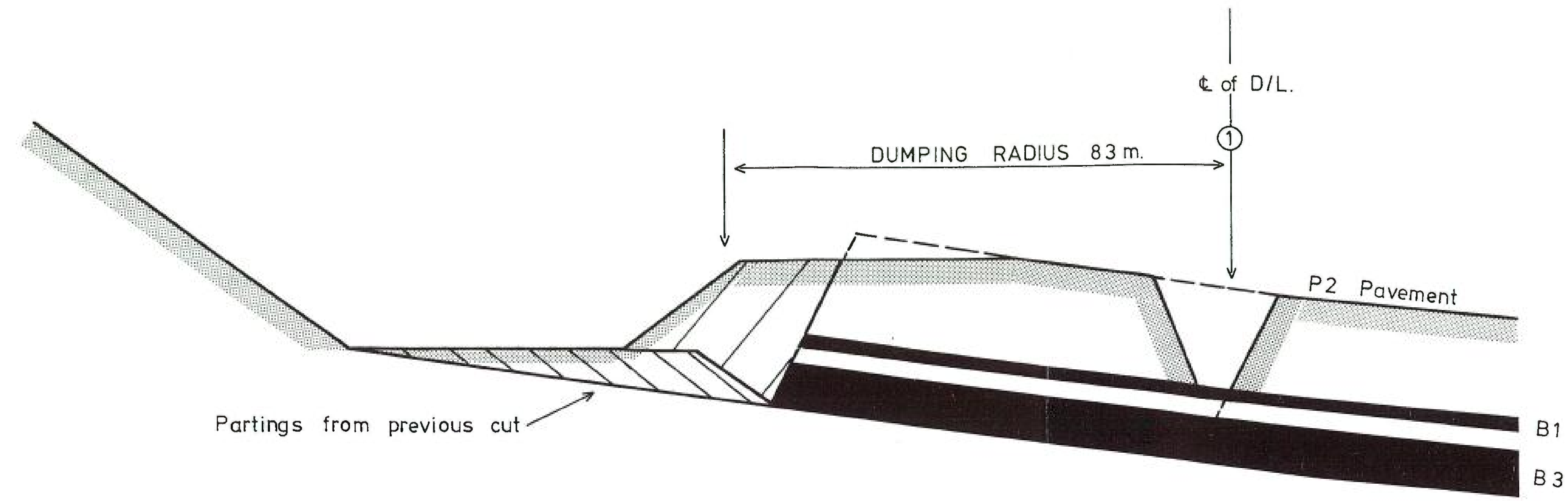
CONTRACT DWG NUMBER: Q 201-05



SCALE		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	NEWCASTLE HARBOUR LOCATION PLAN SHOWING COAL STORAGE AND PORT FACILITIES	FIGURE 18
DRAWN Revised	Dec. '77		DRAWING No. D1/1/803-6
CHECKED KDS	Dec '77		
APPROVED			



SCALE		THIESS - SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	SCHEMATIC LAYOUT OF PORT OF NEWCASTLE COAL LOADING FACILITIES	FIGURE 19
DRAWN	Dec.'77		DRAWING No. D1/1/803-12
CHECKED KOS	Dec '77		
APPROVED			



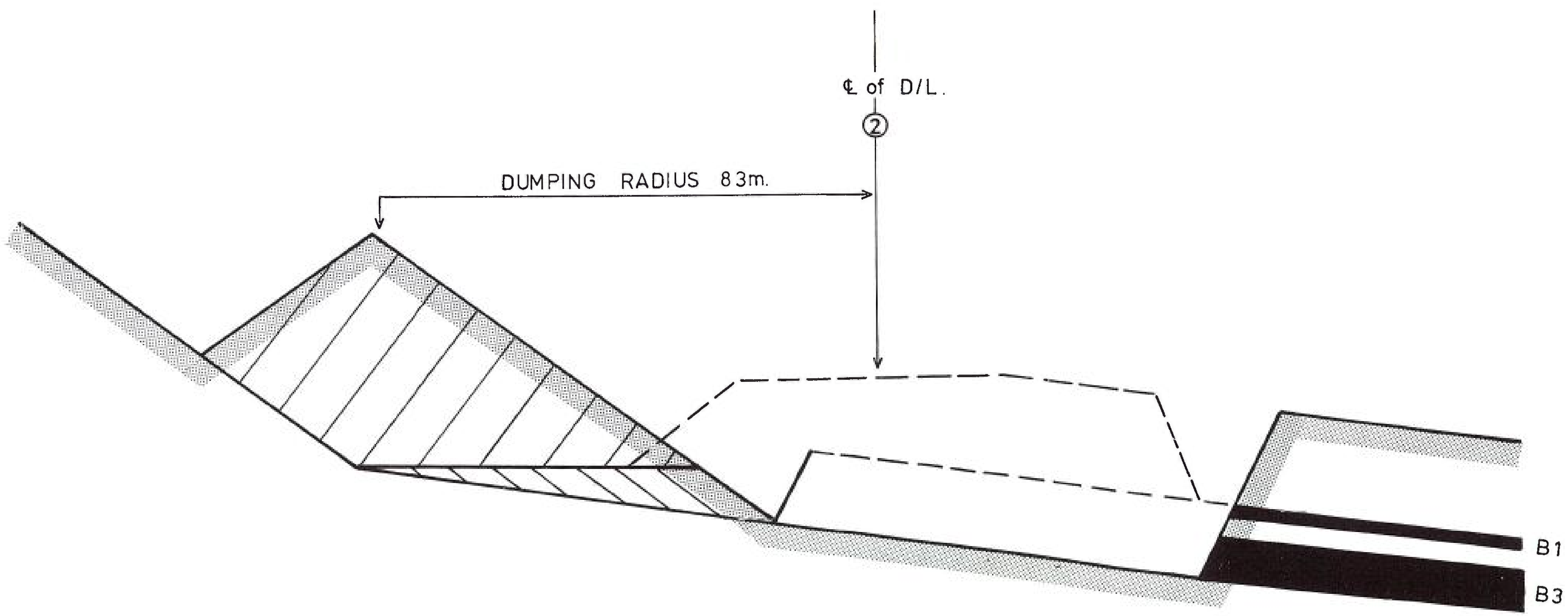
STAGE 1

(a) Remove overburden above P2 by shovel & truck.

(b) Drill and blast overburden P2 to B1.

(c) Level blasted overburden with dozer.

(d) Dig key cut and cast to form extended working bench for dragline.



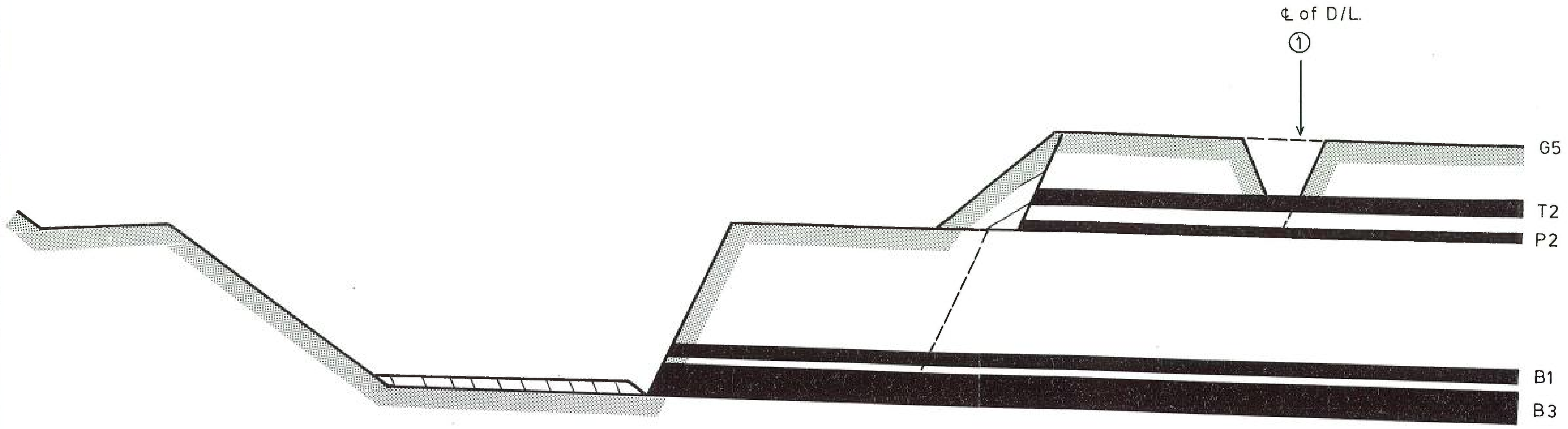
STAGE 2

(e) Move dragline to position 2, cross drag from key cut and cast to form final spoil profile.

(f) Remove Balmoral coal and partings by shovel & truck.

Highwall Angle	63°
Spoil Angle	36°
Swell Factor	30%
Cut Width	70 m.
Rehandle	22%

SCALE 1:1000		THIESS - SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE TYPICAL SPOIL DIAGRAM EAST PIT - CUTS 1E to 5E	REVISION FIGURE 20A
DRAWN C.C.	NOV '77		DRAWING No. D1/1/502-6
CHECKED KOS	Dec '77		
APPROVED			



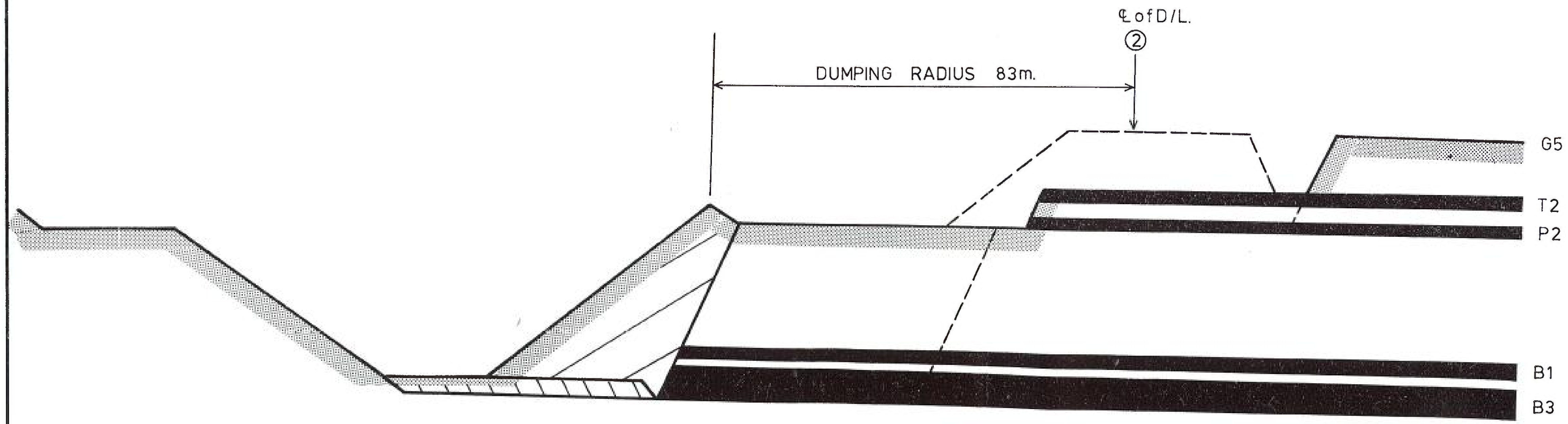
STAGE 1

(a) Remove overburden above G5 by shovel & truck.

(b) Drill and blast overburden G5 to T2.

(c) Drill and blast overburden P2 to B1.

(d) Dragline in position 1 dig key cut and dump spoil over edge of upper highwall.



STAGE 2

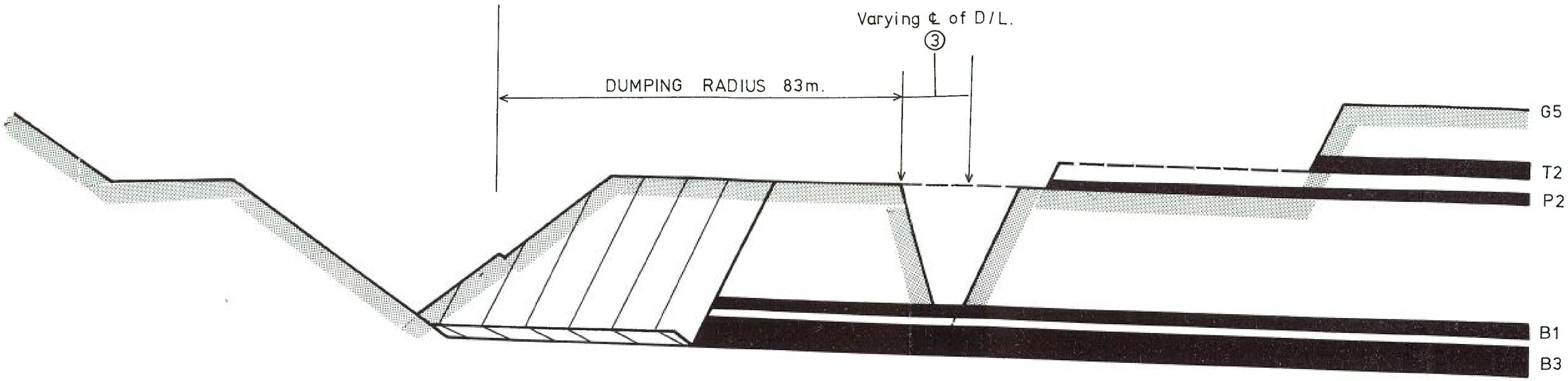
(e) Move dragline to position 2 dig remaining overburden and rehandle and dump spoil over lower highwall.

(f) Remove T2 coal.

(g) Drill and blast T2 to P2 parting.

Highwall Angle	63°
Spoil Angle	36°
Swell Factor	30%
Cut Width	50 m

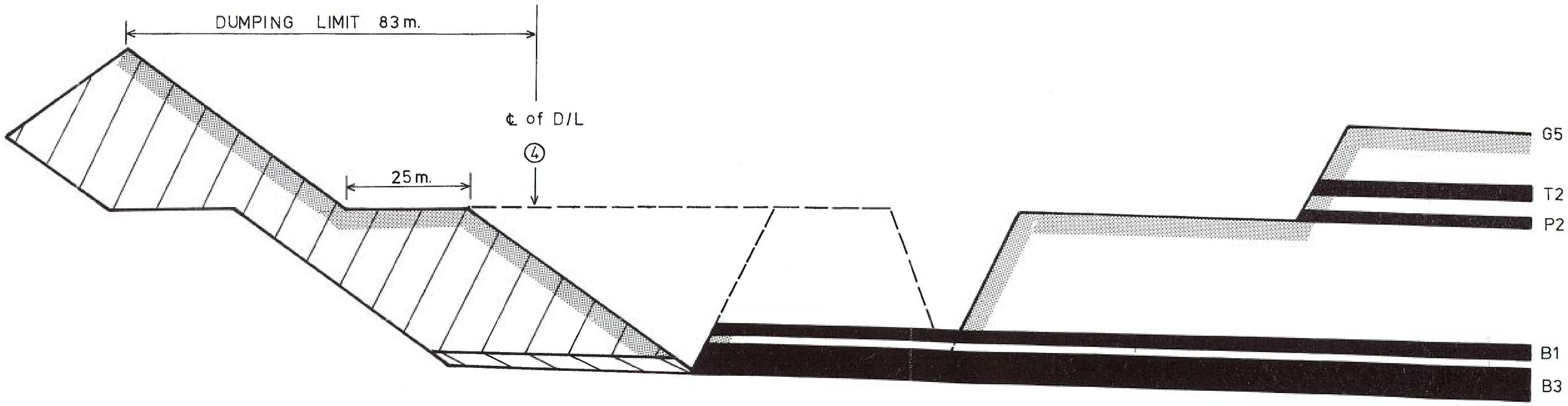
SCALE 1:1000		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE TYPICAL SPOIL DIAGRAM EAST PIT — NORTH CUTS 6E to18E	REVISION FIGURE 20B
DRAWN C.C.	NOV. '77		DRAWING No. D1/1/502-7
CHECKED KDS	Dec '77		
APPROVED			



STAGE 3

(h) Chop parting T2 to P2 and cast to extend dragline bridge.

(i) Dig key cut from P2 to B1 and cast spoil to extend bridge out from highwall.



STAGE 4

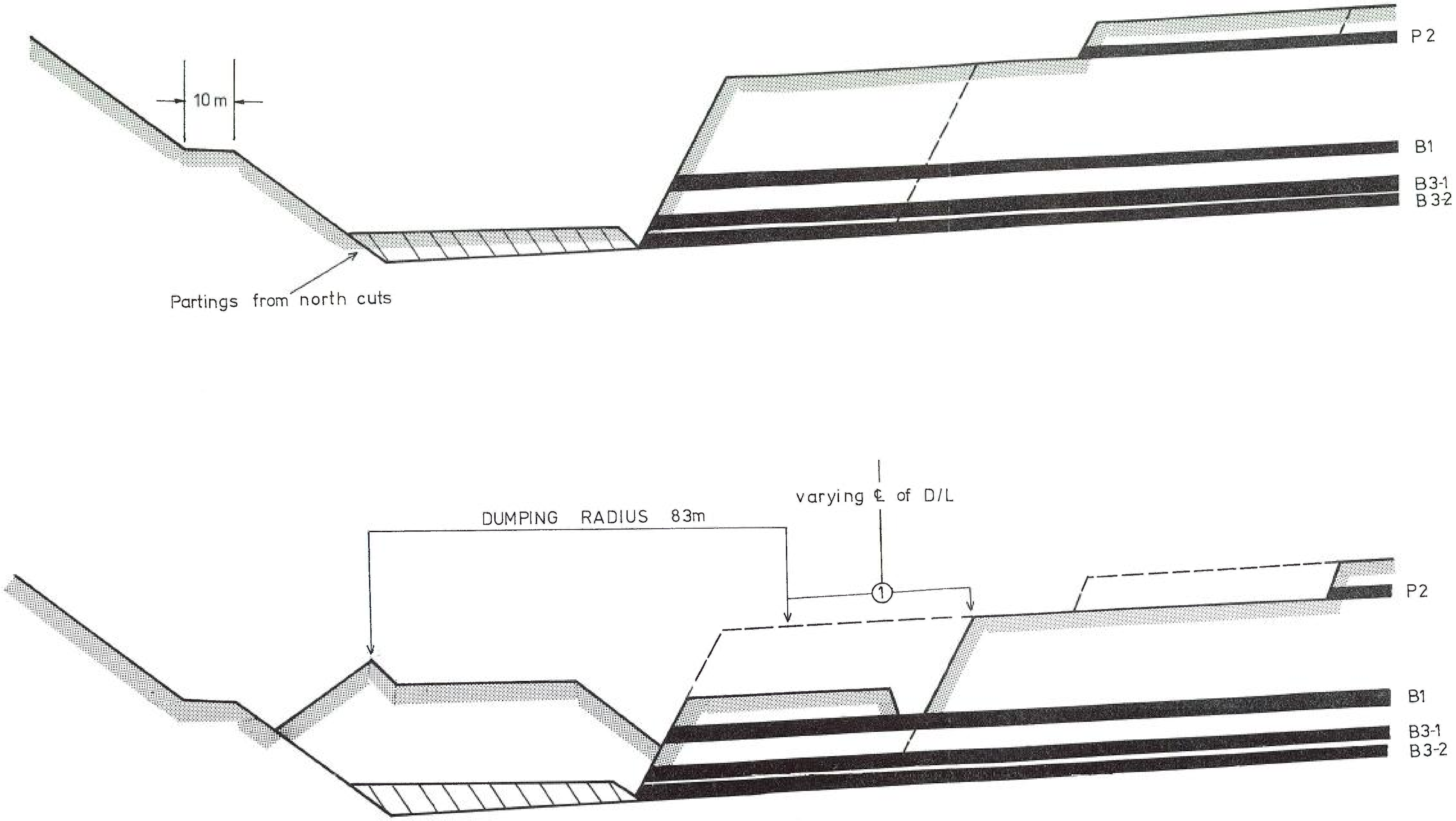
(j) Extend dragline bridge with spoil from P2 to B1.

(k) Move dragline to position 4, crossdrag from key cut and cast to form final spoil profile.

(l) Remove P2 coal and Balmoral coal and partings by shovel & truck.

Highwall Angle	63°
Spoil Angle	36°
Swell Factor	30%
Cut Width	50 m.
Rehandle	46%

SCALE 1:1000		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE TYPICAL SPOIL DIAGRAM EAST PIT - NORTH CUTS 6E to 18E	REVISION FIGURE 20C
DRAWN C.C.	DEC.'77		DRAWING No. D1/1/502-8
CHECKED KDS	Dec '77		
APPROVED			



STAGE 1

(a) Drill and blast overburden to P2.

(b) Drill and blast overburden to B1.

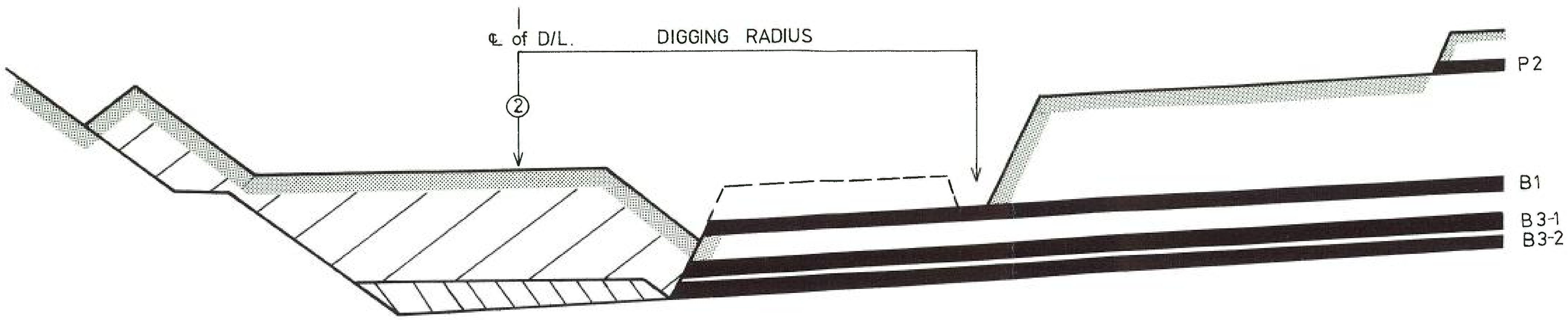
STAGE 2

(c) Chop and cast overburden to P2.

(d) Dig and cast part of overburden to B1 to form bench on dragline spoil.

Highwall Angle	63°
Spoil Angle	36°
Swell Factor	30%
Cut Width	50 m.

SCALE 1:1000		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE TYPICAL SPOIL DIAGRAM WEST PIT - SOUTH CUTS	REVISION FIGURE 21 A
DRAWN C.C.	NOV. '77		
CHECKED KDS	Dec '77		DRAWING No. D1/1/502-14
APPROVED			



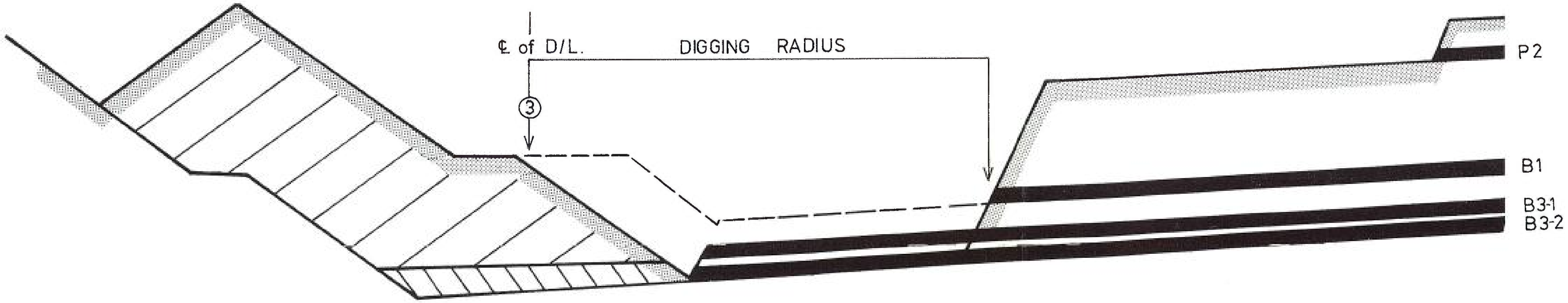
STAGE 3

(e) Deadhead Dragline along spoil bench to central access Ramp.

(f) Cross-drag and cast remaining overburden exposing B1 coal.

(g) Remove B1 coal by shovel & truck.

(h) Drill and blast B1 to B3 1 Partings.



STAGE 4

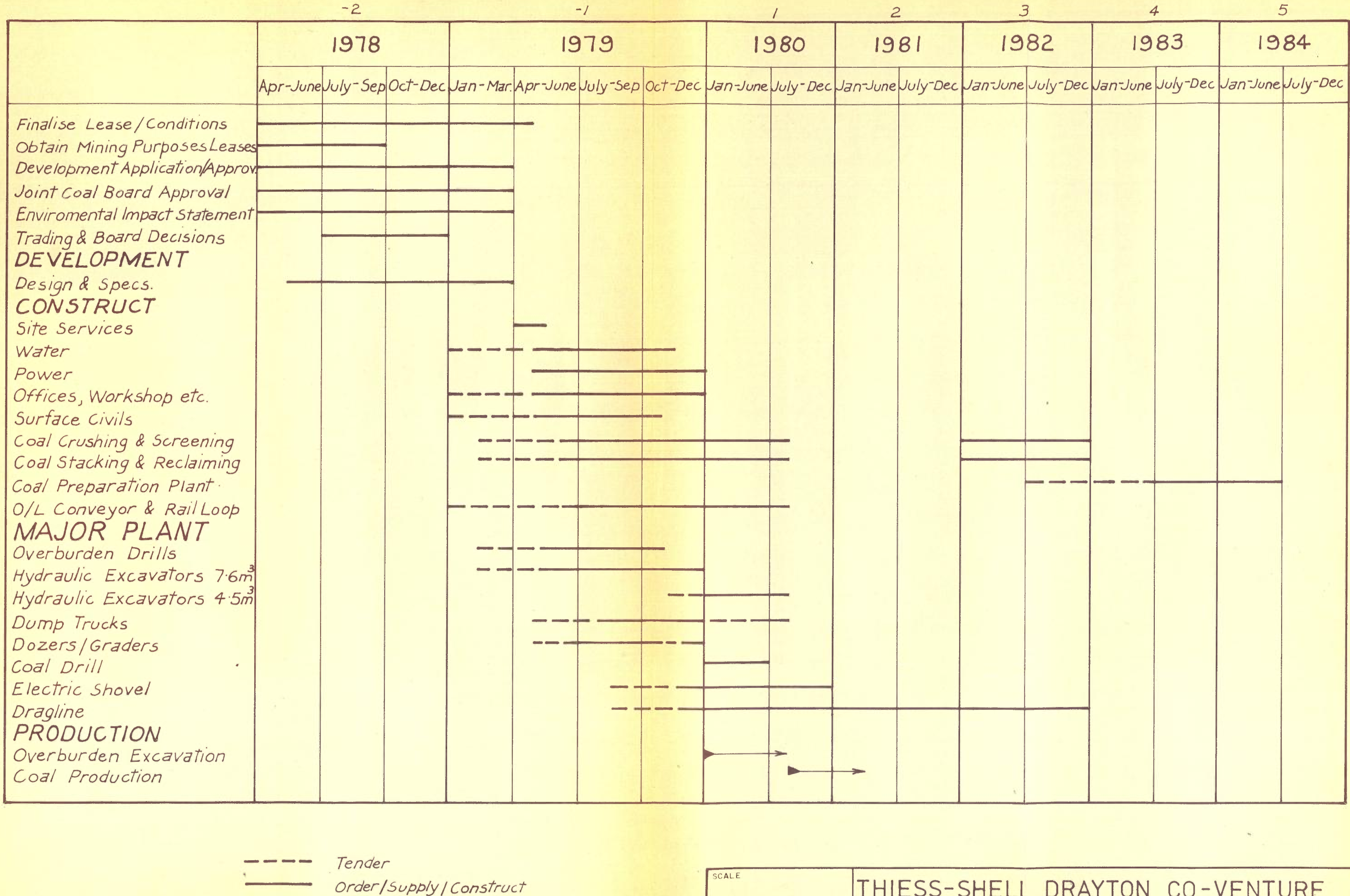
(i) Deadhead Dragline along spoil bench to central access Ramp.

(j) Chop and cast parting above B3-1. Dig and cast, rehandle spoil to form final spoil profile.

(k) Remove remaining coal and partings by shovel & truck.

Highwall Angle	63°
Spoil Angle	36°
Swell Factor	30%
Cut Width	50 m.
Rehandle	19%

SCALE 1:1000		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DRAYTON COAL MINE TYPICAL SPOIL DIAGRAM WEST PIT - SOUTH CUTS	REVISION FIGURE 21B
DRAWN C.C.	NOV. '77		DRAWING No. D1/1/502-15
CHECKED KDS	Dec '77		
APPROVED			



SCALE		THIESS-SHELL DRAYTON CO-VENTURE	
DRAWING	DATE	DEVELOPMENT SCHEDULE SUMMARY	FIG. 22
DRAWN R B	11/7/78		DRAWING No D1/1/101-15
CHECKED			
APPROVED			

Borrower's name	Date	Ext

RAECO