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THE MINING OF THICK, FLAT COAL SEAMS BY A LONGWALL BOTTOM SLICE WITH CAVING AND DRAWING

K. BARRON

MINING RESEARCH CENTRE

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The Mining of Thick, Flat Coal Seams by a Longwall Bottom Slice With Caving and Drawing

by

K. Barron*

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ABSTRACT

A method, developed in France, for mining thick, flat lying coal seams is described in detail. A longwall face is retreated along the footwall of the thick seam and is supported by powered supports which incorporate a special banana prop at the rear. The coal overlying the face caves behind it and is drawn through the banana props onto a rear face conveyor. Face advance is achieved either by hand mining or by using a conventional double drum shearer. Details of the mining method, problems encountered and good and bad mining practice are discussed. The capital and operating costs, the production and productivity in France are given. Ground control and environmental control problems are also discussed.

Consideration is then given to the potential use of this method for mining thick, flat lying, coking coal seams in Western Canada. It is shown that the operating costs depend greatly on the seam thickness and that with current metallurgical coal prices the method, at best, would not be profitable in seams less than 28 ft thick in which a panel operating cost of $5.50/short ton raw coal might be achieved. In these circumstances, daily production from a 100 m face would be 2750 short tons/raw coal at a panel productivity of 13.45 short tons/man shift and a face productivity of 29.5 short tons/man shift.

The method is confined to seams that dip at less than 20° and the face must be operated on retreat and down dip. Serious ventilation problems are envisaged in very gassy coal seams. The very friable nature of Western Canadian coal seams in the Rocky mountains could lead to serious problems in supporting the face and roof immediately ahead of the powered supports; caving in this region could pose almost insuperable problems. It is therefore recommended that an experimental face must be operated using this method before a decision as to the viability of the method in Canada can be taken.

Key words: coal mining: thick seams: flat seams: longwall: caving and drawing: production: productivity: costs: ground control: environmental control:

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CHAPTER 1. INTRODUCTION

1.1 Background

The assessment of the coal reserves of Western Canada is of prime importance to the formulation of a national energy inventory. Currently the Mines Branch, Mining Research Centre is contributing to this program by studies aimed at deriving mineability criteria for the plains coals of Saskatchewan and Alberta so that the economically recoverable portion of the estimated geologic reserves may be determined with some degree of accuracy.

The next step is to similarly assess the economically recoverable reserves of the "Inner Foothills Belt" of Alberta and British Columbia. (This belt contains over 70% of the estimated geologic coal reserves of Western Canada.) However, this assessment poses problems of a higher order of difficulty; the region has been severely geologically distorted and as a result the major portion of this coal is locked up in thick and/or inclined coal seams which are, for the most part, only accessible by underground mining methods. Mining conditions in these seams are exceedingly difficult and it is chastening to realize that current underground mining methods are confined to only a small fraction of relatively flat seams; most of the coal is technically and/or economically unmineable today. Two major problems are thus apparent; firstly, to assess the economically recoverable coal reserves, it will be necessary to establish realistic mineability criteria, taking into account the geologic, engineering and economic constraints. However, in these complex geologic conditions such mineability criteria must inevitably be dependent on the mining method. Unfortunately proven mechanized methods for the underground excavation of thick and inclined coal seams in Western Canadian conditions do not exist; thus, secondly, mining technology must be developed or adapted before this coal can be considered as recoverable.

A logical step in considering this necessary development of underground mining technology is to review carefully mechanized mining techniques carried out elsewhere in the world and to carefully select those which offer the potential for use in or adaptation to Canadian conditions for further detailed study. This review was carried out and it became evident that mining developments in France offered considerable potential for further study.

1.2 Objectives

The Departmental objectives are "to ensure the effective use of mineral and energy resources available in Canada for the present and future benefit of the nation by ascertaining the resource potential and improving the means of ----mining ----these resources". Within these objectives the Departmental sub-objectives include "the development of mining technology that is important for exploiting Canadian resources".

Within this framework of Departmental objectives the prime objective of this project was to study technologies that may be applicable to the mining of coal seams in Western Canada that are not being mined at the present moment; e.g. inclined seams with dips between 25° and 45°, steeply dipping seams from 45° to 90° and thick seams with a thickness of greater than about 10 ft. A second objective was to gather data that would assist in the assessment and development of mineability criteria for such coal seams.

To achieve these objectives a team of three engineers was sent to France to study in detail the technology developed for mining in such seams and to assess the adaptability of both the technology and the economics to the Western Canadian scene. Each of these engineers was assigned to study one or more methods in different coal basins in France.

The author was assigned to study the mining methods for very thick, flat lying, coal seams carried out in the Blanzy coal basin. The mining method used here is basically a retreating longwall bottom slice, coupled with caving and drawing of the overlying coal. The main objectives of the study of this specific method were to determine the details of the mining method, it's advantages and limitations, the problems that have been encountered and how these have or have not been overcome. Likewise full details of ground control, environmental control, production and productivity, capital and operating costs were to be obtained and, where possible, related to the potential applicability in Western Canada. It is, of course, recognized that is it impossible to transfer directly a mining method used elsewhere to conditions prevalent in Canada; nonetheless it is believed there are many aspects of this technology that might be successfully modified or adapted to the Canadian scene. Detailed studies of the above nature should certainly enable a much better assessment to be made of the probabilities of both technical and economic success of such methods in Canada.

To achieve the above objectives the author spent a period of 10 weeks in France; the first seven weeks were spent studying the mining operations in the Blanzy coal fields; one week was spent discussing ground control and environmental control problems with research engineers of Cerchar, one week was spent visiting selected equipment manufacturers and one week was spent at Carmaux where another thick seam mining method was briefly examined (1).

This report summarizes the results of these studies in the Blanzy coal fields. The layout of this report is as follows: Chapter 2 gives a general background of the geology, organization, mine location and layout together with a brief history of mining development in the Blanzy area. Chapter 3 gives the results of detailed studies carried out on three operating faces in the Darcy mine. Chapter 4 does the same for the Rozelay mine. Chapter 5 considers aspects of ground control and Chapter 6 considers aspects of environmental control. Finally, in Chapter 7, the probable production, productivity and costs in Canadian conditions are assessed as are the engineering, geologic and environmental constraints on the potential Canadian application of this method. To avoid overloading the text with specific details, and to preserve continuity, such details have been assigned to the accompanying appendices.

CHAPTER 2. A GENERAL BACKGROUND OF THE BLANZY COLLIERIES

2.1 Location and General Geology

The Blanzy Collieries comprise the two working areas, Decize and Blanzy, about 100 km apart. Decize is in the Department of Nievre close to the town of La Machine; this area contributes about 12% of the coal production. Blanzy, the major producing area (88%) for the collieries, is in the Department of Saone and Loire close to the town of Montceau-les-mines.

Figure 2.1 illustrates the general geology of the region. The coal beds of Blanzy, Le Creusot and Bert are distributed around a large Permian basin in the N.E. of the Massif Central. The Permo-Carboniferous formation rests on granites, gneiss and ancient rocks; it is bordered to the north by the massifs of Morvan and St. Leon and to the south by that of Charolles. The basin is elongated in the SW-NE direction and has a length of about 100 km with a width varying from 4 km at the ends to 14 km in the central part (2).

The most important group of coal beds in this basin are those in the long, narrow, continuous belt which outcrop on the SE border of the basin and which constitute the Blanzy zone. Figure 2.2 shows a generalized stratigraphic section of the Blanzy zone. A total of seven coal seams, of varying thickness, are present in the zone.

Figure 2.3 shows a more detailed plan and sections of the Blanzy zone in the vicinity of the current mining activity. Major faults in the region are shown on this figure as are the regions of previous extraction. At present only two underground mines are operating in this region. The Darcy mine exploits the No. 4 seam, of mean thickness 12 m, at a mean depth of 1000 m. The Rozelay mine exploits the No. 2 seam at an average depth of 320 m; it also has a mean thickness of 12 m but this includes two bands of hard sandstone of 1.5 and 0.5 m mean thickness, respectively.

2.2 Organization

Charbonnages-de-France is administratively split into three main coal producing regions:- Lorraine, Nord Pas-de-Calais and Centre-Midi. Blanzy collieries are one of the seven colliery groups within the Centre-Midi region.

Appendix 1 gives the organization chart for the Blanzy collieries. Whilst this chart is self-explanatory the following points should be noted. The mining operations of the collieries are split into three main operations, Decize, Darcy and Rozelay mines plus a small open pit operation. Each of these mining operations has its own engineering staff and can function as relatively independent producing units. However, selection and purchasing of mine equipment is not the responsibility of these units; a division of "underground studies and equipment" (B.E.F.) carries out this function for all the mines in the Blanzy collieries. Each individual operation "rents" each item of equipment in use from B.E.F. A daily rental fee for each equipment item is established by B.E.F. and charged to the individual operations; this daily rental fee includes all amortization, interest, repair and maintenance

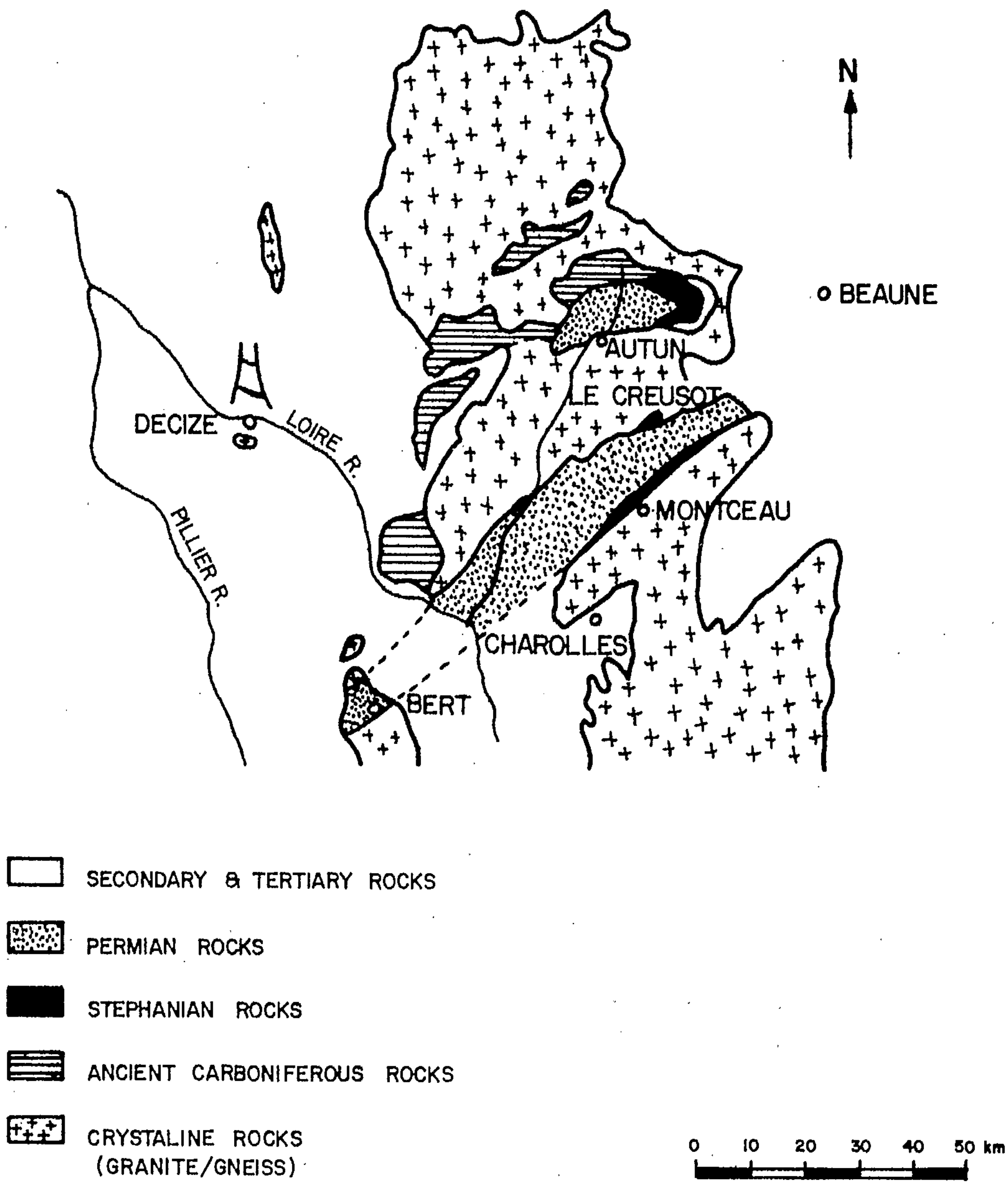

FIGURE 2.1 : LOCATION and GENERAL GEOLOGY of BLANZY COAL BASIN

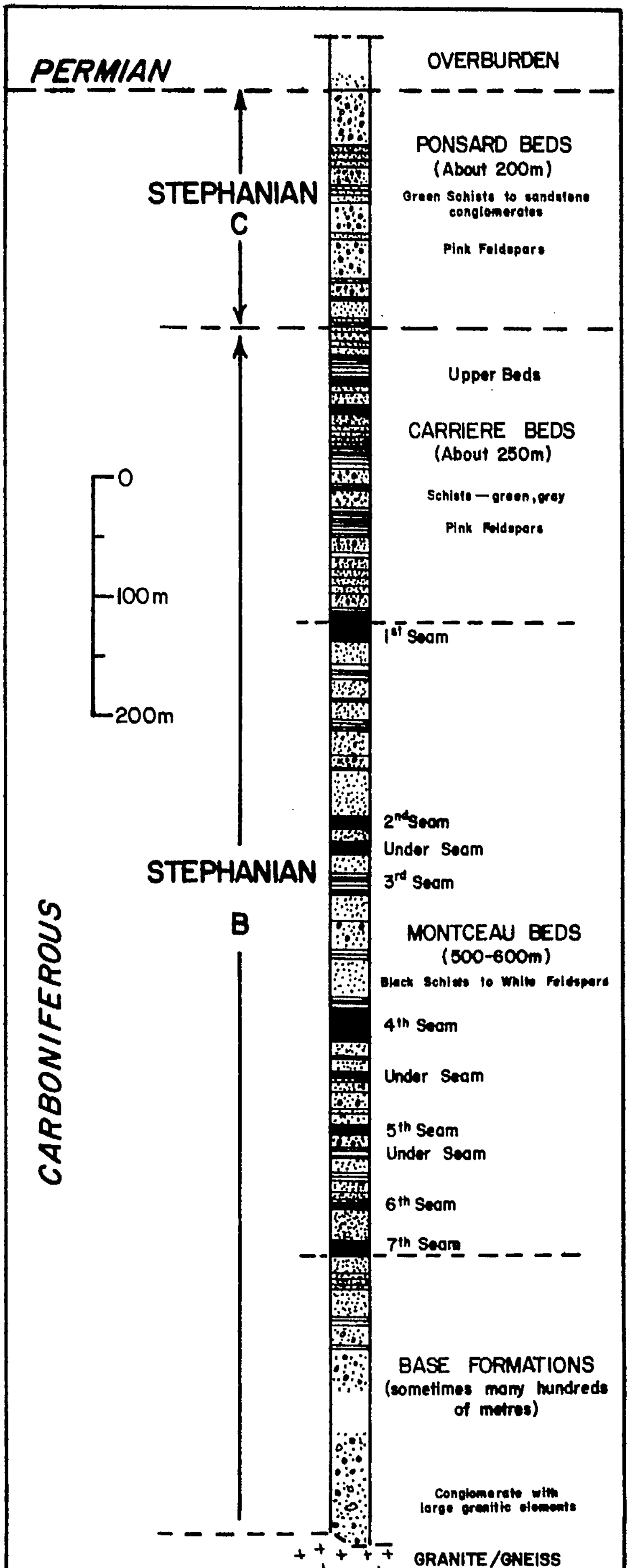

FIGURE 2.2: TYPICAL STRATIGRAPHIC SECTION THROUGH BLANZY COAL MINES

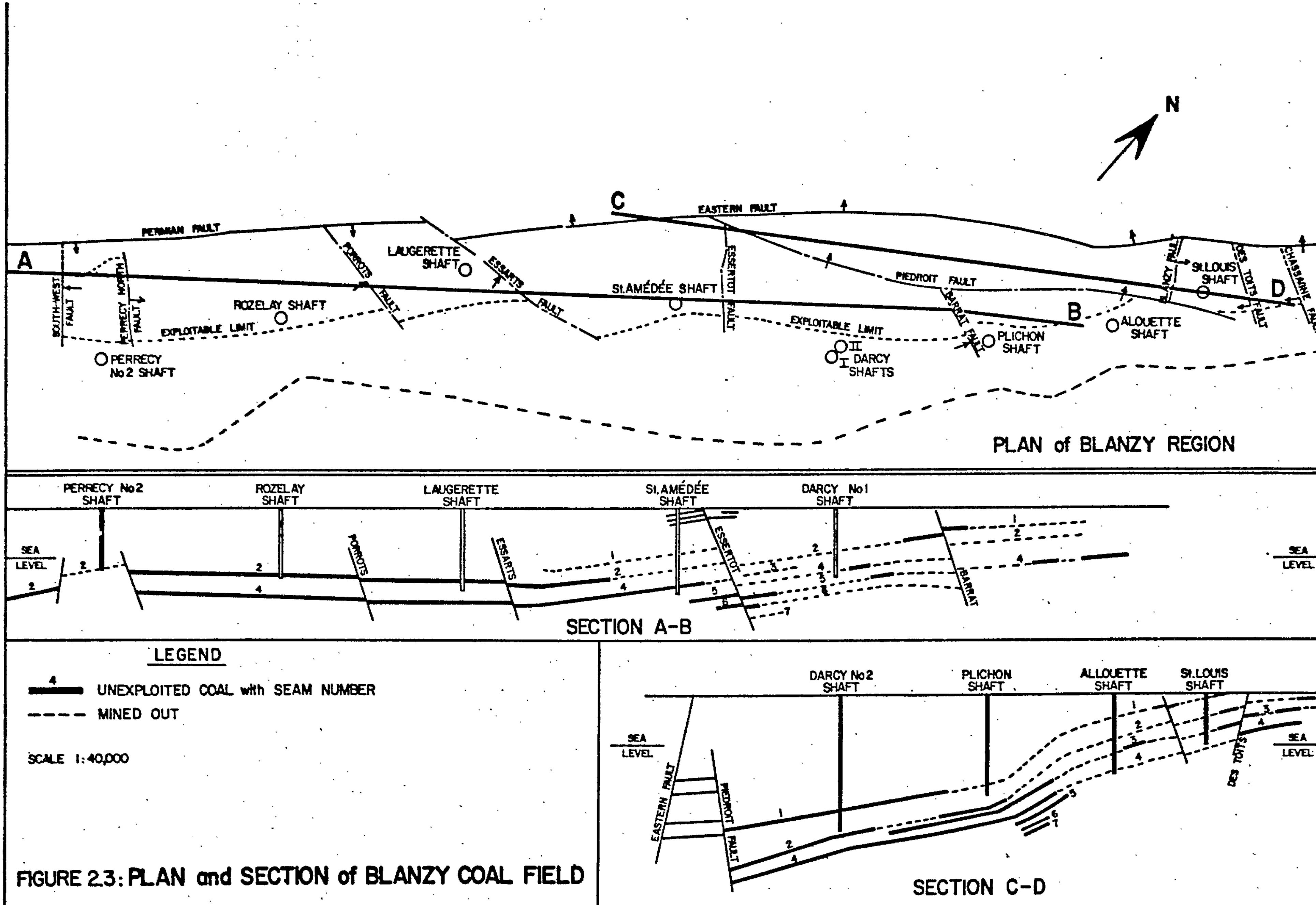

FIGURE 2.3: PLAN and SECTION of BLANZY COAL FIELD

costs for the equipment. This factor should be borne in mind when considering the presentation of cost figures given later in this report.

2.3 Coal Quality and Use

Table 2.1 below gives an approximate distribution of the quality and distribution of the coal produced by the Blanzy collieries:

TABLE 2.1

Coal Quality and Percentage Production from the the Blanzy Collieries

Coal Type	Percentage Volatile Matter	Percentage Production
Flame coal	35-40%	} 38%
Gas coal	28-33%	
Semi-lean (mi-gras)	16-23%	40%
Lean coal and anthracite	9-14%	21%

The main use of the coal is in the generation of electricity. Two local 40 MW power stations produce power for the mine and supply 57 MW to the national grid. These use about 150,000 tons* per year. A large 240 MW power station, on site, consumes an additional 600,000 tons/year.

In addition 150,000-200,000 tons/year are shipped to Chalon-sur-Saone to supply two 125 MW power stations.

About 350,000 tons/year (half in briquette form) are sold for domestic furnaces and about 150-200,000 tons/year are sold to various industrial consumers.

2.4 Production, Productivity and Cost

Figure 2.4 shows production, productivity and labour force data for the Blanzy collieries over a period of years. Both production and the labour force have been steadily dropping in line with a policy of gradually phasing out operations of the Blanzy collieries. The Decize operations will cease in 1974 and those of Blanzy between 1980 and 1985. A small but steady increase in productivity has been maintained over this period, reflecting mainly the influence of mechanization. Table 2.2 below lists the current statistics.

2.5 Evolution of the Mining Methods

The main method of coal extraction used in this region over many years was the so called "Blanzy" method. This comprised mining of coal by ascending horizontal slices, with hand backfilling into descending sub levels.

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* metric tons are used throughout this report unless otherwise stated (1 metric ton = 2,200 lb).

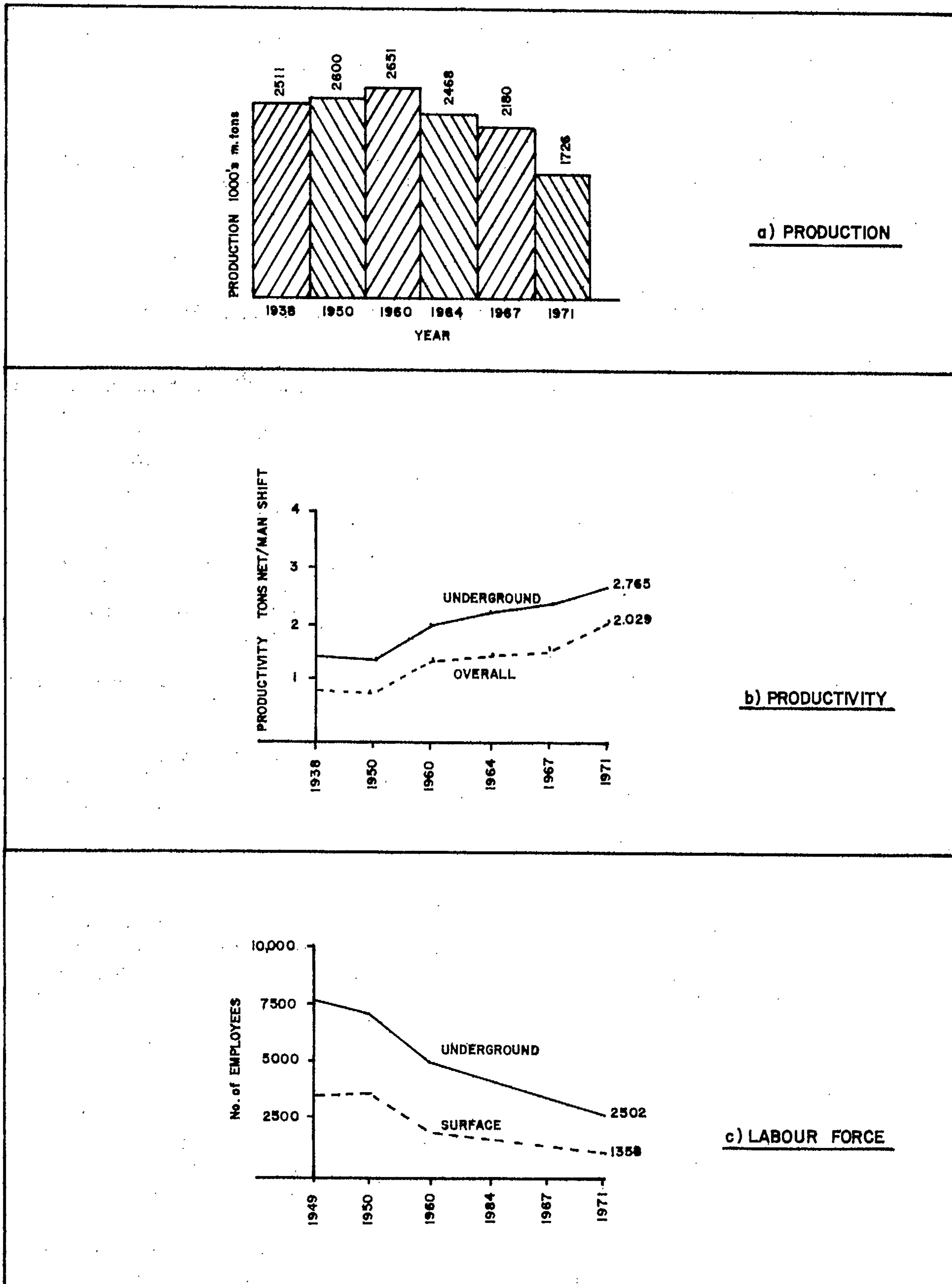

FIGURE 2.4 : PRODUCTION, PRODUCTIVITY and LABOUR FORCE – BLANZY COLLIERIES

TABLE 2.2

Production, Productivity and Costs - Blanzy Collieries 1971/72

Production 1971	Underground 1,471,643 t. net		Open pit 253,741 t. net	Total 1,725,384 t. net
Productivity, 1st 6 months 1972	Underground 3.047 t. net/man shift		Overall 2.216 t. net/man shift	
Producing mine & mean daily production.	Darcy 3200 t. net/day	Rozelay 2000 t. net/day	Decize 800 t. net/day	Open Pit 600 t. net/day
Costs	Production cost 115 F/t. net $23/t. net		Sale price 80 F/t. net $16/t. net	Loss 35 F/t. net $7/t. net

At the end of the last war hand backfilling was no longer economically practical. Pneumatic backfilling or caving was introduced and longwall mining was applied.

A method of horizontal longwall slices, descending from the roof to the footwall was then introduced; this is illustrated in Figure 2.5. Flexible flooring of wood and mesh was laid on the bottom of each slice, and may be used for several slices, to form the "roof" for the succeeding slice. Longwall panels were operated on retreat with the 3 or 4 horizontal slices being worked simultaneously. Panel outputs in the range 1500 - 2000 t. net/day were achieved.

In 1964 the longwall caving and drawing method was introduced for the first time in this area. Because the coal was relatively hard it was initially thought that in thick seams ($>$ 9 m) it would be necessary to mine in two slices as illustrated in Figure 2.6. The first longwall slice was retreated at about 3 m above the footwall to induce the overlying coal to cave; a floor of mesh and wood was laid to form the "roof" for the second slice. The second footwall slice was mined about 30 m behind the first face. The caved coal was drawn through windows cut in the mesh, onto rear conveyors on the second face. At this stage of development face advance was achieved by traditional hand mining methods, the face was supported by both hydraulic and friction props. For coal seams less than 9 m thick the method of using two slices of 3 m thickness was not practical and for seams between 3 and 9 m it was decided to try a single footwall slice, as shown in Figure 2.7, with both caving and drawing being achieved by the one face. The face support system was the same as above, but in this case a mesh was placed over the top of the face supports, sagging to the footwall after passage of the supports, to prevent the caved coal spilling onto the face. As before, the caved coal was drawn, through windows cut in the mesh, onto a conveyor.

Experience with this method indicated that, in the Blanzy conditions, even in very thick seams it was not necessary to use two slices; the single footwall slice was adequate to ensure caving of the overlying coal although, on occasion, it was necessary to induce caving by shot firing. The current mining methods used in Blanzy are merely improved mechanized versions of this longwall, bottom slice, caving and drawing method. Mechanization has been achieved with the introduction of self-advancing hydraulic supports and the use of two conveyors on the face allowing face advance and drawing to proceed simultaneously. The latest innovation has been the introduction of a double drum shearer for face advance. These current systems are described in detail in Chapters 3 and 4.

Appendix 2 gives a more detailed description of the evolution of mining methods in this region.

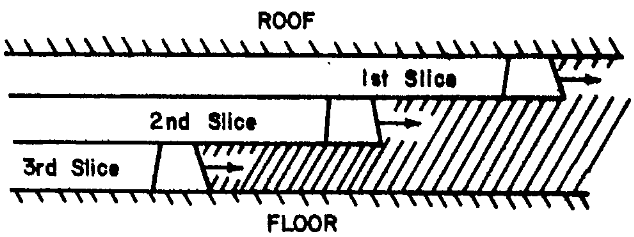

FIGURE 2.5 : METHOD of DESCENDING HORIZONTAL LONGWALL SLICES with CAVING or BACKFILL

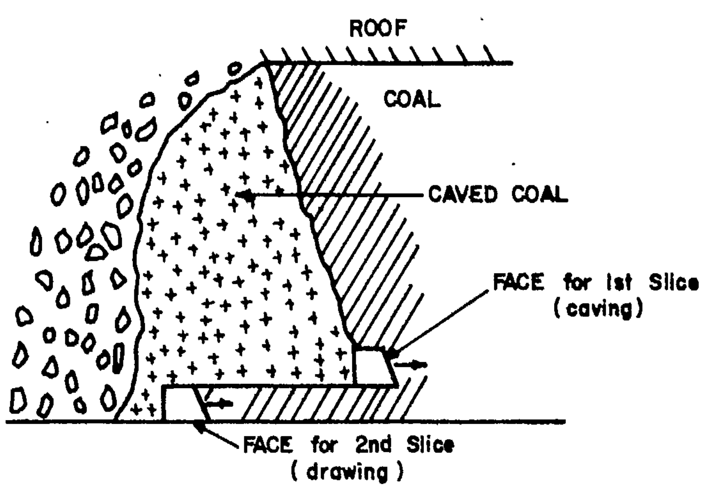

FIGURE 2.6 : TWO LONGWALL SLICES ; ONE for CAVING , ONE for DRAWING

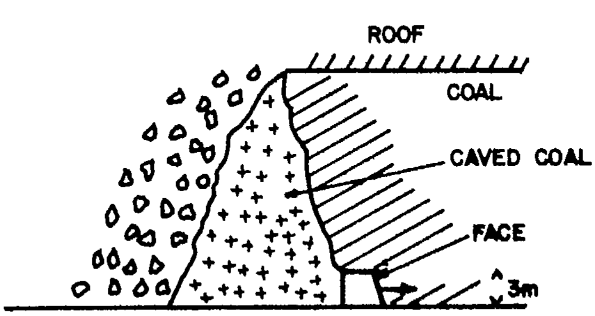

FIGURE 2.7 : SINGLE FOOTWALL SLICE for both CAVING and DRAWING

CHAPTER 3. THE DARCY MINE

3.1 Geology

Figure 3.1 shows a plan of the Darcy mine together with the layout of the mining panels. The contours of the roof of the seam are marked on this plan as are the major faults in this region. Figure 3.2 shows two simplified cross sections (section AA' and BB' on Figure 3.1) through the mine.

It is seen from these figures that the Piedroit fault represents a major down throw of 400 - 500 m and effectively cuts off the previously extracted mining area to the south of the fault from the current mining area to the north of the fault. The current mining area, with a depth of cover of about 500 - 650 m, is again cut off to the north by the Eastern fault, another major down throw of unknown extent. The ends of the elongated current mining area are also faulted off.

In the western end of the mining region the seam contours indicate a relatively flat lying bowl shape for the seam, in the eastern end the seam topography is more irregular. Seam thickness varies from about 6 m to 20 m with an average thickness of 12 m. Figure 3.3 shows a typical stratigraphic section in the vicinity of the No. 4 seam taken from the drill hole marked on Figure 3.1 (in panel E). The immediate roof and floor of the seam are composed of relatively weak schists.

3.2 The Mine Layout

3.2.1 Panel layout

Figure 3.1 shows the past, present and future mining panels, lettered C to W, in the Darcy No. 4 seam. Panels G^a, G^b, I and J have been previously mined out. Panels C, D and T are the current mining panels and were those studied in this mine. The remaining panels are planned for future mining.

All panels have been laid out for retreat mining down dip or down apparent dip. These are very important points to be followed for panel layout with this mining method. The retreat mining minimizes the risks of spontaneous combustion; retreating the face down dip is essential for the maintenance of good face working conditions. Practice has shown that an angle of about $5 - 10^o$ down dip is most suitable with a maximum of about 15^o being practical. A maximum inclination along the face of about 15^o can also be accommodated. This, coupled with the maximum apparent dip in the face retreat direction of 15^o, means that the maximum true dip of the seam that is suitable for this mining method is approximately 20^o. Local variations of greater than this can be overcome but it is safe to say that problems are greatly accentuated with increasing dip angle and that a true dip of greater than 20^o or an apparent dip of greater than 15^o should not be exceeded.

In the eastern end of the mine the irregularity of the seam contours have resulted in fairly extensive barrier pillars being left between different

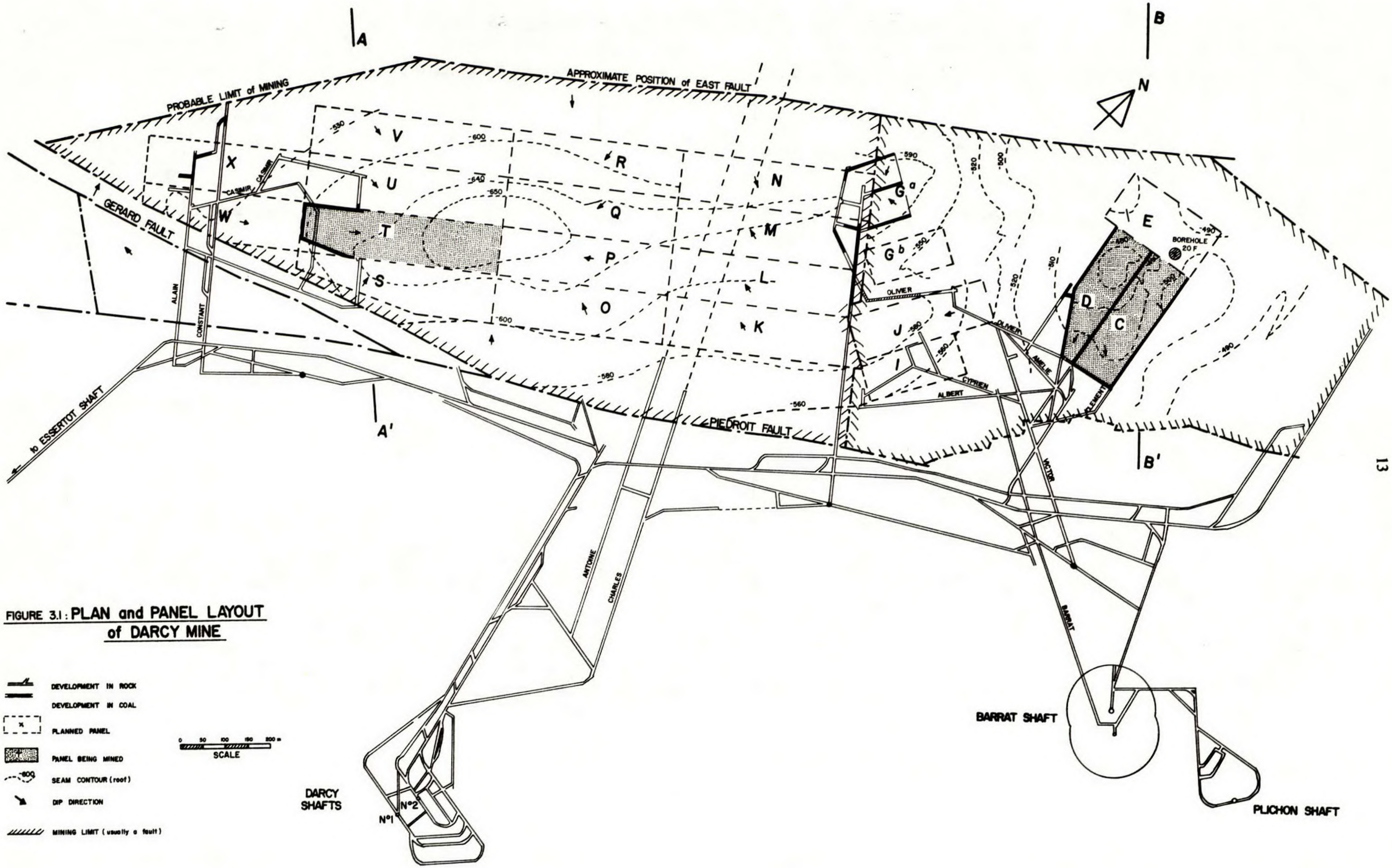

FIGURE 3.1: PLAN and PANEL LAYOUT of DARCY MINE

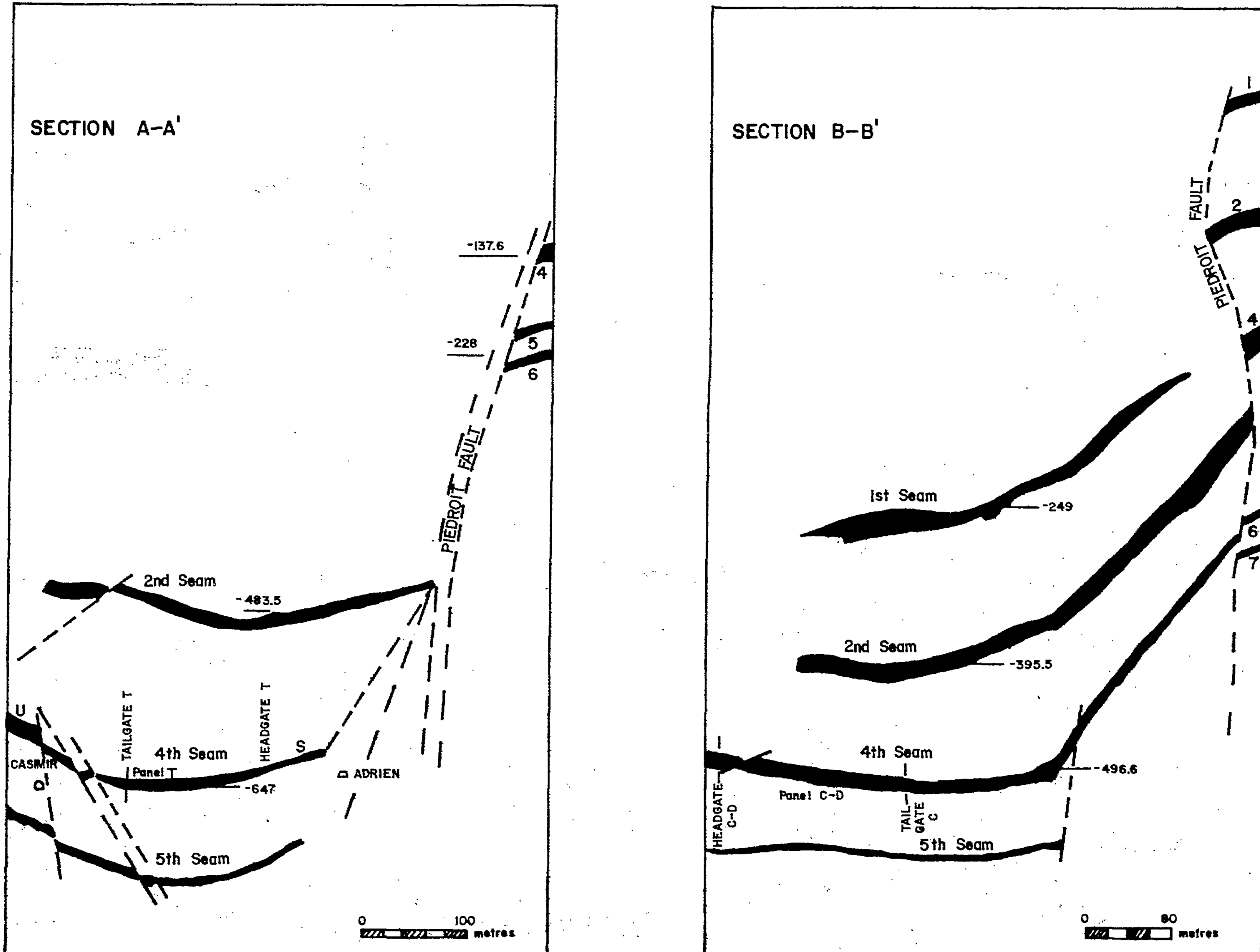

FIGURE 3.2: SEAM CROSS SECTIONS — DARCY MINE

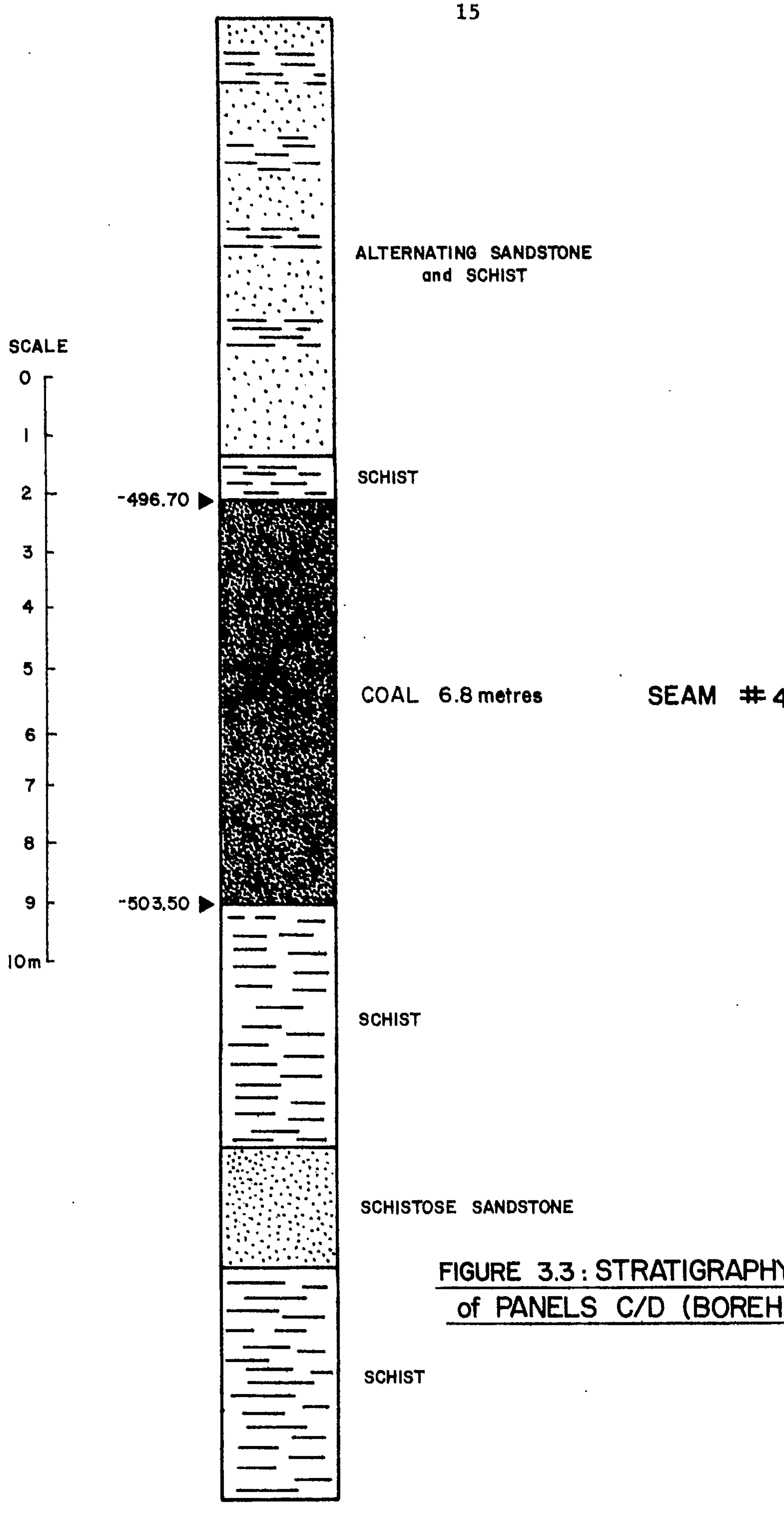

FIGURE 3.3: STRATIGRAPHY in VICINITY of PANELS C/D (BOREHOLE 20F)

panels; this is the result of meeting the above geometric requirements and was not a requirement for ground control considerations. For panels K to W in the western end of the mine, the bowl shape of the seam has allowed the panels to be laid out immediately adjacent to each other with no barrier pillars between them.

Figure 3.4 shows a more detailed plan of panels C and D. These panels are being worked as a double unit, being supplied via a single joint headgate with separate tailgates. At first sight it seemed possible for panels C and D to be extended for 400 m in length (i.e. including panel E). However this proved to be impractical for two reasons:

(a) The start of the faces would then have been over 900 m from the main haulages, raising supply problems, and

(b) a local fold cutting across panel E and part of panel C on the east side diminished the potential reserves.

These panels were therefore only 300 m long; at this length the face C was shortened initially to avoid the effects of this local fold, this accounts for the small rectangular area unmined in panel C. Panel D is trapezoidal in shape; the narrowing of the face at one end was necessitated by the seam contours. Figure 3.5 shows cross sections through the headgate, tailgate and face of panels C and D giving an idea of the variations in seam thickness and of inclinations. The average dip of the gate roads is about 7^{o}, with local variations up to 15^{o}.

Figure 3.6 shows a similar layout for panel T; in this case the panel length is approximately 400 m. Figure 3.7 gives cross sections through the face and gate roads of panel T. It will be noted that there is a much wider variation of seam thickness in this panel and, in parts, the seam thins to an extent that roof and floor rocks intrude into the face section (e.g. Figure 3.7 (c)) giving difficult face advance conditions and leaving no coal available for caving and drawing.

3.2.2 Transportation

(a) Coal discharge from the mine. The faces are identically equipped; each has two 50-cm wide chain conveyors, one for removal of the coal from the face advance in front of the supports and one behind the supports for removal of the caved coal. The front conveyor is powered by two 500-V, 36-kW motors; the rear conveyor is powered by two 500-V, 48-kW motors. Both conveyors have a speed of 50 cm/sec.

Each tailgate is equipped with similar chain conveyors, 50 cm wide, powered by two 36-kW motors with a speed of 73 cm/sec. These extend for the distances marked on Figure 3.4 for panels C & D and Figure 3.5 for panel T.

In the tailgate of panel C, see Figure 3.4, a 1-m wide belt conveyor, speed 2.2 m/s, powered by a 48-kW motor delivers coal to the transfer point at the intersection of roads Clement and Victor. A similar belt conveyor

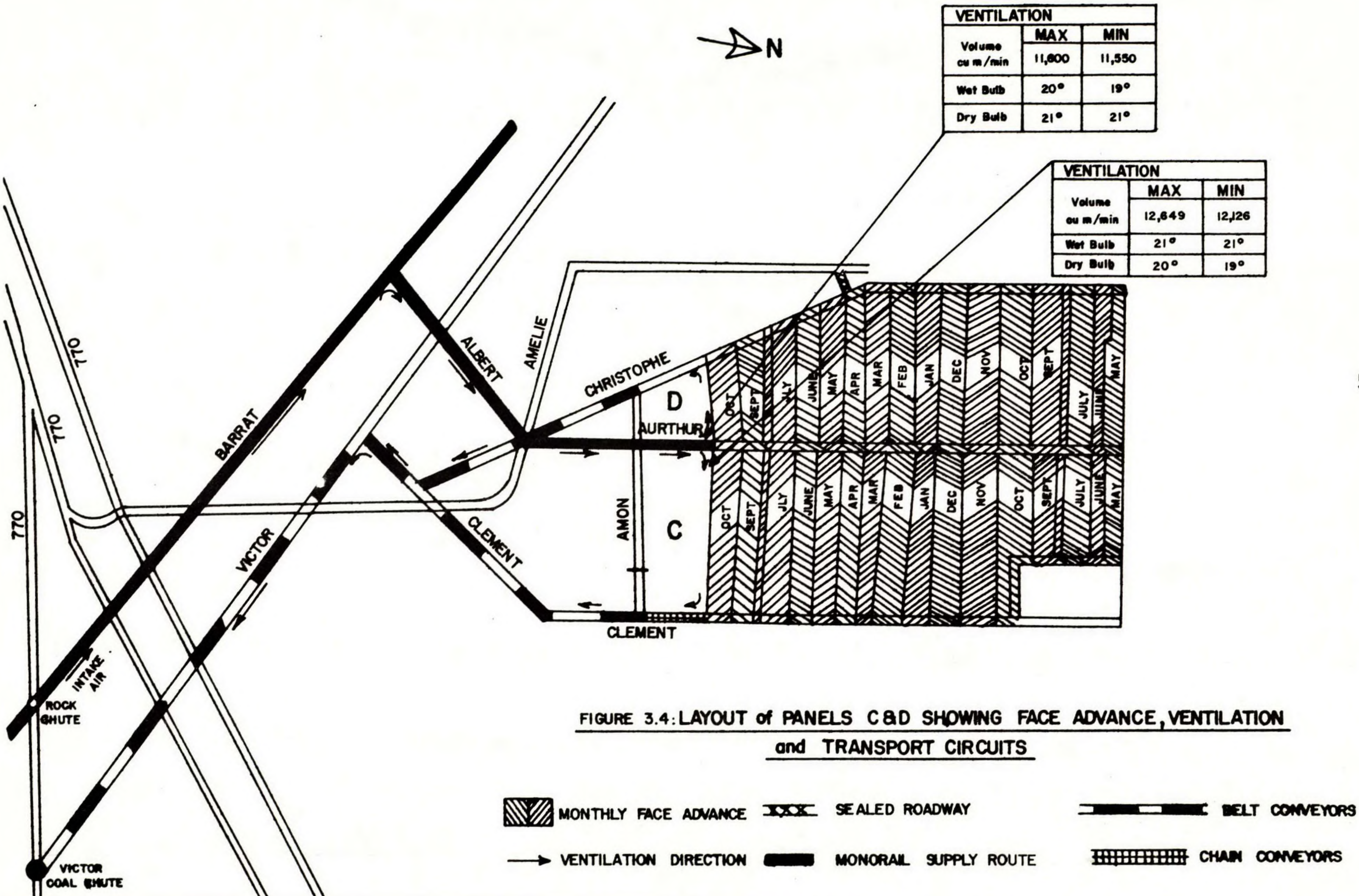

FIGURE 3.4: LAYOUT of PANELS C&D SHOWING FACE ADVANCE, VENTILATION and TRANSPORT CIRCUITS

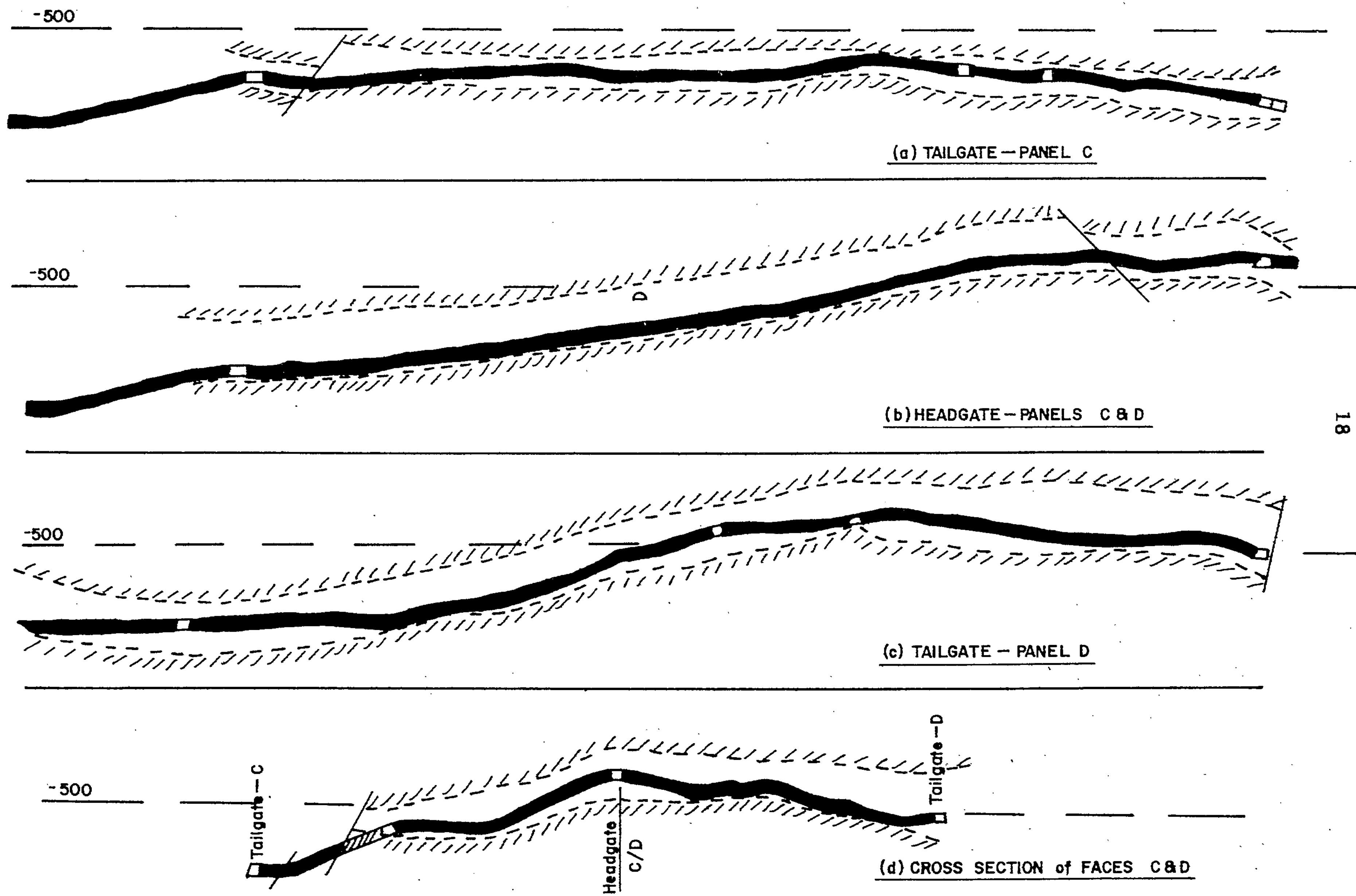

FIGURE 3.5: CROSS SECTIONS THROUGH HEADGATE, TAILGATES and FACE of PANELS C & D

SCALE 0 5 10 metres

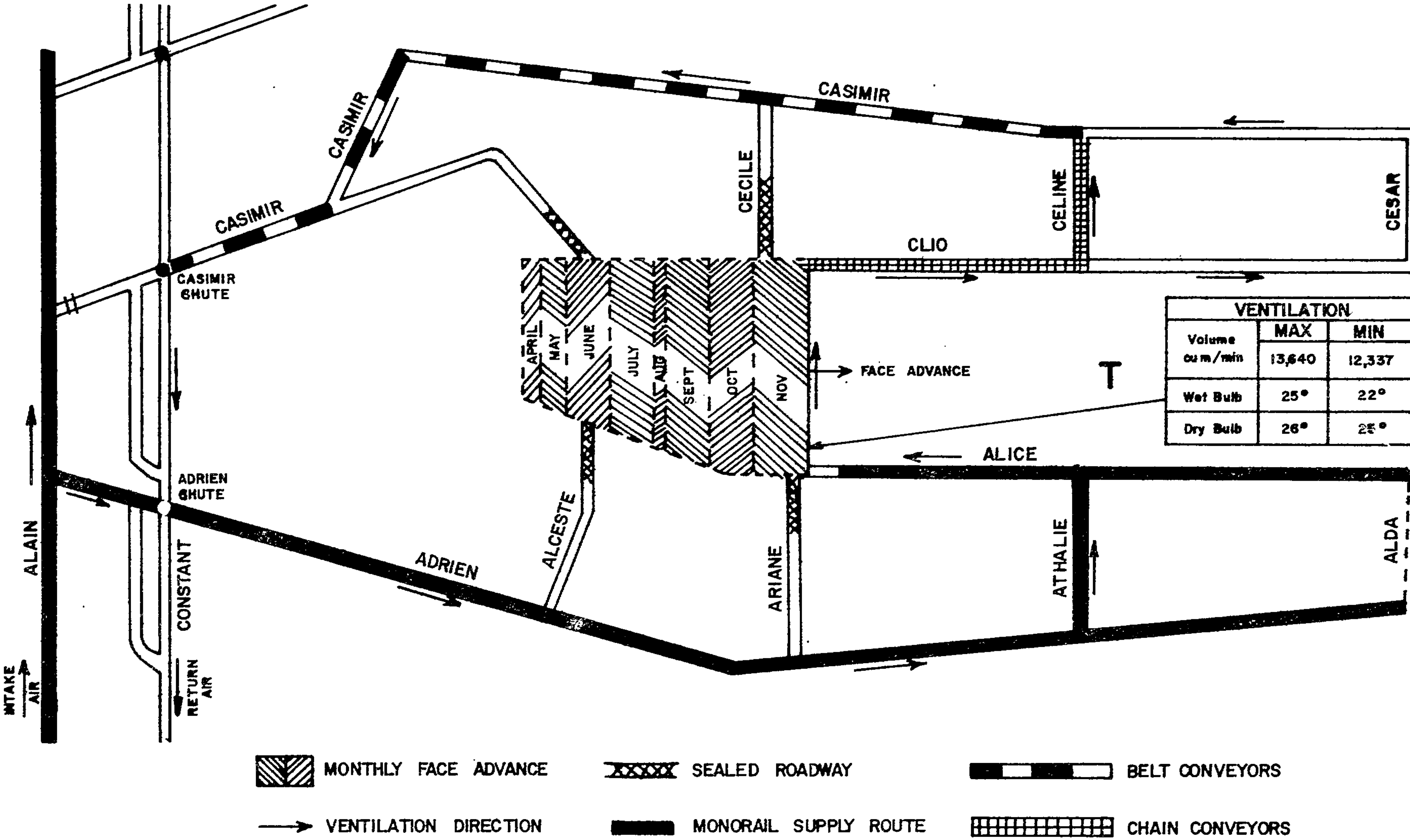

FIGURE 3.6: LAYOUT of PANEL T, SHOWING FACE ADVANCE, VENTILATION and TRANSPORT CIRCUITS

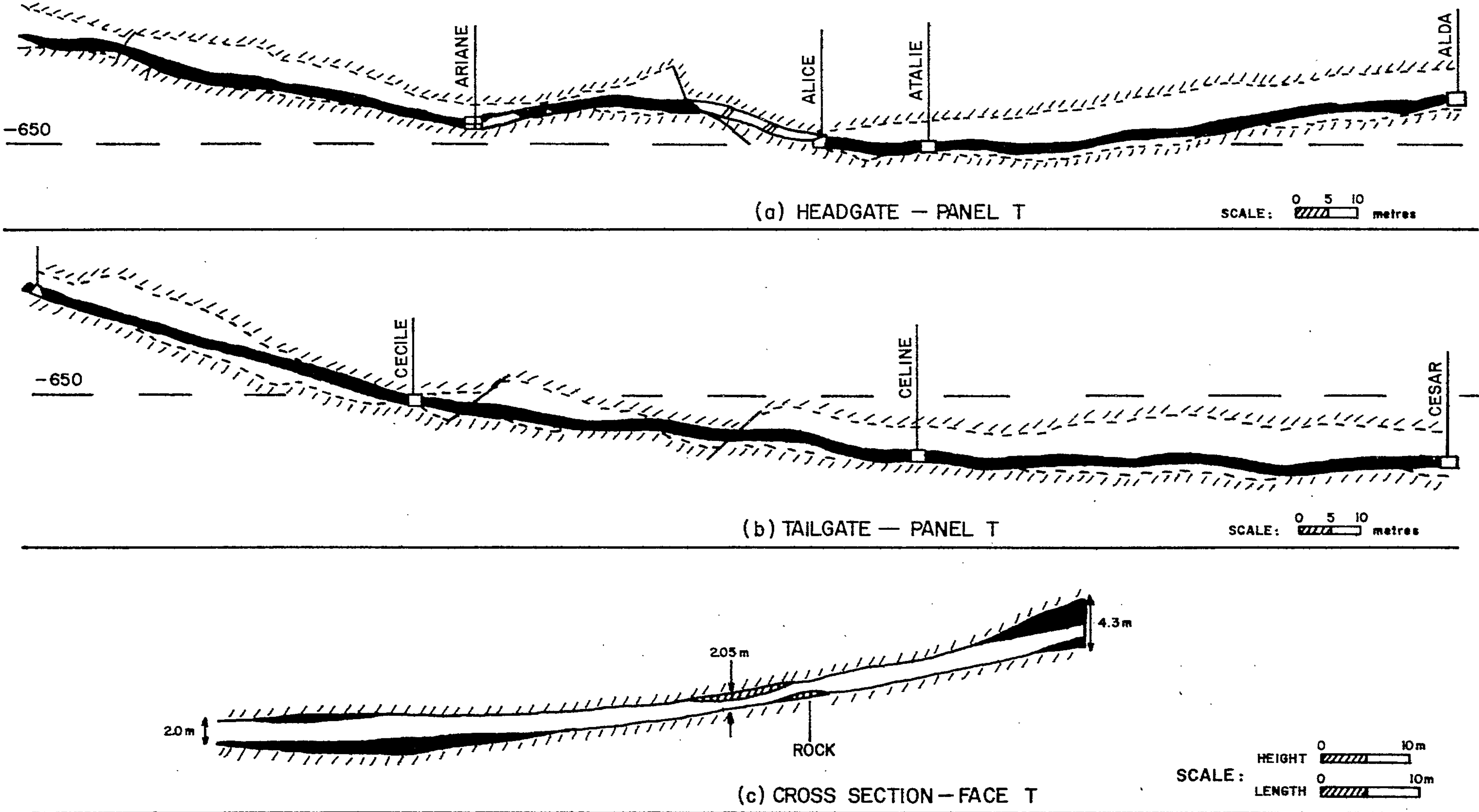

FIGURE 3.7: CROSS SECTIONS through HEADGATE, TAILGATE and FACE of PANEL T

powered by a 36 kW motor delivers coal from panel D, via a transfer point at the junction of roads Christoph and Clement, onto the panel C conveyor. Coal from both faces is transferred in road Victor onto a 1-m wide belt conveyor, speed 2.5 m/s, powered by two 500-V motors each of 180 kW. This conveyor delivers coal into the Victor storage chute. From the storage chute, wagons in the footwall road 770 are loaded for transportation to the shaft and out of the mine.

Figure 3.6 shows the similar layout for panel T. Similar conveyors are used in the face and tailgates to transport the coal to the Casimir storage chute from which wagons are loaded for transport to the surface.

All coal transport is restricted to return airways.

(b) Supplies. All supplies are taken from the shaft to within 50 m of the face in the headgates by means of diesel powered monorail locomotives. These locomotives weigh about 7.2 tons and can handle a pay load of up to 4.5 tons being transported up a 30% slope. Details and photographs of this monorail system are given in Appendix 3. The supply transportation routes, via intake airways, are marked on Figures 3.4 and 3.6 for panels C/D and T respectively.

In the headgates, 50 m from the face, a 35-cm wide chain conveyor equipped with 24-kW motors is used to transfer supplies to the face.

3.2.3 Ventilation

Figures 3.4 and 3.6 show the ventilation circuits in the vicinity of panels C/D and panel T respectively. Some typical face air flow volumes and wet and dry bulb temperatures are also marked on these figures. Gas emission in the current Blanzy seams does not pose a ventilation problem since it is only of the order of 2 m^3/ton produced.

3.2.4 Previous mining areas

Figure 3.8 shows the areas of previous mining which may affect the ground conditions during mining of current and future panels in Darcy No. 4 seam. Extensive areas of No. 2 seam, about 100 - 150 m above the current mining regions, have been previously extracted. In No. 4 seam panels G^a, G^b, I and J have been mined out.

3.2.5 Development

The main haulage and supply roads are driven in the underlying footwall rocks about 20 - 30 m below the seam. Inclines are then driven up to the seam and the panel is developed by driving the gate roads within the coal.

For the purposes of this study the term "panel" will be defined as including the face, the gate roads, the rock inclines and the haulage/supply roads as far as the storage chutes. In the Darcy mine the average block of coal to be mined in the panel is about 400 m long by 86 m wide; this

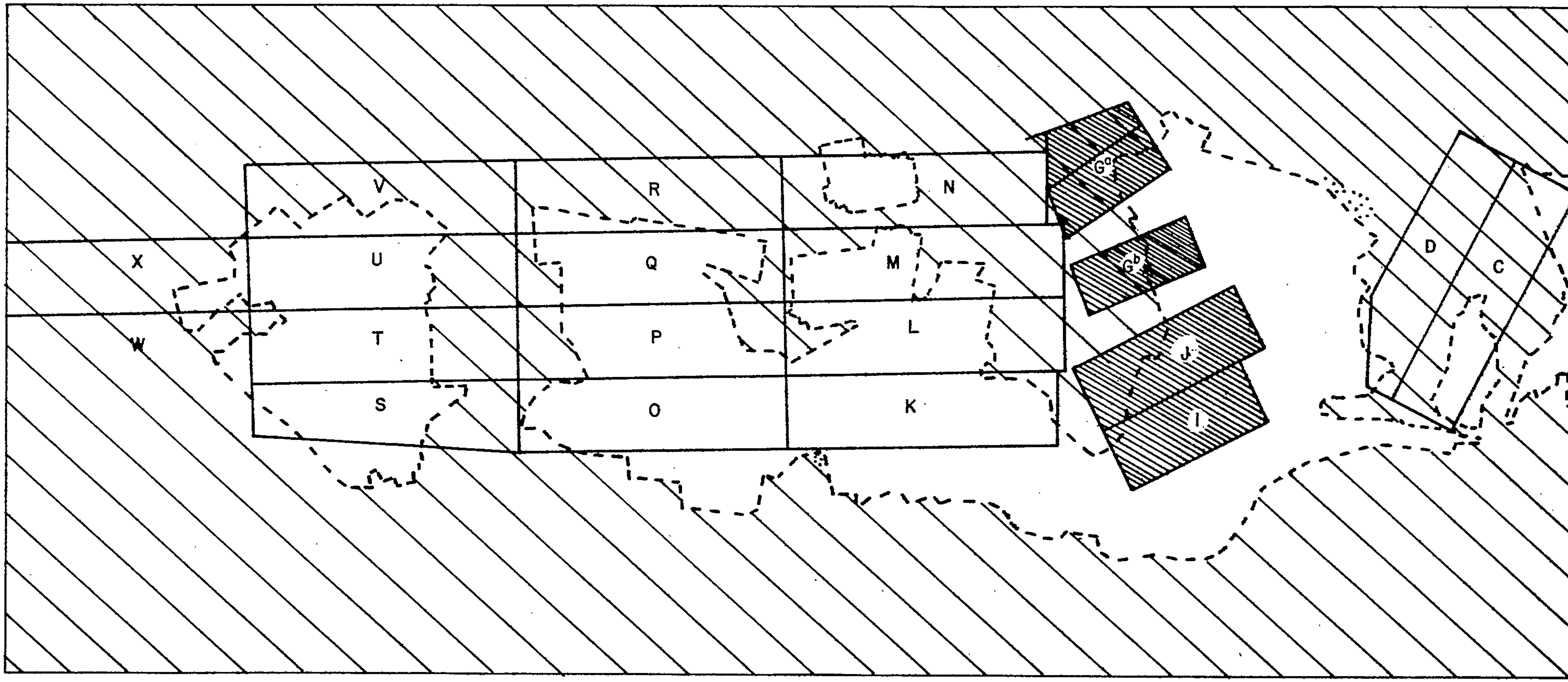

FIGURE 3.8 : PREVIOUS MINING at DARCY

necessitates the development of approximately 900 m of roadway in coal. The amount of rock development within the panel varies considerably but a very crude estimate indicates that this is of the order of 400 m per panel.

Development headings in the Darcy mine are driven using standard drilling, blasting and mucking techniques (in this mine one Alpine continuous miner is also now being used for development in coal; this machine will be discussed in more detail in Chapter 4 in the discussion of Rozelay mine). Support for the roadways, which are about 4 m high by 4.5 m wide, is provided by 5-element steel arches, type TH 470. These are spaced at intervals of about 1 m in rock and about 0.5 m in coal; timber lagging is placed behind the rings.

The average daily advance of development in the Darcy mine during the first 9 months of 1972, was 7.43 m/day for the whole mine; or for an "average panel" a rate of 2.25 m/day at an average productivity of 3.89 cm/ man shift (see Table A6.6 in Appendix 6). The cost of development, calculated in this Appendix, of F7270 ($1454) per metre appears to be excessive and reliance should not be placed on this figure.

3.3 The Mining Method

As described in Appendix 2, the longwall caving and drawing method was first introduced into this area in 1964. Since that time the method has been gradually mechanized with the introduction of two conveyors on the face (one for face advance and one for drawing), the use of walking supports (with the banana prop) and finally the use of shearers for face advance. In the Darcy mine, shearers have not yet been introduced (although a trial was carried out some time ago) and the face is still advanced by hand mining methods.

3.3.1 The mining cycle

Figure 3.9 illustrates the four phases of the mining cycle:-

Phase 1 The face is shown in the "closed" position with a minimum face width of approximately 5 m. The front conveyor is as far forward as possible, immediately against the face. The rear drawing conveyor is as close to the rear of the supports as possible and is protected by the banana prop.

Phase 2 The coal face is advanced, using explosives and hand picks, and the coal is loaded onto the face conveyor. An alley, 1.2 to 1.5 m wide is advanced for the full length of the face. Temporary support is placed as shown in more detail in Figure 3.10. This temporary support comprises a timber placed perpendicular to the face and supported at one end by the canopy of the walking support and at the other end by a friction prop placed hard against the face. These timbers provide temporary roof support, and are spaced about 0.75 m apart along the face. Wooden planks are placed against the face behind the friction props to prevent the face spalling into the advanced alley. The face is now in the "open" position with a width of about 7 m.

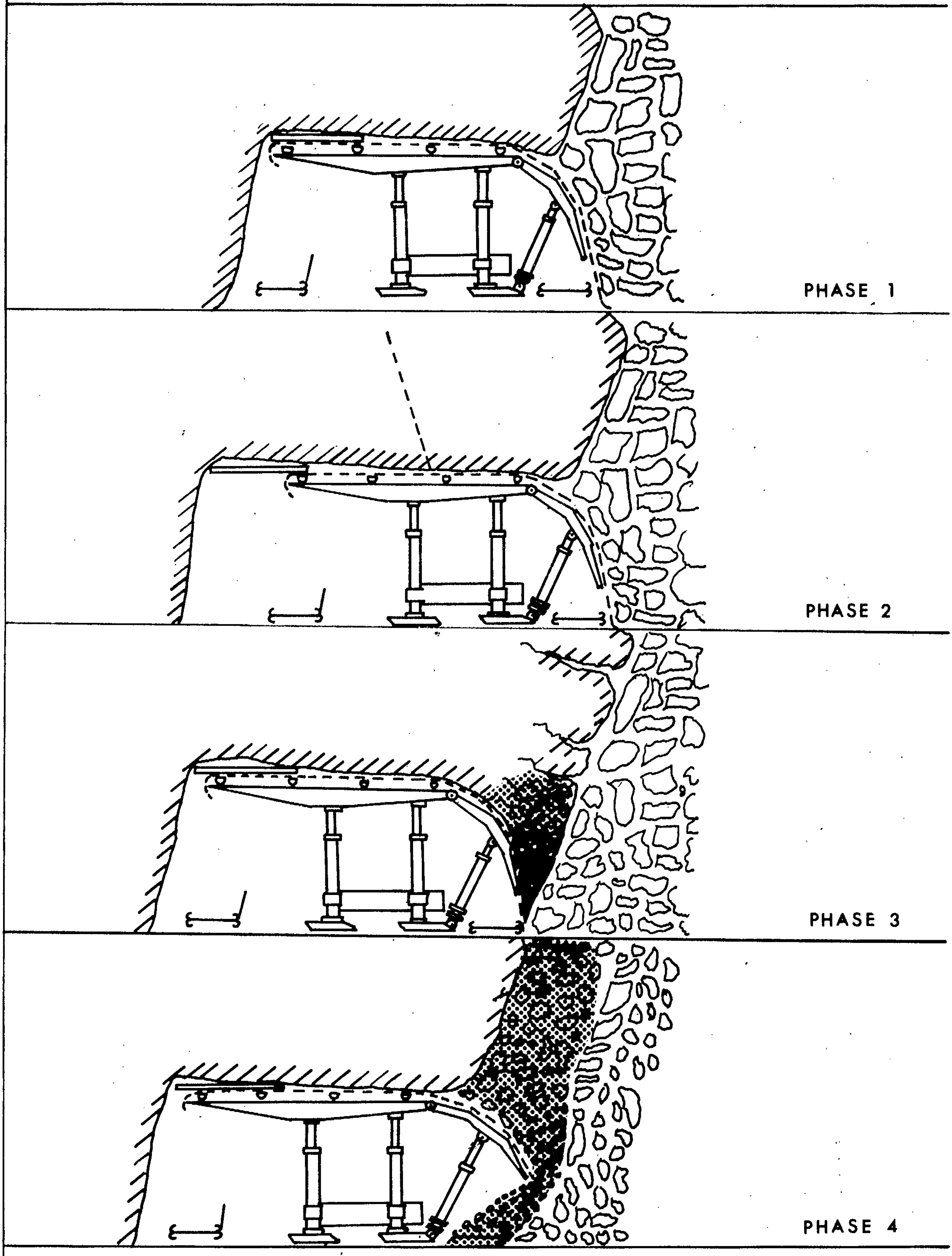

FIGURE 3·9 : THE MINING CYCLE AT THE FACE

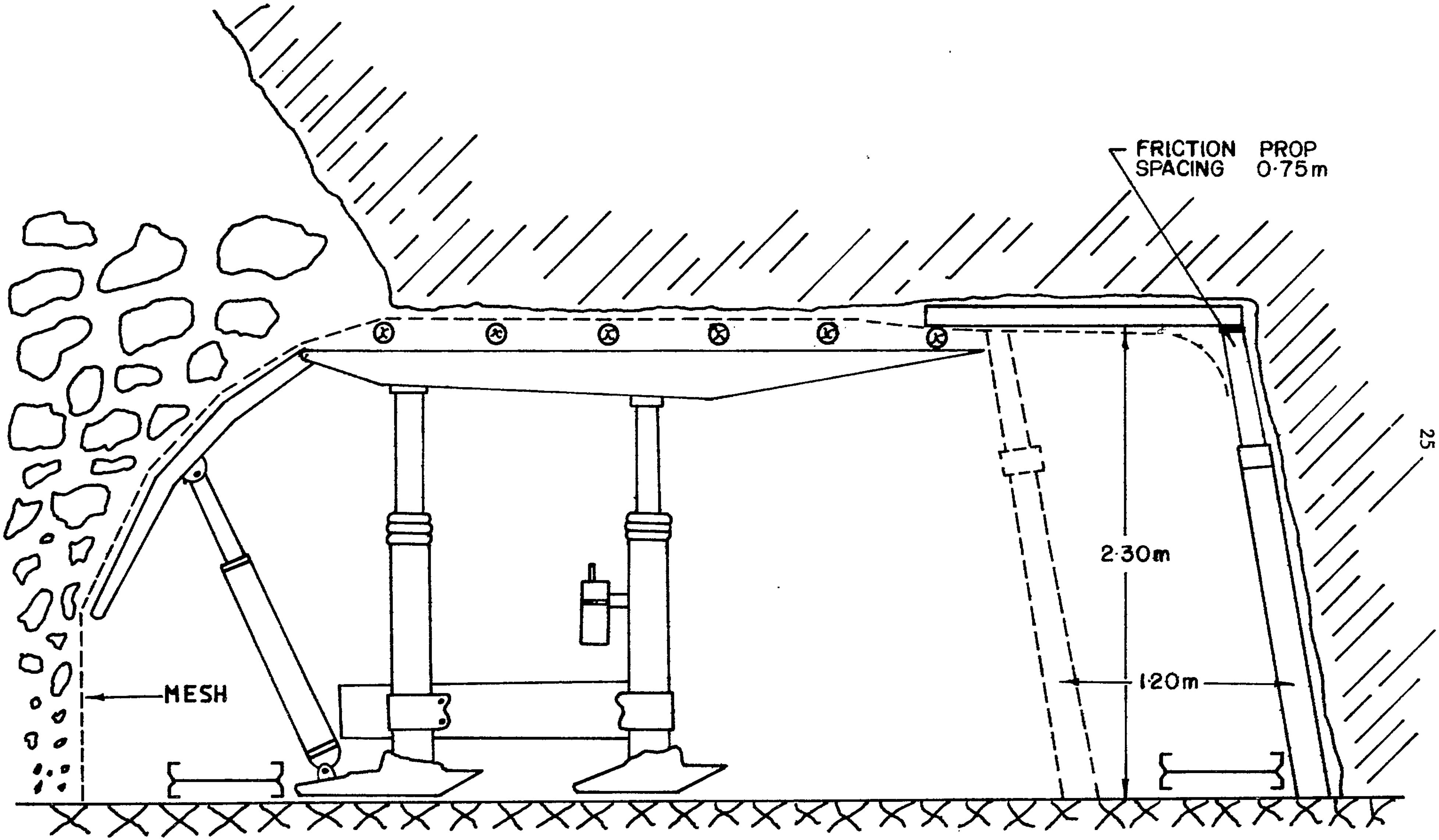

FIGURE 3·10 PROP AND TIMBER PLACEMENT FOR HAND MINING FACE ADVANCE

Phase 3 On completion of the face advance, the face conveyor is pushed forward for about 60 - 65 cm using independent jacks attached to the walking props. The wire mesh overlying the canopy of the supports is extended by laying out a mesh roll of about 1 m wide by 10 m long along the face and lacing this to the mesh already over the canopy, using wire lacing. The props are then sequentially lowered and advanced by about 60 - 65 cm (the maximum "step of the prop"). When the prop is in the lowered position, prior to the advance a timber is placed over the canopy, under the mesh, parallel to the face; this provides support to the mesh between the canopies of adjacent props. The rear conveyor is then pulled forward, using a second set of independent jacks attached to the supports. This cycle of front conveyor forward, prop advance, rear conveyor forward, is repeated for a second time so that the full 1.20 - 1.50 m face advance is achieved by the supports and the face is again in the closed position.

Phase 4 Caving windows are now cut, about every 5 m, in the mesh behind the props. Caving of the coal is assisted by pumping the banana prop up and down, causing the caved coal to flow onto the rear conveyor. About 5 caving windows are drawn simultaneously. Drawing is continued until caved roof rock appears at the caving windows; these windows are then closed by lacing mesh over the previously cut holes. In practice, of course, after the first cycle of advance of the face, Phases 1 and 4 are carried out simultaneously; the face being advanced by hand at the same time as coal from the previous advance is being drawn.

The above mining sequence is carried out on a daily cycle during three shifts. During the morning shift the hand face advance is commenced as is the caving and drawing. During the afternoon shift the advance of the face is completed, as is the drawing, and, if time allows, the mesh is extended and the support advance is started. The night shift is concerned solely with completing the advance of the supports and with carrying out any necessary maintenance of the supports, etc. With this system the face advance is limited to approximately 1 "alley" of 1.2 - 1.5 m per day; the face production in that time depending primarily on the volume of coal retrieved during caving and drawing which is in turn primarily dependent on the seam thickness.

Appendix 5 shows a series of photographs of the face operations.

3.3.2 The face ends and gate roads

Figure 3.11 (a) shows a sketch of the face and gate roads with three specific regions marked where specific jobs must be carried out to ensure good face advance.

(i) Support on the face and in the headgate adjacent to the face is provided by the walking supports. However, in the tailgate additional coal may be extracted by caving in the tailgate end. Here support is provided by timbers held with friction props; at the end of the tailgate a tripod support, as sketched in Figure 3.11 (b), is placed and planks behind this tripod helps control drawing of the caved coal. The tailgate chain conveyor is extended to the tailgate end,

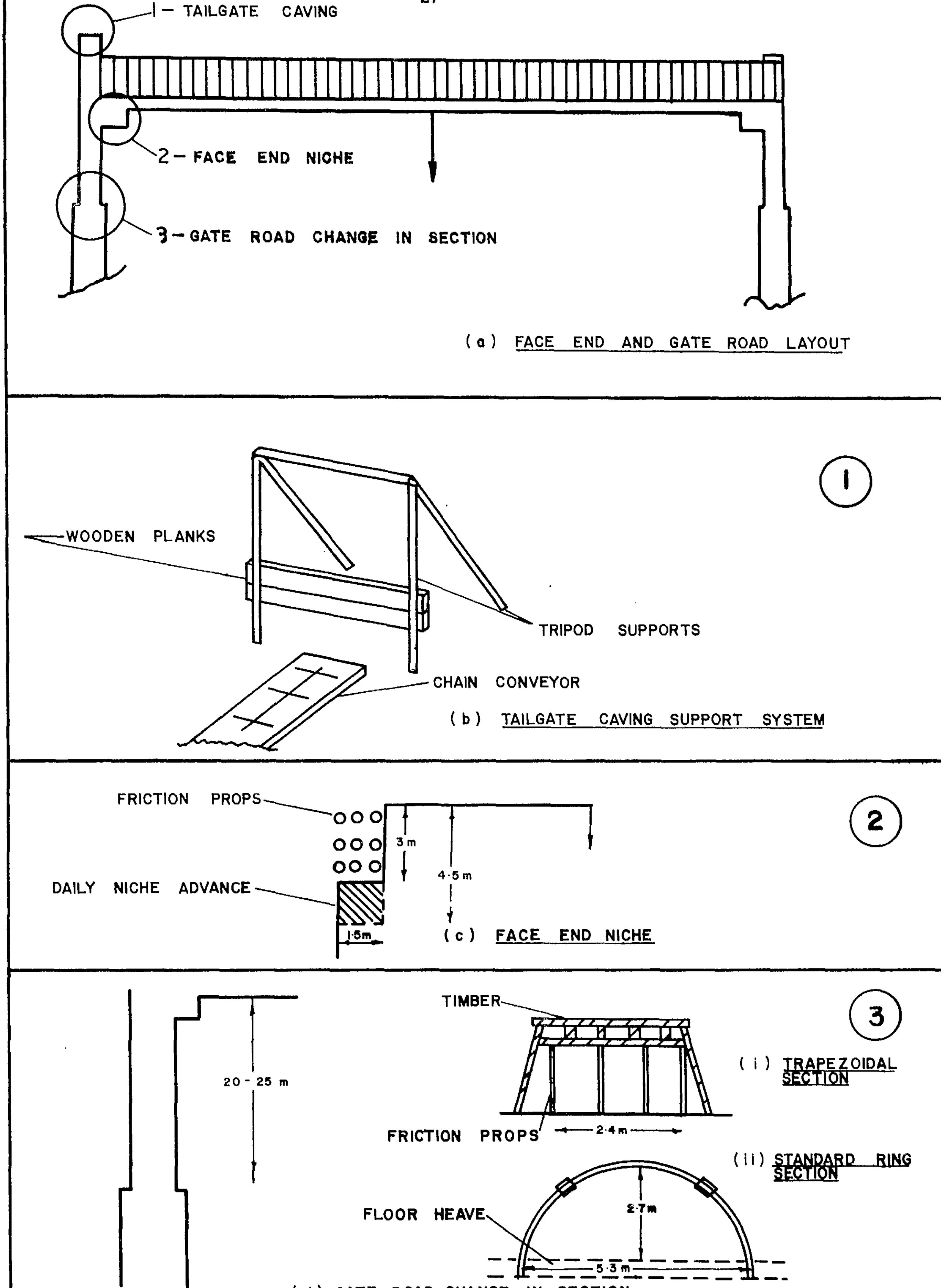

FIGURE 3·11 FACE END AND GATE ROAD SUPPORT AND LAYOUT (NOT TO SCALE) about 3.5 m behind the face caving line, to allow the coal to flow directly onto the conveyor. In an isolated double panel, such as panels C/D, where two panels are served by one headgate and two tailgates, caving and drawing is carried out in both tailgates. In a more usual situation, such as panel T, coal is only caved and drawn on tailgate side; the headgate of the adjacent panel (panel U in this case) is then driven immediately adjacent to the tailgate of panel T., i.e. when panel U is mined the coal will already have been drawn from above the headgate by the caving and drawing in the tailgate of T.

(ii) At each end of the face a niche, approximately 3 m deep by 1.5 m wide, is hand mined in advance of the face and supported by friction props. This niche provides the room necessary for the conveyor motors to be advanced. This is shown in Figures 3.11 (a) and (c).

(iii) At about 20 - 25 m ahead of the face, the gate road sections are changed from the circular arch support used in development to a trapezoidal section of timbers and props immediately ahead of the face; approximate dimensions are shown in Figure 3.11 (d). This allows recovery of the arch supports. At the same time the floor which may have heaved due to the abutment load ahead of the face is bottom-brushed.

3.3.3 Face equipment

The face conveyors have been briefly described in 3.2.2. The only other major equipment used in these hand advanced faces are the walking supports. The key element in the mechanization of longwall caving and drawing systems has been the development of walking supports with the "banana" prop for controlling drawing. Figure 3.12 shows a photograph of these walking supports. Although details may vary, the supports produced by various manufacturers are quite similar in design. Four different types of walking supports are used in the Blanzy mines; Appendix 4 details specifications of these supports and also gives a breakdown of capital and maintenance costs and performance data for these supports.

3.4 Good and Bad Mining Practice

The following sections describe the general principles which should be followed to achieve good face advance, caving and drawing and also indicate a number of factors which should be avoided. In addition, a number of miscellaneous problems, observed during study shifts in the Darcy mine, Panels C, D and T are described.

3.4.1 Face layout and advance

(i) The mining panel should be laid out for retreat mining down dip or down the apparent dip. Mining up dip should be avoided at all costs since it leads to bad face conditions, as illustrated in Figure 3.13; the supports become heavily loaded and the chances of the roof caving between the front of the supports and the face is increased.

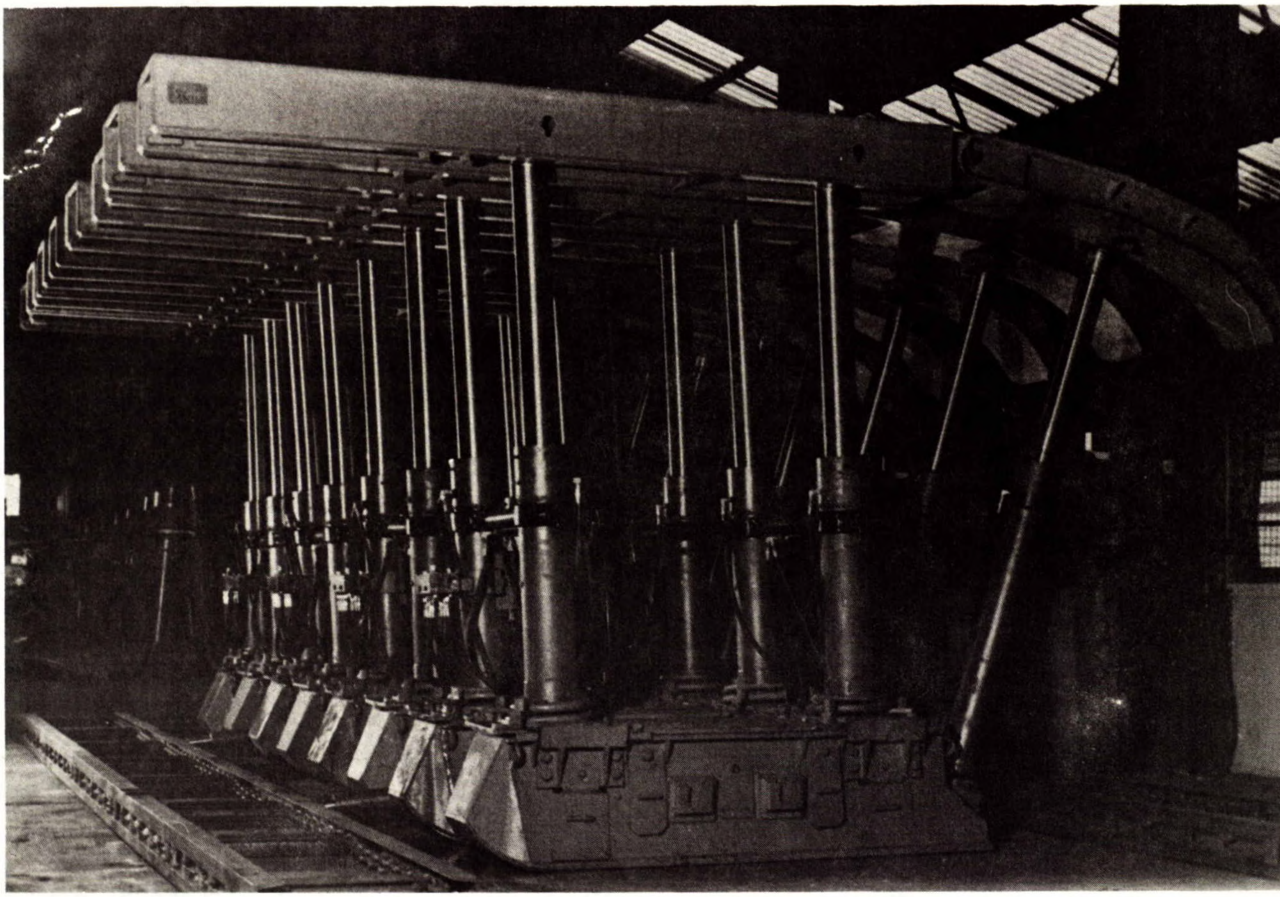

FIGURE 3·12 : S.M.F. POWERED SUPPORTS FOR CAVING AND DRAWING

MAXIMUM HEIGHT 2·72 m. MINIMUM HEIGHT 1·72 m.
SETTING LOAD 60tons/prop YIELD LOAD 70tons/prop

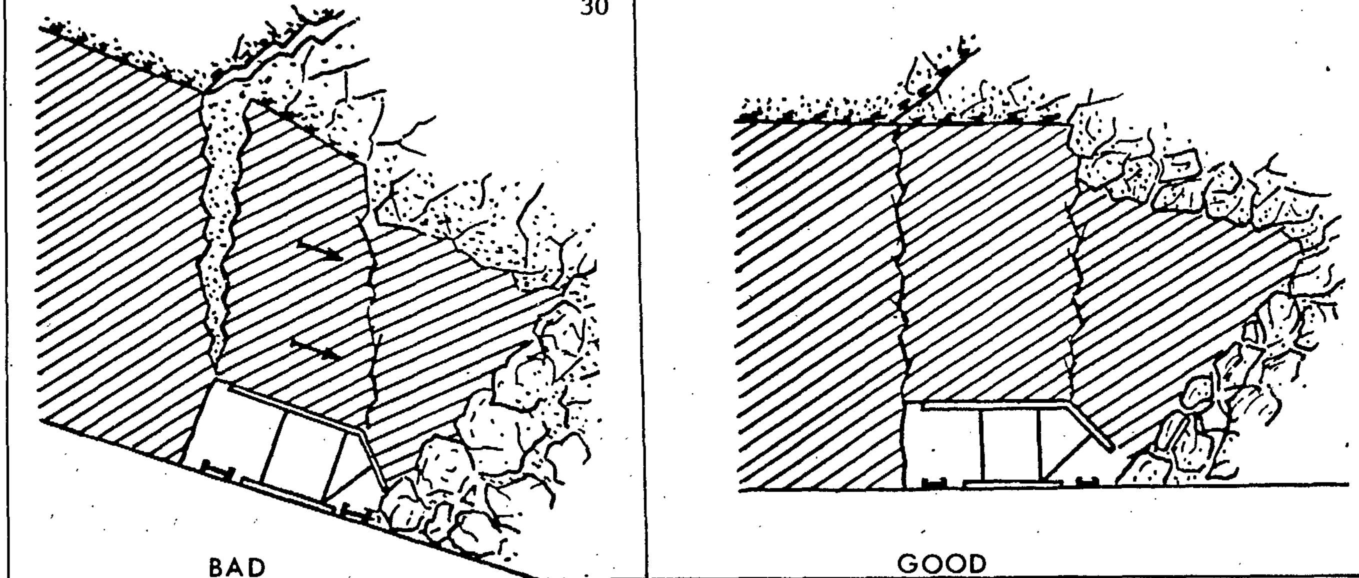

FIGURE 3·13 : Up dip mining gives bad face conditions; supports may become heavily loaded and the danger of caving between the front of the supports and the face is increased.
If possible, a level or slightly descending front is to be preferred; always avoid a rising front.

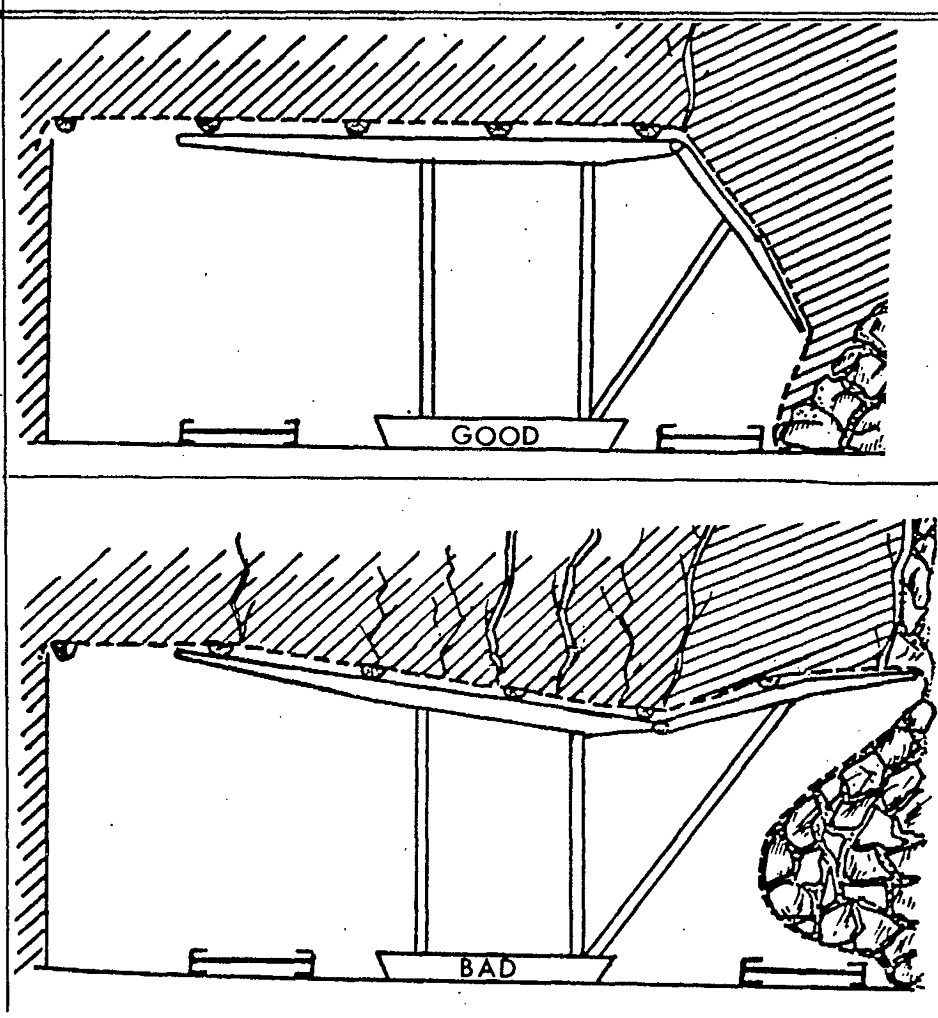

FIGURE 3·14

The banana prop should not be considered as an element of the roof support; it should be regarded only as a brace to protect the rear chain conveyor.
The walking support is more stable when the banana prop is in a lowered position.

(ii) If possible, the face should be started against the boundaries of existing mining operations where the roof will cave quickly (boundary of previous working, fault zone, etc.).

(iii) The face should be on or close to the footwall, depending on local conditions such as load, floor heave, seam roll, etc.

(iv) In most conditions, the roof should be lagged with wire mesh; this mesh passes over the top of the supports and then follows the banana prop down to the floor and passes beneath the caved coal. When the face is started, if wire mesh cannot be installed at the face, it should be placed before the supports are installed and held in position with timber. The mesh should, at the start of the face, be solidly anchored to the floor at the rear of the supports by bolts.

(v) The effective height of the face should be less than that of the maximum of the powered supports; it is a compromise between ease of operating face equipment and the necessary chock loading. It should be reduced to the minimum compatible with easy mining and drawing.

(vi) The width of the face depends on local conditions and equipment to be used; it should however be kept to a minimum (Figure 3.15). Too wide a cut in front of the supports and a raised banana prop at the rear can produce prohibitive face spans, overloading the supports, causing yielding and premature break up of the overlying coal. This could lead to caving at the face ahead of the supports, and to coal being drawn from above the supports.

(vii) The rear banana prop should not be considered as an element of the roof support; it should only be regarded as a brace to protect the rear chain conveyor and as a means of controlling the drawing (Figure 3.14). The powered supports are more stable with the banana prop in the lowered position.

(viii) High prop setting loads (close to the yield load) are preferred and tend to produce good support stability, better face and roof conditions and better caving.

(ix) Good alignment of the supports along the face allows maximum advance of the rear conveyor, thus reducing the face span (Figure 3.16). Poor support alignment gives inadequate protection at the face.

3.4.2 Caving and drawing

(i) The caving 'span' is a function of the thickness of the coal to be caved and of the roof and coal properties. The optimum step appears to be between 1.2 and 2 m. For a face height of 4.5 m the maximum step is 1.8 m. When the roof breaks in large blocks the step should be reduced to perhaps as short as 1 m. (Figure 3.17).

(ii) Drawing should be carried out with the banana props in the lowered position and the rear conveyor as close to the supports as possible.

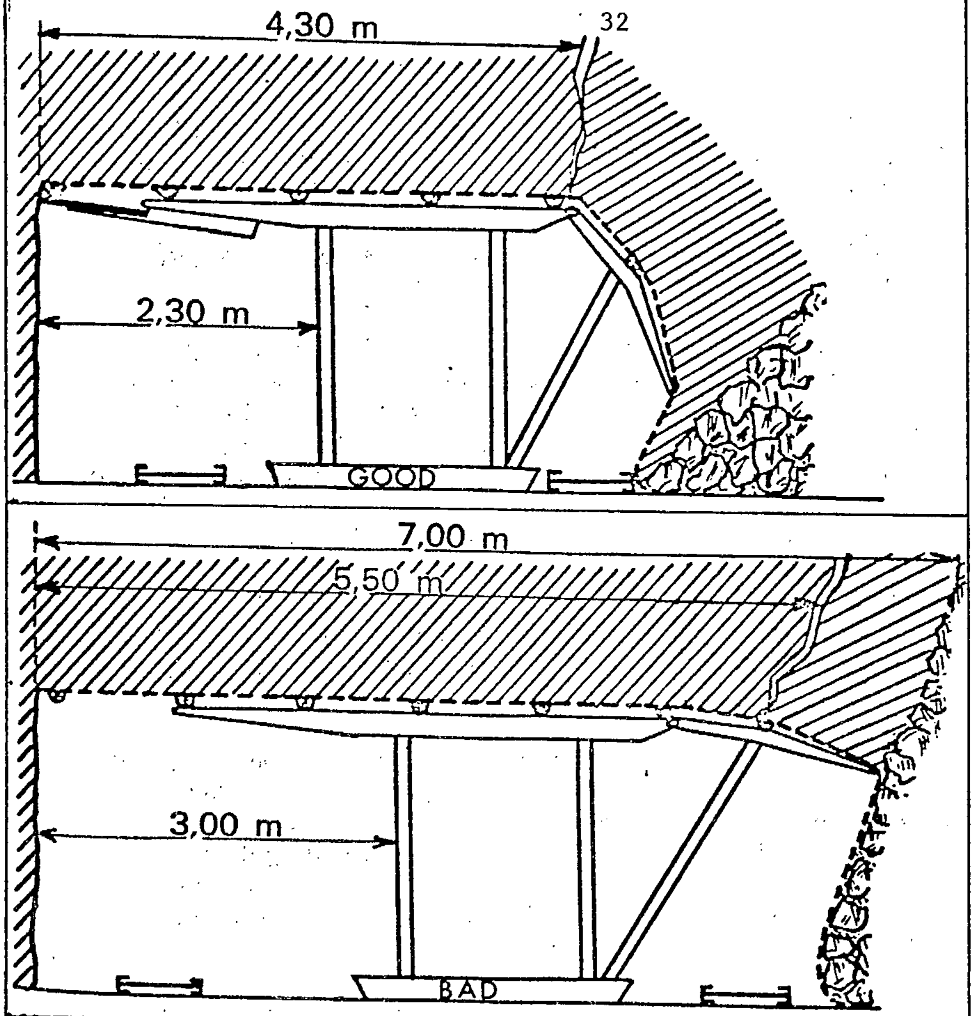

FIGURE 3·15

Too wide a machine alley and a wide rear alley with an extended banana prop produce a prohibitive face span.

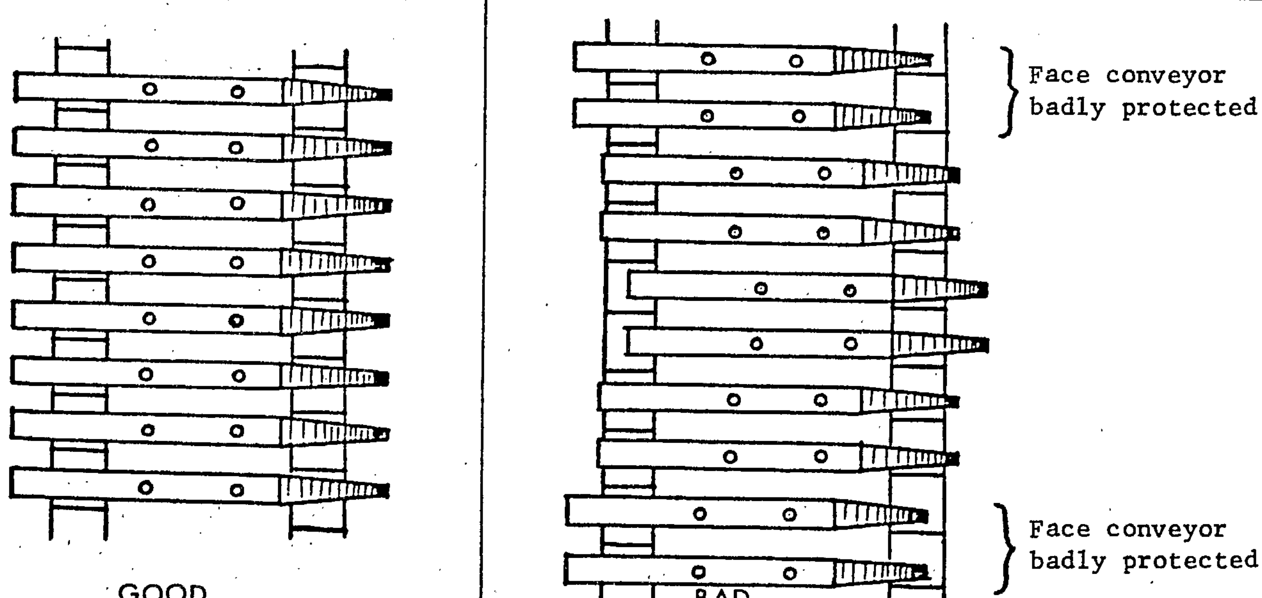

FIGURE 3·16 : Good alignment of the supports on the face allows maximum advance of the rear conveyor and thus reduction of the span; poor alignment correspondingly increases this span, and leads to inadequate protection at the face.

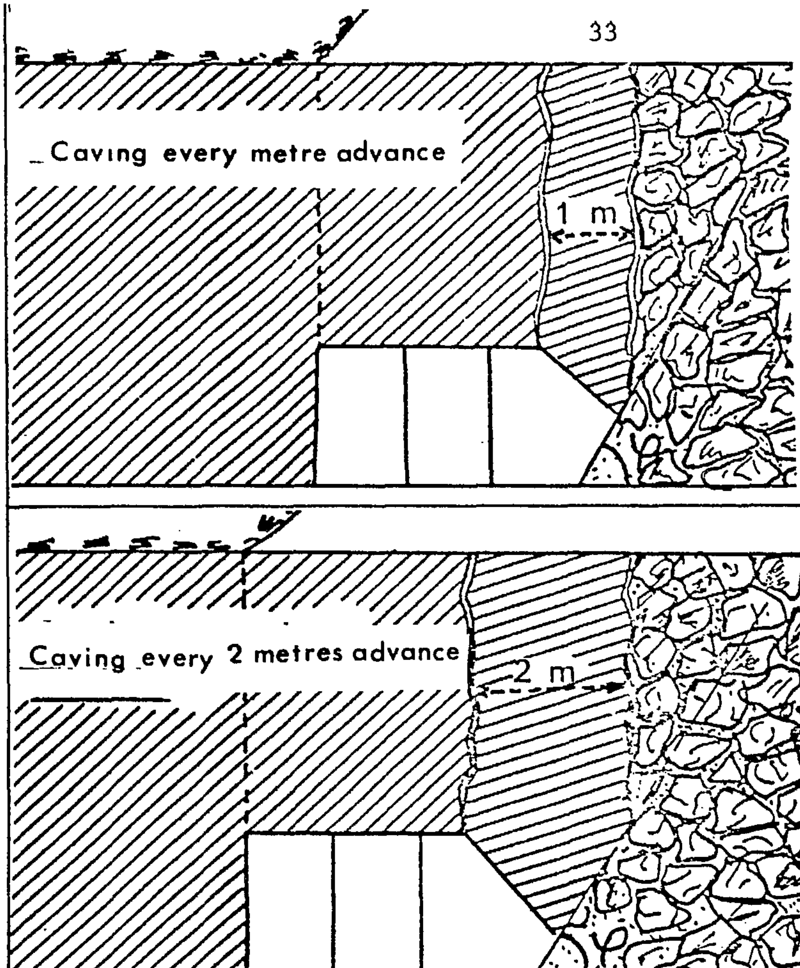

FIGURE 3·17

The step of caving is a function of thickness of coal to be caved and of roof and coal properties.

The optimum step appears to be between 1.2 m and 2 m. For a height of 4.5 m, the maximum step is 1.8 m. Nevertheless this may be reduced as far as 1 m when the roof breaks in large blocks.

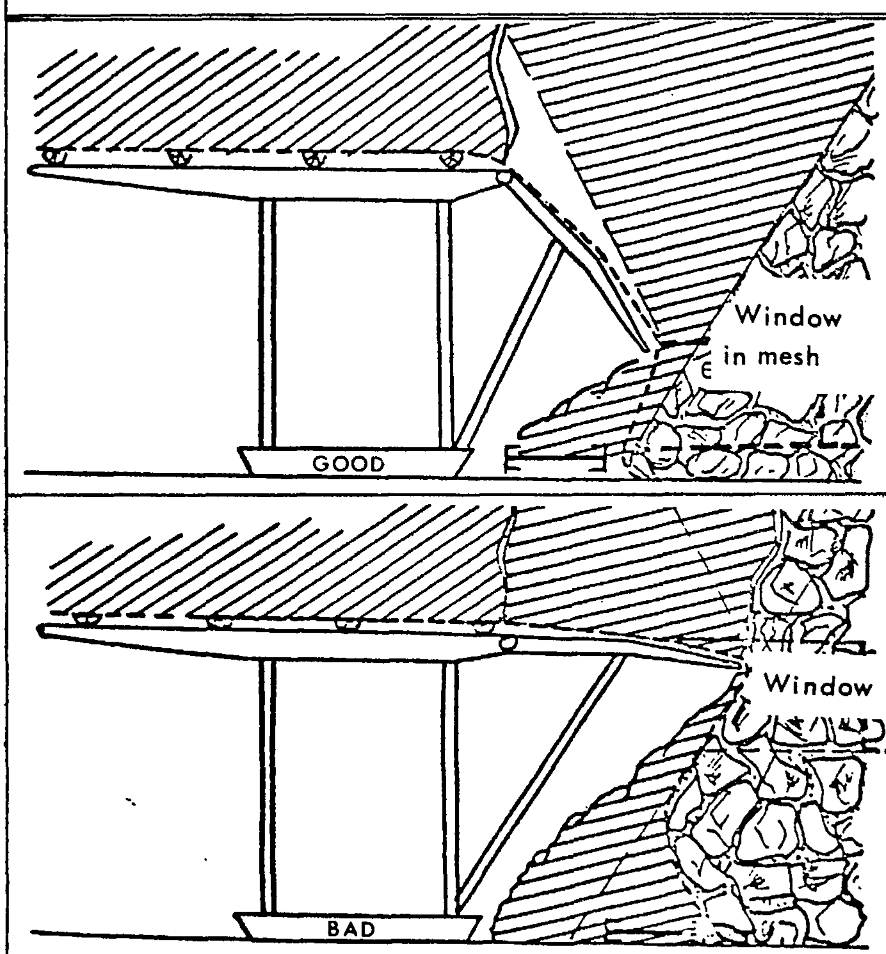

FIGURE 3·18

Caving should be carried out with the Banana props lowered and the rear conveyor as close to the supports as possible. Drawing windows in the mesh should be cut as low as possible.
Windows should be about 50 cm x 50 cm and should be closed on completion of drawing.

The drawing windows cut in the mesh should be as low as possible. Windows of approximately 50 cm x 50 cm seem to be most suitable. These windows should be closed with mesh on completion of coal drawing, otherwise an unattended open window may spill significant volumes of roof rock onto the conveyor during drawing at a position farther along the face (Figure 3.18).

(iii) Drawing at a single point is forbidden in order to avoid the hour-glass effect which leads to dilution and loss of coal. Drawing window spacing is a function of caving height. At the start of drawing the spacing may be 3 m and reduced to 1.5 m apart at the end of the draw. Drawing should be started at about 15 m from the head or tailgate. The drawing time from each window should be limited to 3 - 4 minutes (approximately 10 tons); thus drawing is completed over several passes at each window. Drawing from successive windows in line is preferable to a massive long draw from windows far apart (Figure 3.19).

(iv) With the banana props in the lowered position during drawing, there should be no hesitation about ramming the banana props up and down to prevent formation of pillars and to break up lumps.

(v) Drawing should be stopped when the caved rocks begin to appear at the drawing window. If possible, drawing should be stopped before coal is pulled from above the supports. This can be helped by specifying the quantity to be drawn each day (depending on seam thickness and advance). In general, exceptionally large recovery indicates extraction from above the supports which can lead to poor face conditions. In addition, an excessively large draw on one day usually means a low recovery the next day; large daily fluctuations in production can disrupt the mine planning and lead to additional operating costs (Figure 3.20).

3.4.3 Miscellaneous problems

The following problems were observed in the course of study shifts carried out on faces C, D and T. These observations must be regarded merely as a random sampling and in consequence it is not possible to generalize on the frequency of occurrence of such problems. None of these problems was particularly serious but each caused some delay and they might be regarded as typical of the day to day minor problems on the face.

(i) Caving operations in the tailgate of face D produced excessive amounts of dust. It is normal practice to infuse the coal ahead of the face with water (as described in detail in Chapter 6, Section 6.1.1) to help control the dust produced during caving. On the face this appeared to be exceedingly effective. However, as will be seen from the water infusion hole layouts shown in Figure 6.2, Chapter 6, there is little chance for the infused water to penetrate the coal above the tailgate; as a result the dust conditions in the tailgate during caving are exceedingly bad.

(ii) On occasion during drawing, large blocks of rock will completely fill the drawing window although there is still plenty of coal behind this

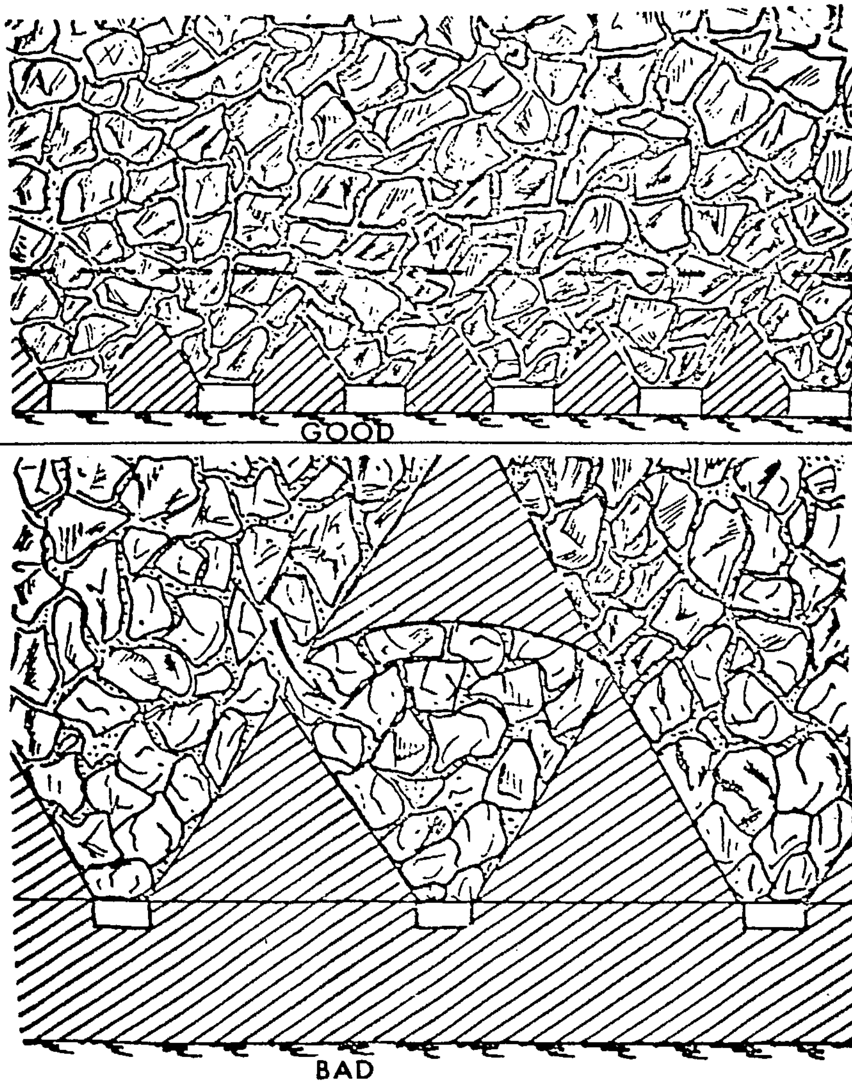

FIGURE 3·19

Draw window spacing is a function of the caving height. At the start they should be ~3 m apart and reduced to at least 1.5 m apart by the end of drawing.
The drawing team should start about 15 m from the head or tail gate. Drawing time at each window should be limited to 3-4 minutes (~ 10 tm). Drawing is thus in several passes.
Drawing from successive windows in line is preferable to massive drawing from windows far apart.

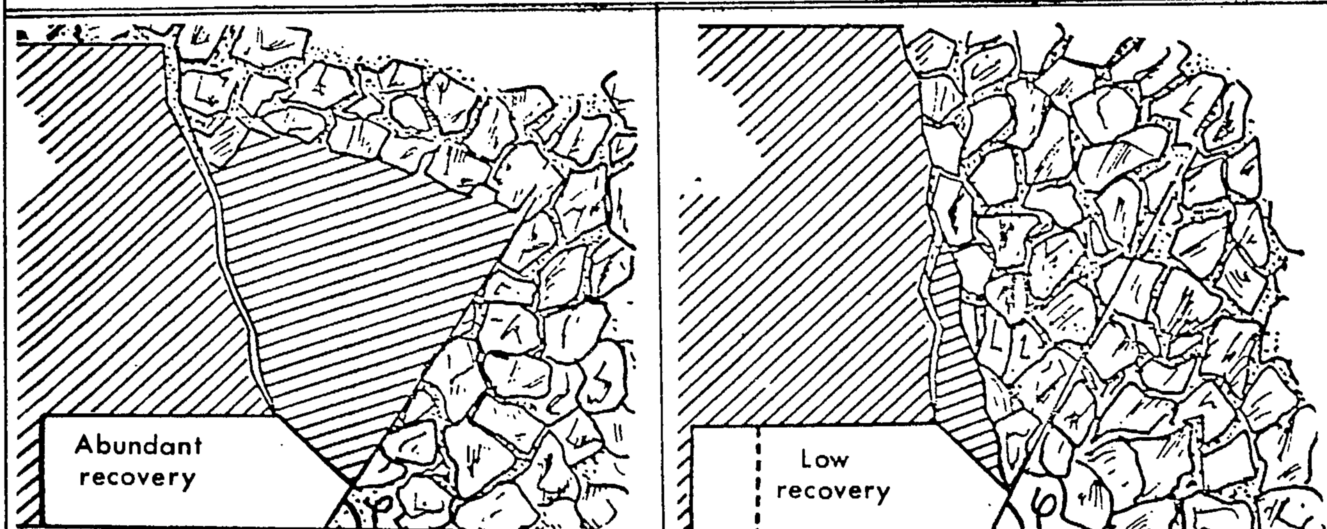

FIGURE 3·20: Stopping the Draw.

(a) Do not extract from above the supports.
(b) If the roof breaks into small blocks, cease drawing when the stones appear at the window.
(c) Specify the quantity to be drawn each day. In general exceptionally large recovery indicates extraction from over the face and precedes a low recovery period.

rock ready for drawing. It is then necessary to stop the rear drawing conveyor and to use an air pick to break up this rock to free the draw point. The broken rock is then spilled onto the conveyor and is carried off with the raw coal, causing dilution.

(iii) On one occasion the author observed roof rock being drawn onto the rear conveyor from one draw point for at least 5 minutes, representing probably 10 - 20 tons of rock. Such practice is forbidden but from the miners' point of view has the advantage of increasing "raw coal" production on the shift; what happens at the cleaning plant is of course someone else's problem! Any rock drawn during these operations is mixed with the raw coal drawn as there is no place on the face where this rock could be separated or otherwise removed from the face.

(iv) On one occasion it was noticed that when one side of the walking support was advanced the caved material, held back by the mesh, followed the support advance. It was then impossible to release the weight off the rear banana prop; neither could the banana be moved into an upright position from the skew position to which it had been pushed. Eventually, after much manoeuvring, a timber was placed under the adjacent banana prop which was then jacked up. This pushed the mesh and caved material back, allowing the banana prop to be released and pushed upright again. It took approximately 20 minutes to advance this powered support compared with the usual 6 - 8 minutes.

(v) On a number of occasions it was observed with the SMF supports that the pins connecting the piston legs to the canopy had sheared; this caused a delay until the repair man arrived.

(vi) During advance of the supports on face T on a night shift it was observed that, at one stage, of 41 supports being advanced no less than 8 were awaiting maintenance in one form or another. This represented almost 20% of the supports requiring maintenance on this section of face during this shift. This seems to be excessive; however, it is not known whether this could be regarded as a typical sample.

3.5 Manpower Distribution

Manpower distribution in the faces and in the panels will vary depending on both mining conditions and on absenteeism. Tables 3.1 and 3.2 below give typical manpower figures in terms of jobs done and numbers of men required per shift for both the panel and for the faces C and D. Each shift is 8 hours long from pit head to pit head; this represents approximately 6½ - 7 hours panel working time when transportation time is taken into account.

In Darcy mine, shifts 1 and 2 (morning and afternoon) are the production shifts during which time the face is advanced and the caved coal is drawn. Shift 3 (night shift) is purely a maintenance shift during which time the supports are advanced.

TABLE 3.1

Typical Distribution of Manpower in Panels C and D

Job Description	Shift 1	Shift 2	Shift 3	Total
On Face C	41	34	23	98
On Face D	39	32	23	94
Gate road maintenance - Tailgate C	14	14	14	42
Gate road maintenance - Headgate C/D	16	14	18	48
Gate road maintenance - Tailgate D	16	14	18	48
Coal transportation	9	8	9	26
Materials & supplies	14	25	14	53
Electricians	15	13	13	41
Miscellaneous labour in panels	17	7	6	30
Water infusion of coal ahead of face (dust suppression)	8	8	5	21
Total workers	189	169	143	501
Supervisors ("Fire bosses")	14	10	10	34
Total manpower panels C and D	203	179	153	535

3.5.1 Manpower distribution in the panel

Table 3.1 gives the manpower distribution, by job description, for panels C and D. The panel is defined as including the face, the gate roads, etc. up to the delivery of coal from the belts to the storage chutes (i.e. Victor coal chute, Figure 3.4, for panels C and D).

This table does not include panel labour on development. Appendix 6 (Table A6.4) includes statistical data for the panels C, D and T averaged over 6 months from which labour required for an "average panel" is derived. This will be further discussed in section 3.7. For the average panel a total of 254.2 shifts per day are required. (This includes panel development).

3.5.2 Manpower distribution on the face

Table 3.2 below gives a typical manpower distribution on the faces C and D by job description for each shift.

TABLE 3.2

Typical Manpower Distribution on Faces C and D

Job Description	Shift 1	Shift 2	Shift 3	Total
Miners	36	20	8	64
Shot firers	6	4	-	10
Drawing coal	10	10	-	20
Tailgate caving & drawing	4	-	-	4
Headgate caving & drawing	2	2	-	4
Advancing powered supports	-	8	24	32
Face maintenance	4	4	-	8
Hydraulic maintenance	4	4	4	12
Chain conveyor operations	4	4	2	10
Others	10	10	8	28
Total face labour	80	66	46	192

Again, Appendix 6 gives completely detailed face statistics for faces C, D and T over a number of months and derives mean figures for an "average face". The breakdown of these mean figures into specific jobs will follow closely the job breakdown given in the Table 3.2 above.

On average, the fraction of face labour to total panel labour (including the face) is 0.32.

3.6 Production, Productivity and Operating Costs for an Average Face and Panel

Appendix 6 gives production, productivity, operating cost and development statistics for the faces and panels C, D and T over a period of up to 10 months. From these figures the requisite statistics for an "average panel" have been derived. These statistics for the average panel are listed in Table 3.3.

The following important points are also shown in the analyses given in Appendix 6.

(a) The percentage seam recovery from a panel using this method, on average, exceeds 95%.

(b) The prime variable affecting productivity on the face and in the panel is the seam thickness. The effects of face length and face advance rate on productivity are minimal.

(c) The average face is 84 metres long, the average seam thickness is 10.2 metres. The average daily production is 990 tons net (or 1280 tons gross) with a face productivity of 13.0 tons net/man shift (16.8 tons gross/man shift) and a panel productivity of 3.89 tons net/man shift (5.04 tons gross/man shift).

(d) Total face operating costs are 28.8 F/ton net (\$5.77/ton net) and panel operating costs are 70.37 F/ton net (\$14.07/ton net).

(e) Labour accounts for 56.6% of the face operating costs and 72.5% of the panel costs.

3.7 Major Capital Costs for an Average Face and Panel

3.7.1 Face

The mean face in length is 84 metres, requiring 54 self advancing supports; assume that the face is equipped with SMF supports.

1. 54 supports at F43,960	2,373,840 F	\$474,768
2. 2 face conveyors at F350,000	700,000 F*	\$140,000*
Total - Face	3,073,840 F	\$614,768

3.7.2 Panel

3. 2 gateroad chain conveyors at 50 m each. 100 m at F350/m = 350,000 F* = \$ 70,000*

4. 425 metres belt conveyors in gateroads at F500/m = 212,500 F* = \$ 42,500*

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* Author's estimate.

TABLE 3.3

Statistics for an Average Face and an Average Panel in Darcy Mine

	ITEM	FACE		PANEL	
A	DIMENSIONS				
1	Face length - metres	84.3			
2	Face height - metres	2.1			
3	Seam thickness - metres	10.22			
4	Monthly face advance - metres	15.5			
5	Mean daily face advance	0.83			
B	PRODUCTION AND PRODUCTIVITY				
6	Percentage seam recovery	95.8		95.8	
7	Tons net/tons gross %	77.1		77.1	
8	Monthly production tons/net	18,210		18,210	
9	Mean daily production tons net	990		990	
10	Productivity tons net/man shift	13.0		3.89	
11	Monthly production tons gross	23,200		23,000	
12	Mean daily production tons gross	1,280		1,280	
13	Productivity tons gross/man shift	16.80		5.04	
C	LABOUR				
14	Face labour	Shifts/1000 tons net	Shifts/day	Shifts/1000 tons net	Shifts/day
	Hand mining face and niche	31.7	31.4		
	Caving and drawing	10.8	10.7		

TABLE 3.3 (continued)

	ITEM	Shifts/1000 tons net	Shifts/day	Shifts/1000 tons net	Shifts/day
	Setting props, timber, etc. Advancing supports & conveyors Transport, repairs, timbering Conveyor operators, etc. Other face work	2.7 20.3 0.6 14.1 2.1	2.7 20.1 0.6 14.0 2.1		
	Total face labour	82.2	81.3		
15	Panel labour			Not quite compatible due to different data sources.	
	On face Development Services (i) Installation & dismantling (ii) Transport (iii) Maintenance (iv) Supplies (v) Safety and other			83.0 58.8 11.8 20.9 31.4 32.2 18.8	82.2 57.9 11.7 20.7 31.1 31.9 18.6
	Total services to panel			115.4	114.1
	Total panel labour			257.2	254.2
16	Fraction $\frac{\text{Face labour}}{\text{Panel labour}}$	0.32		0.32	

TABLE 3.3 (continued)

	ITEM	FACE		PANEL	
D	OPERATING COSTS				
17	Labour costs	Francs/ton net	$/ton net	Francs/ton net	$/ton net
	Salaries - underground workers	6.84	1.368	21.4	4.280
	Additional emoluments	0.79	0.158	2.47	0.494
	Bonus on results	1.42	0.284	4.45	0.890
	Fringe benefits	7.04	1.408	22.0	4.40
	Injuries, absenteeism, etc.	0.23	0.046	0.72	0.144
	Total labour costs	16.32	3.26	51.0	10.20
18	Supply costs				
	Timber	1.09	0.218	1.53	0.306
	Metal arches, friction props, etc.	-	-	1.03	0.206
	Self advancing supports	0.99	0.198	0.99	0.198
	Explosives	0.31	0.062	1.04	0.208
	Dismantling and loading	0.04	0.008	0.04	0.008
	Conveyors, etc.	0.88	0.176	1.75	0.350
	Monorail, etc.	-	-	0.18	0.036
	Electrical supplies	0.09	0.018	0.18	0.036
	Others	1.18	0.236	2.36	0.472
	Total supply costs	4.58	0.92	9.16	1.83

TABLE 3.3 (continued)

	ITEM	FACE		PANEL	
19	Rental costs	Francs/ton net	$/ton net	Francs/ton net	$/ton net
	Self advancing supports	5.65	1.130	5.65	1.13
	Dismantling & loading	0.13	0.026	0.13	0.026
	Conveyors	1.12	0.224	2.42	0.484
	Monorail	-	-	0.08	0.016
	Special electrical materials	0.59	0.113	1.18	0.236
	Others	0.18	0.036	0.36	0.072
20	Maintenance costs				
	Self advancing supports	0.10	0.020	0.10	0.020
	Conveyors	0.08	0.016	0.08	0.016
	Other	0.08	0.016	0.15	0.030
	Contracted maintenance	0.01	0.002	0.02	0.004
	Total maintenance costs	0.27	0.054	0.37	0.074
21	Total operating costs	28.84	5.77	70.37	14.07
22	% Labour costs to total costs	56.6%		72.5%	

5. 950 m of developed roadway; steel arches on 0.5 m spacing.

1900 arches at F450/arch =	855,000 F*	$171,000*
Total panel (including face)	4,491,340 F	$898,268

Capital cost/annual ton net/panel (990 t/day, 250 days) =
18.1 F/ton net $3.63/ton net

* Author's estimate.

CHAPTER 4. THE ROZELAY MINE

4.1 Geology

Figure 4.1 shows a plan of the Rozelay mine together with the layout of current and proposed mining panels. Panels A2 and A3 (S3b) are currently being mined, the remaining panels are proposed. Panel A1, not shown, parallel to panel A2 is mined out. Figure 4.2 shows a section through the seams (section AA' Figure 4.1).

It is seen from these figures that the mine area is limited by the Rozelay, Permienne and Porrots Faults. The area bounded by level 320, panel A5 and the ends of panels 2, 3 and 4 is barren; the coal having been washed out. In this mine, seam No. 2 is currently being mined at a depth of cover of approximately 300 m. The seam varies in thickness from about 2.5 m to about 12 m. An important factor in this seam is the presence of a thick band of relatively competent sandstone in the middle of the seam; the position of the band in the seam varies relative to the footwall and likewise its thickness varies from 0.5 m to over 3 m. On average the thickness of this band is 2.17 m and on average is 4.57 m above the footwall. Thus about 5.26 m of coal overlies the sandstone band. Hence the sandstone band must be caved and drawn in order to liberate the overlying coal.

4.2 The Mine Layout

4.2.1 Panel layout

Figure 4.1 shows the layout of the current and proposed panels lettered A2 to A6. Panel A1 parallel to panel A2 is not shown. The main transportation roads $Theo^t$ and $Theo^b$ run about 20 - 30 m beneath the coal seam roughly along the axis of the bowl. Consequently panels A2, A3 and A4 will retreat from the fault boundaries towards these roads. Panels A5 and A6 will in fact be double panels, retreating from each boundary towards the centre The average panel dimensions are approximately 380 m long by 125 m wide.

As in the Darcy mine the panels are laid out for retreat down dip, with a maximum inclination both parrallel and perpendicular to the face of about 15^o, giving a maximum true dip of 20^o as a limit.

Inclines of roughly 100 m long are driven into the seam from $Theo^b$. Total development required for each panel is therefore approximately 200 m in rock, plus 760 m in coal for the gate roads, plus 125 m for the face.

As a general rule no barrier pillars are left between panels; the gate road being driven immediately adjacent to the previously extracted area. Exceptions are the small barrier pillars left between panels A2 and A3 and between panels A1 and A2; this was due to changing the mining method. Panel A1 was mined with 3 single descending slices. Panel A2 had a top slice mined under the hangingwall and the remaining coal in the seam is now being mined using a footwall slice with caving and drawing. Panel A3 (also called S3b) is being mined completely by the footwall slice with caving and drawing; this

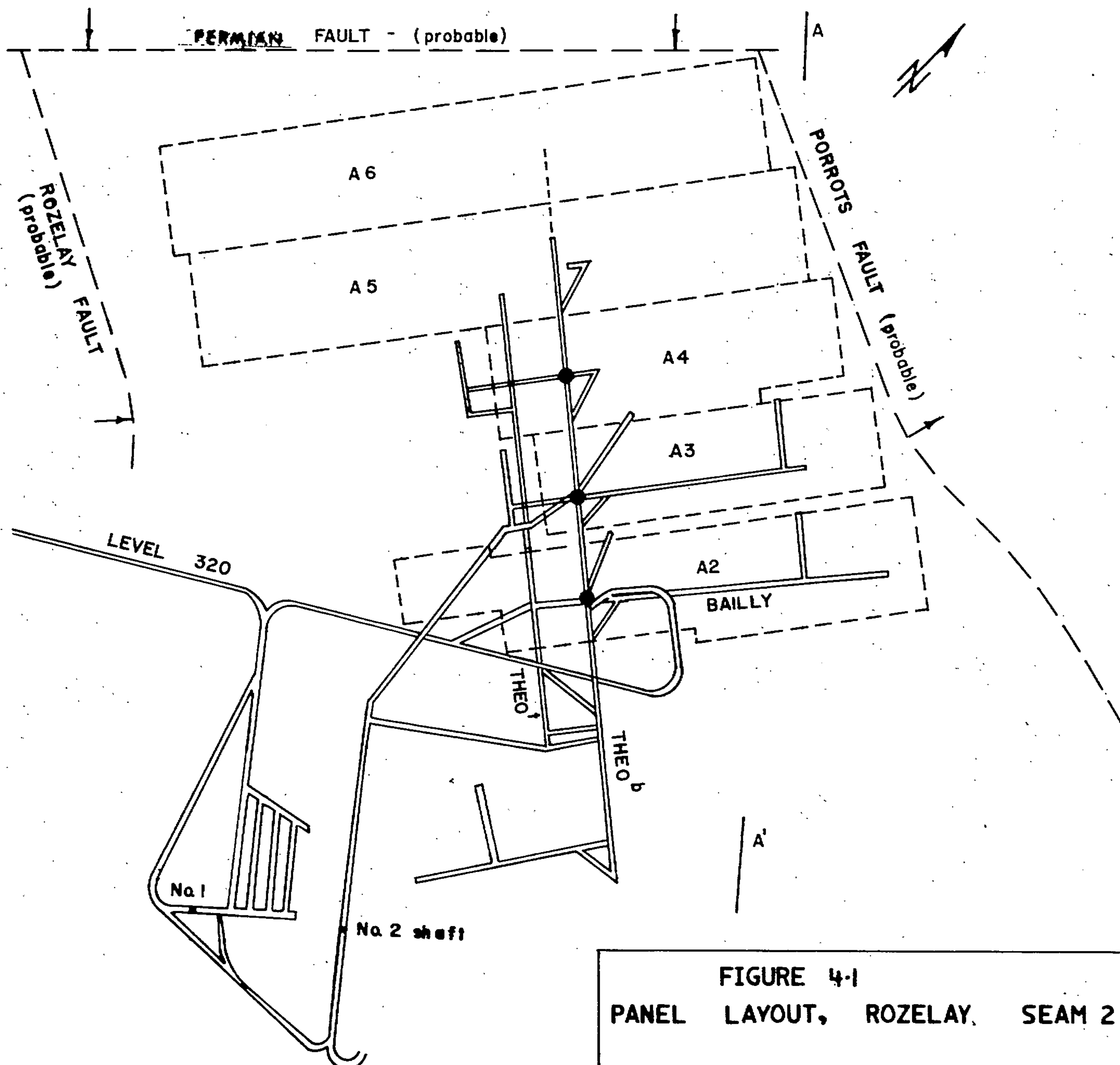


FIGURE 4·1
PANEL LAYOUT, ROZELAY SEAM 2

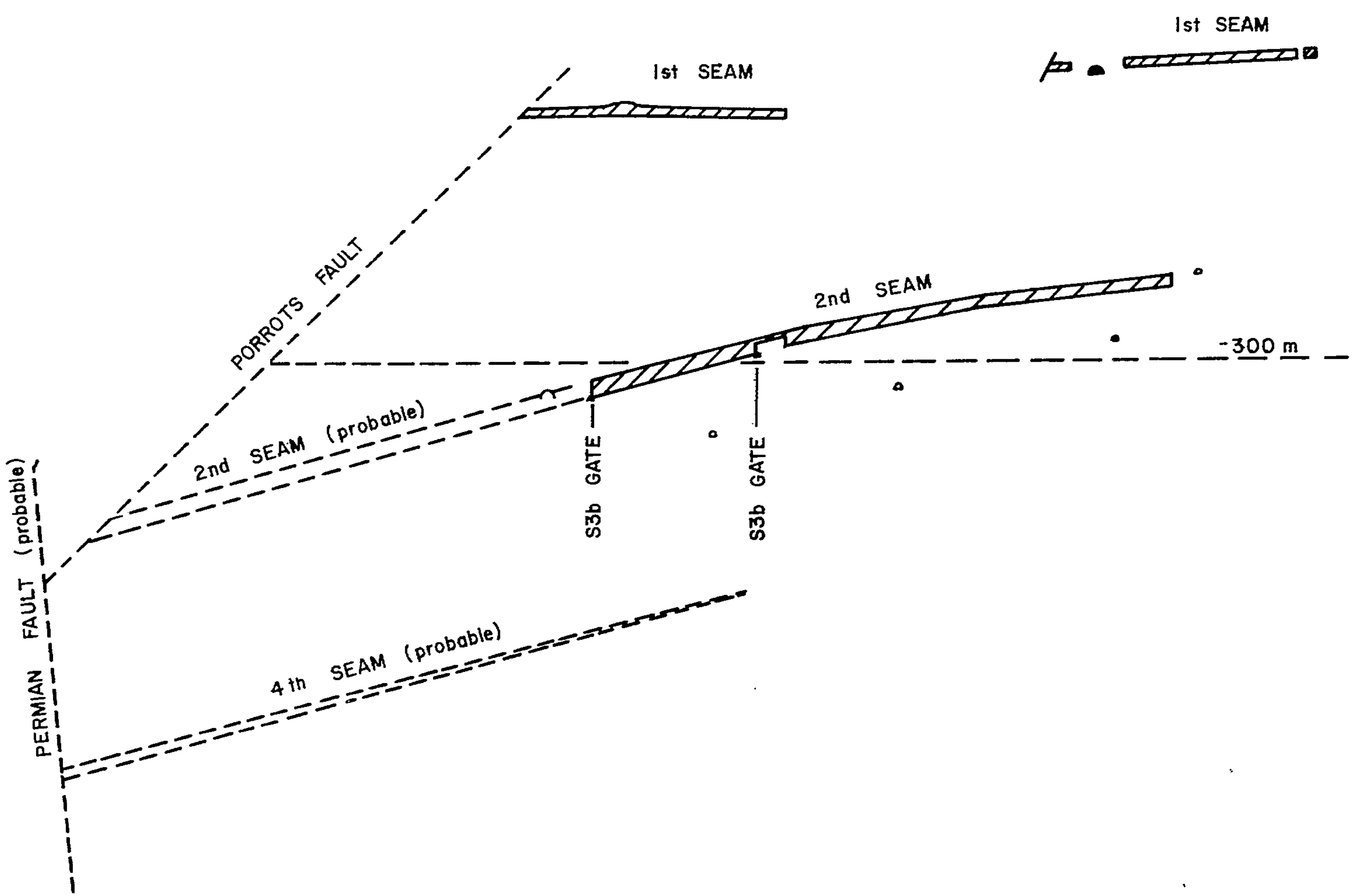


FIGURE 4-2 : CROSS SECTION A – A', ROZELAY MINE.

was the panel studied in this mine.

Figure 4.3 shows a detailed layout of panel S3b (A3); the panel is approximately 375 m long by 110 m wide. Figure 4.4 shows cross sections through the headgate, tailgate and face of this panel.

4.2.2 Transportation

(a) Coal discharge from the mine. The rear conveyor is a 50-cm wide chain conveyor powered by two 500-V, 48-kW motors; the conveyor speed is 50 cm/sec. The front conveyor, over which the shearer rides, is a Westfalia PF1 chain conveyor, 70 cm wide, powered by two 1000-V, 64-kW motors having a speed of 65 cm/sec.

The tailgate is equipped with a 50-cm wide chain conveyor, powered by two 500-V, 36-kW motors with a speed of 73 cm/sec. Coal from this tailgate chain conveyor is transferred at the junction with the Camille incline (see Figure 4.3) to a 1-m wide belt conveyor, speed 2.2 m/s, powered by two 500 V, 48-kW motors, whence it is transported to the storage chute No. 3.

All coal transport is restricted to return airways.

(b) Supplies. All supplies are delivered from the shaft to within 50 m of the face in the headgate by means of a diesel powered monorail locomotive. Appendix 3 gives details of this monorail system. The supply transportation route is marked in Figure 4.3. In the headgate, 50 m from the face, a 35-cm wide chain conveyor equipped with 24-kW motors is used to transfer supplies to the face.

4.2.3 Ventilation

The ventilation circuit for panel S3b is marked in Figure 4.3. A typical ventilation volume on the face is 11,500 cu m/minute at 19°C. As in the Darcy mine, the No. 2 seam at Rozelay is not gassy, the volume of gas produced being approximately 2 m^3/ton; gas therefore does not pose ventilation problems.

4.2.4 Previous mining areas

In addition to the previously mentioned mining of panel A1, and the current mining in panels A2 and A3 in the No. 2 seam, extensive areas of the No. 1 seam have been previously mined out. The No. 1 seam lies approximately 100 m above seam No. 2. Figure 4.5 shows these areas of previous mining in No. 1 seam in relation to the current and planned panel layout in No. 2 seam.

4.2.5 Development

As in the Darcy mine, the main haulage and supply roads are driven in the underlying footwall rocks about 20 - 30 m below the seam. Inclines are then driven up to the seam and the panel is developed by driving the gate roads within the coal.

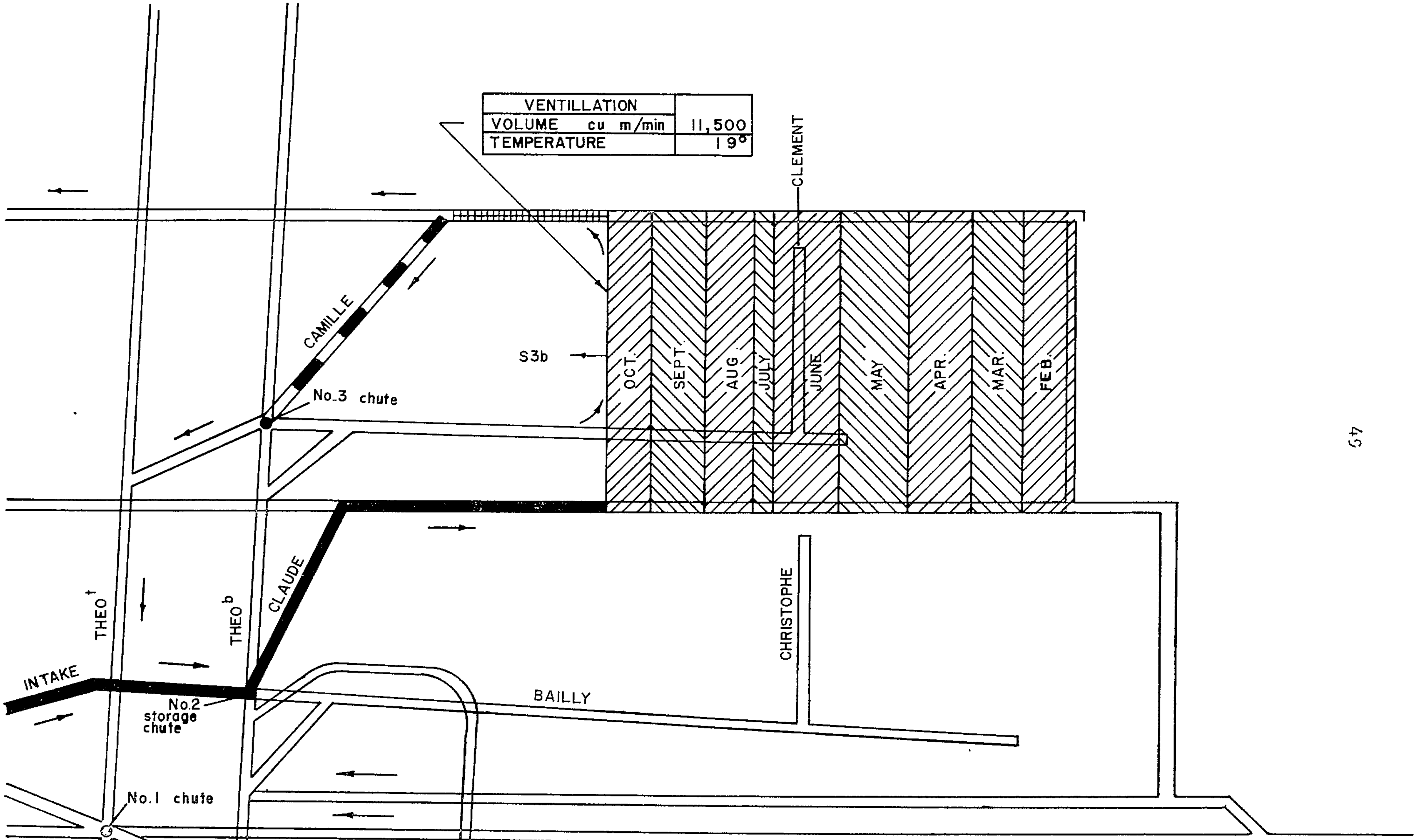


FIGURE 4·3 : LAYOUT OF PANEL S3b, SEAM 2, ROZELAY.

MONTHLY FACE ADVANCE
BELT CONVEYORS
MONORAIL SUPPLIES
CHAIN CONVEYORS
VENTILLATION CIRCUIT

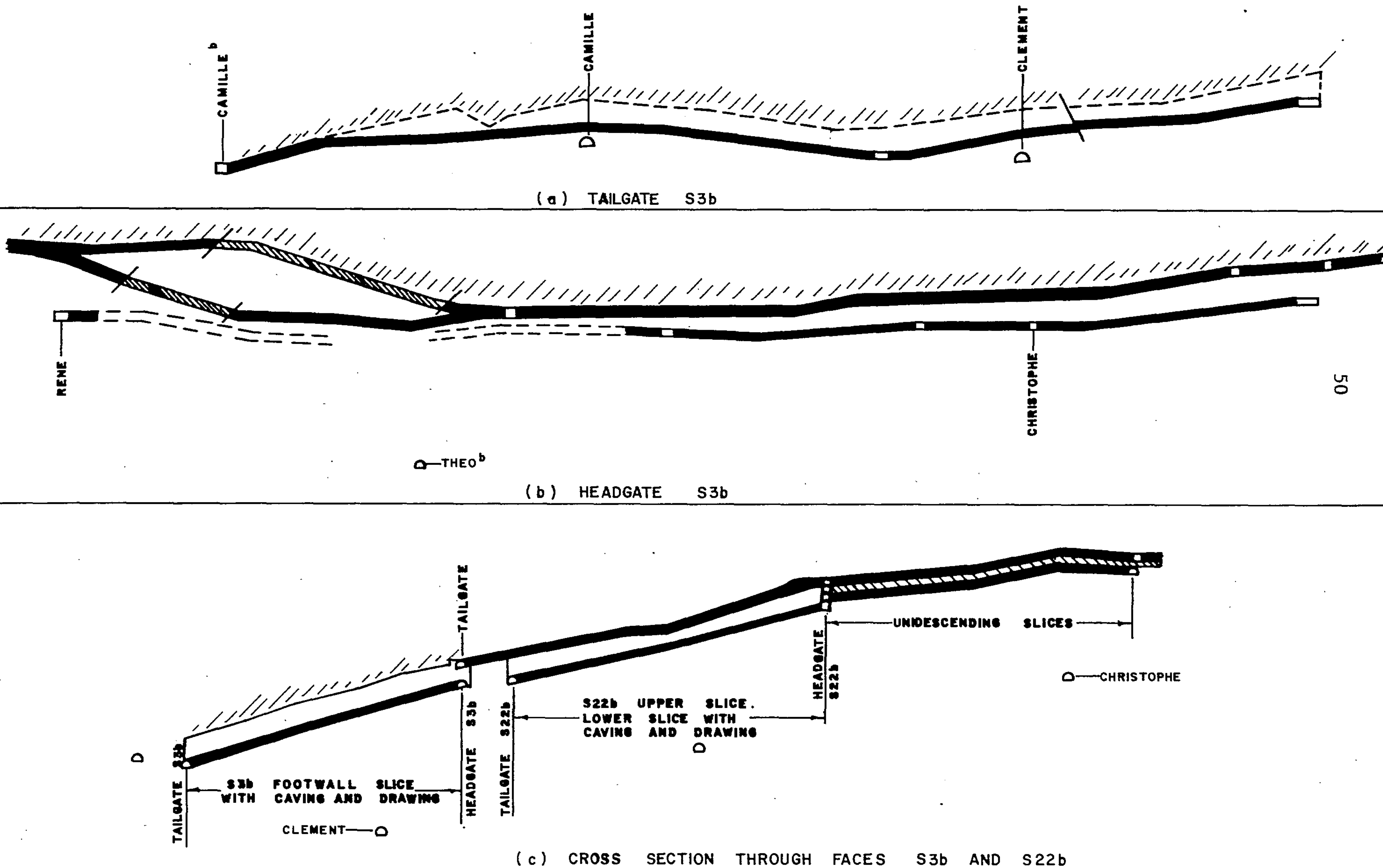


FIGURE 4.4 : CROSS SECTIONS THROUGH HEADGATE, TAILGATE AND FACE OF S3b, SEAM 2, ROZELAY MINE

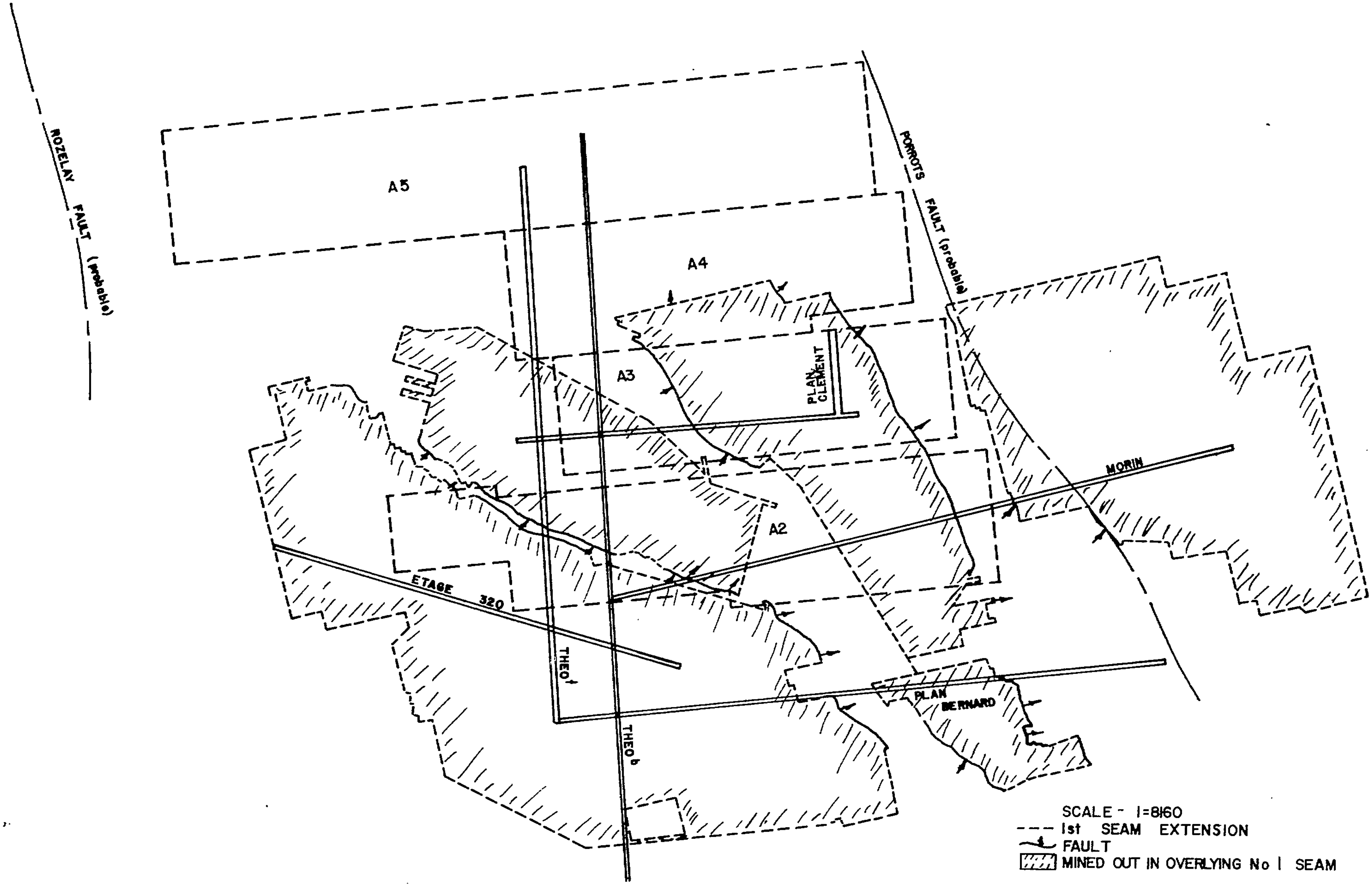


FIGURE 4 5 : PREVIOUS MINING AT ROZELAY MINE

For this study the panel is deemed to include the face, the gate roads, the rock inclines and the haulage roads as far as the storage chutes. In Rozelay mine the average block of coal to be mined by a panel is approximately 380 m long by 125 m wide; this necessitates the development of approximately 885 m of roadway in coal. The amount of rock development within the panel is difficult to estimate, but it is of the order of 300 m per panel. (Note the figure calculated in Appendix 9 appears to be excessive and is probably in error.)

The development headings are driven using Alpine continuous miners; Figures 4.6 (a) and (b) show photographs of the machine. A description of the miner and its specifications is given in Appendix 7. This machine is suitable for development in coal but not in rock. Support for the roadways is provided by 5 element TH 470 steel arches set approximately 1 m apart in rock and 0.5 m apart in coal; timber lagging is placed behind the rings.

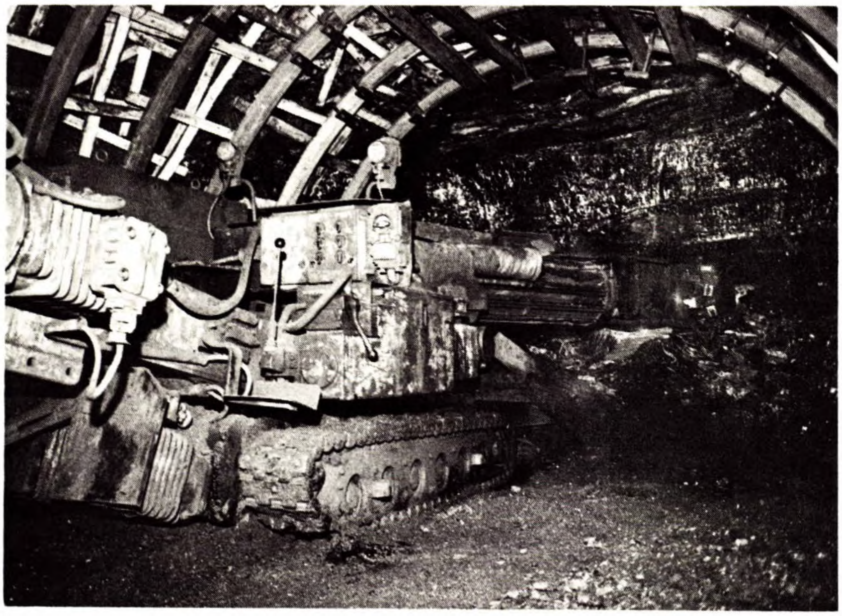
The average daily advance rate of development in the Rozelay mine over a 3 month period in 1972 was 5.86 m/day, or for an "average panel" 5.71 m/day at an average productivity of 18.33 cm/man shift (see Appendix 9). In comparison with the Darcy mine where hand advance methods are used, the productivity with the Alpine miner is 4.7 times better. According to the calculations in Appendix 9 the cost/metre of development is approximately 1544 F/m (309 $/m); however, it is not known how reliable this figure may be.

4.2.6 Setting up the face

Dismantling of equipment from one face and the setting up of equipment on a new face requires careful planning if continuity of production is to be maintained as far as possible. The new face should be developed and supported by timber and friction props prior to commencement of the changeover. P.E.R.T. (Program evaluation and review techniques) assists considerably in planning the logistics of the changeover. At the Rozelay mine the face S12b was dismantled and the equipment was set up in face S3b. Figure 4.7 shows the transportation routes selected for this changeover. The powered supports were transferred from the tailgate of face S12b to the headgate of S3b where they were reassembled and placed on the face. The shearer was transferred from the tailgate of S12b to the tailgate of S3b. All the equipment was transferred using the monorail system. The logistics of these moves are very important in order to prevent a "pile up" of equipment in the head and tailgates of the new face. In Appendix 7 a chart showing the detailed planning program and a comparison with the realized program is given. The original plan envisaged a total of 1132 man shifts being required to completely dismantle face S12b and to install face S3b; in practice a total of 1571 man shifts were required.

The major problem encountered in this changeover was the dismantling and removal of the powered supports on face S12b. The conditions on this face were poor and the face had converged; this lack of headroom made the dismantling of the supports difficult and retarded the project.

(a)



(b)

FIGURE 4·6 : ALPINE CONTINUOUS MINER USED FOR DEVELOPMENT HEADINGS IN ROZELAY MINE

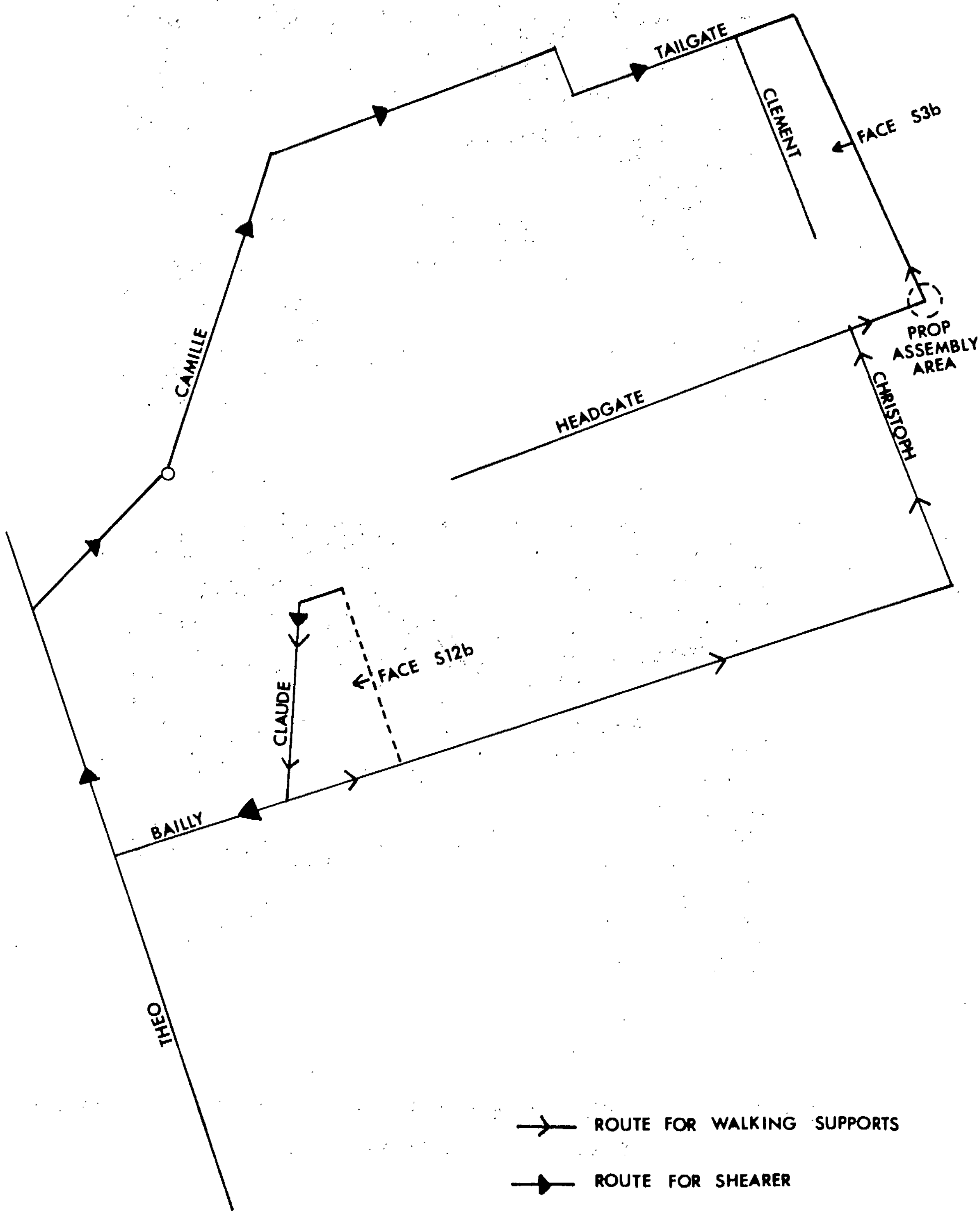


FIGURE 4.7: TRANSPORTATION ROUTES FOR TRANSER OF EQUIPMENT FROM FACE S12b TO FACE S3b

The supports were completely reassembled in the headgate of S3b where the headroom was good. This assembly was done on a special steel working platform which formed a "sledge". The sledge was winched down the face, supports being installed at the tailgate end first, working back towards the headgate. The roof over the support being installed was supported by another powered support aligned parallel to the face; this support was retreated back towards the headgate as each support was installed.

A series of photographs in Appendix 8 shows the sequence of events during dismantling and assembling of the powered supports.

4.3 The Mining Method

The basic mining method - longwall bottom slice with caving and drawing of the top coal - is the same in the Rozelay mine as that previously described for the Darcy mine. However, at Rozelay, mechanization has been taken one step further with the introduction of the double drum shearer for advancing the face.

4.3.1 The Mining Cycle

Phase 1 In Figure 3.9, phase 1, the face is shown in the closed position with a minimum face width of approximately 5 m. The front conveyor, over which the shearer runs, is as far forward as possible, immediately against the face. The rear drawing conveyor is as close to the rear supports as possible and is protected by the banana prop.

Phase 2 The coal face is advanced by the drum shearer running up the face, taking a slice of approximately 0.5 m off the face. The face conveyor is snaked up to the face, after passage of the shearer, using the hydraulic pushing rams on the powered supports. The powered supports are advanced as soon as possible behind the machine. It is recommended that the machine not be allowed to get more than 10 supports ahead of the advancing supports. As before, lateral timbers and wire mesh are placed over the top of the supports.

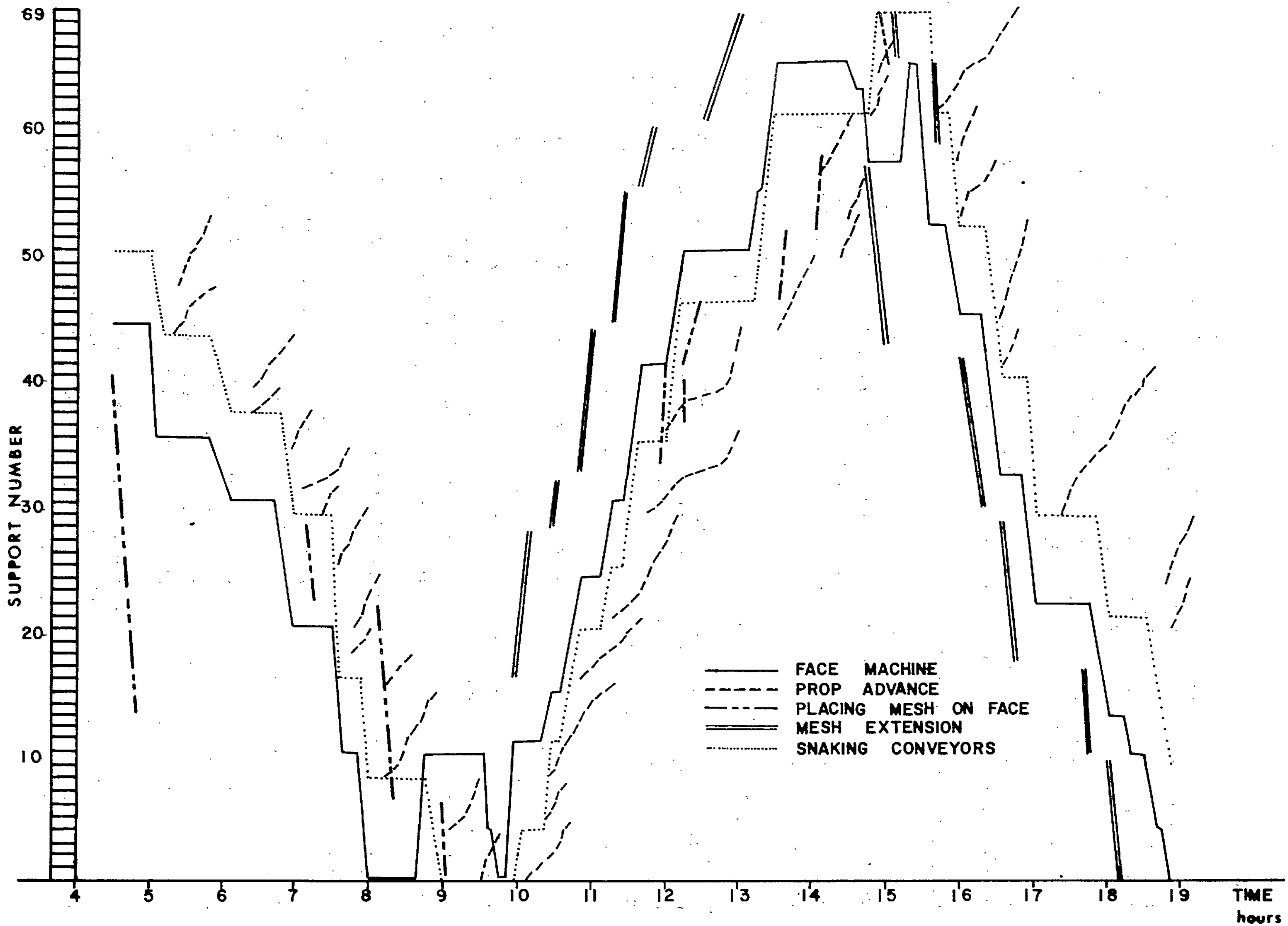
Phase 3 The rear conveyor is snaked forward behind the advanced supports.

Phase 4 Caving and drawing of the coal onto the rear conveyor is commenced. The wire mesh is laid out on the face and attached to that over the props in preparation for the next passage of the machine and the next support advance.

The above sequence of operations is carried out during the morning and afternoon shifts. The night shift is concerned solely with maintenance and clean up of the face. In addition, during the night shift the face is bolted with wooden bolts (about 1 m long) to prevent spalling of the face.

Figure 4.8 shows the face advance cycle over two shifts observed in May 1972. It can be seen from this graph that the shearer spends a considerable portion of time waiting for the prop advance to catch up before it can resume cutting the face. This will be discussed in more detail later.

FIGURE 4·8 ROZELAY FACE S3b - SHIFTS 1&2 - 16 MAY 1972



4.3.2 Face equipment

The face equipment is basically the same as that in the Darcy mine, with the addition of the shearer. This was a Sagem DTS 300 double drum shearer, see Figure 4.9. Appendix 10 gives specifications together with additional photographs.

4.4 Good and Bad Mining Practice

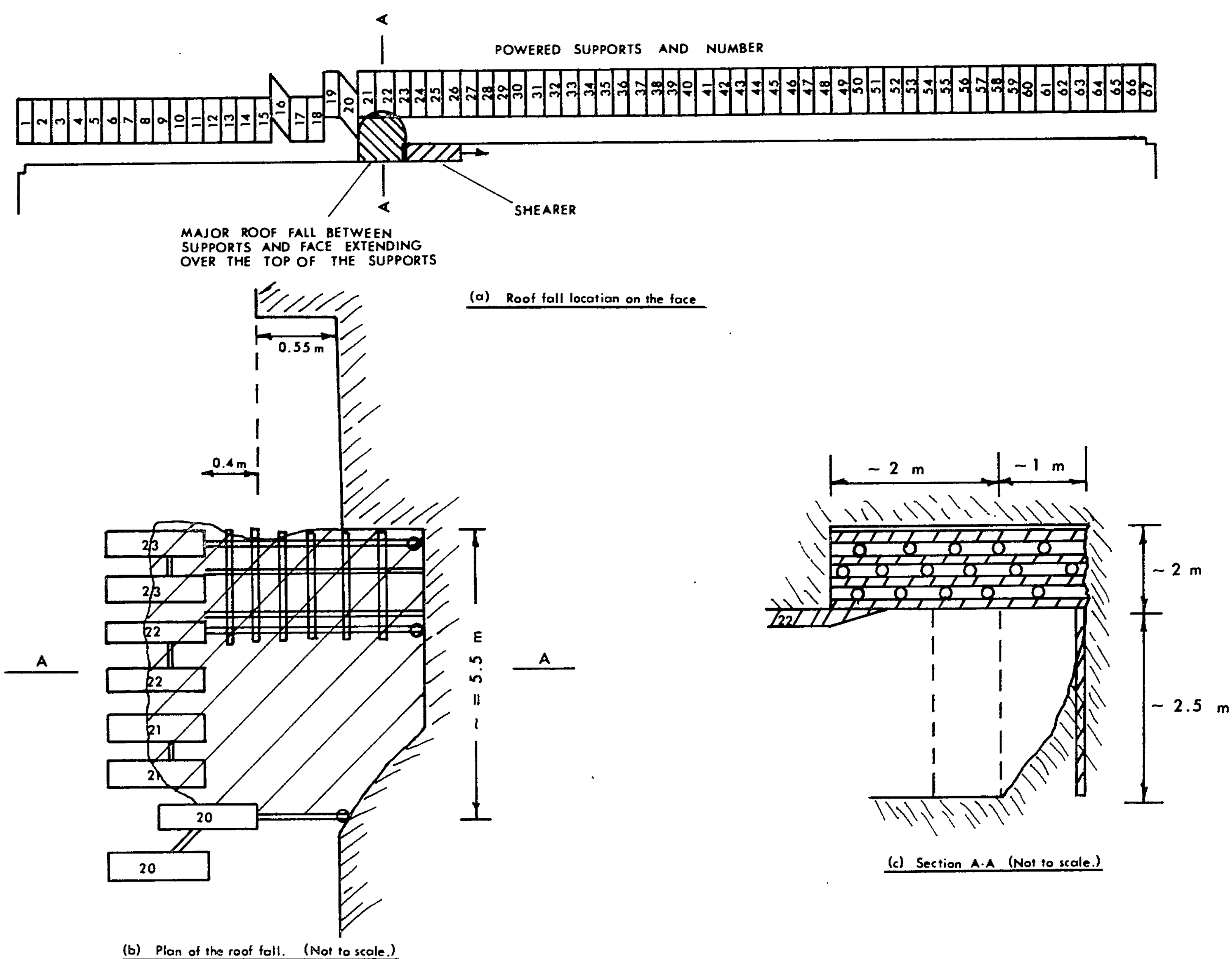
The basic do's and dont's set out in Section 3.4 for the Darcy mine apply equally well to this mine. The use of the shearer and the specific properties of this seam add to this list. Some of the following points are general to the method, other problems were observed on study shifts and the frequency of their occurrence is not known.

1. The shearer is capable of cutting along the face at a faster rate than the powered supports can be advanced behind it. The machine must not be allowed to get too far ahead of the support advance otherwise the excessive extent of unsupported roof between the front of the support canopies and the face may lead to caving ahead of the supports. In this event serious delays are experienced whilst the face is rehabilitated. It is recommended that the machine should not be more than 10 supports (~ 15 m) ahead of the last support advanced (although the author observed on one shift the machine over 40 supports in advance at one stage).

2. The author observed one cave on the face, between the props and the face. This is sketched in Figure 4.10. The coal caved ahead of the face for about one metre and extended back over the props for about the same distance; the length of the cave along the face was between 5 and 6 metres. The coal caved upwards for a height of about 2 metres where the cave was stopped by the presence of the thick sandstone bed in the middle of the seam. In the author's opinion, had this sandstone bed not been present this caving would have continued for the full seam height considerably complicating the rehabilitation process. This cave occurred towards the end of the afternoon shift and it was left for the night maintenance shift to complete the installation of the timber supports. The caved area was filled with a lattice work of timber supported at one end by the canopies of the powered supports and at the other end by friction props tight against the face. The filling of this extensive caved area was not only time consuming (5 men and a shift boss for approximately 4 hours) but also required one man working above the supports placing the timber; this seemed to be an exceedingly hazardous procedure during the initial placement of timbers.

   When the caved area had been fully timbered the face was then advanced, by hand for about 1 metre, over the width of the caved area. This was done so that a new coal roof would be established and so that, when passing this region on the next two cuts, the shearer would not have to cut the coal and possibly reintroduce caving due to the ensuing vibration.

FIGURE 4·9 : SAGEM DTS 300 DOUBLE DRUM SHEARER



(a) Roof fall location on the face

(b) Plan of the roof fall. (Not to scale.)

(c) Section A-A (Not to scale.)

FIGURE 4.10: ROOF FALL OBSERVED ON FACE S3b BETWEEN SUPPORTS AND FACE

Although no records of face falls were available, and thus their frequency of occurrence is not known, conversation with the shift boss indicated that face falls could occur 2 - 3 times each week. If this is so, then it is a major contributory reason why there are only two production shifts per day with the other shift being solely for maintenance. In the opinion of the shift boss, extension pieces on the front of the canopies (as in the Westphalia high bearing capacity supports used in the Darcy mine, and illustrated in Appendix Figure A4.3) are not a solution to this problem. It is essential to keep the support pressures as even as possible to help steer clear of this problem.

3. Caving of the roof between the supports and the face can also result from spalling of the face. If a large slab of coal spalls off the face then an additional area of roof is exposed which, in the normal course of events would not get supported until one or more passes of the machine later. Consequently if a significantly thick slab spalls off the face, this area is also advanced by hand and supported with friction props and timber. In order to minimize spalling of the face, two men on the night shift bolt the face with wooden plugs. Three holes across the vertical section of the face are drilled with an air pick every metre along the face, to a depth of about 1½ m. Wooden stakes are then inserted in these holes and are hammered into the coal.

4. During caving and drawing of the coal, certain problems probably specific to this seam were observed. The presence of the sandstone bed in the middle of the seam resulted in delays in the caving and drawing operations. This sandstone bed frequently caved in large blocks (up to 1 m^3) which would not pass through the drawing windows. It was then necessary to stop the rear conveyor and to break up these blocks with an air pick so that they could pass through the draw point.

   Even after passing through the draw point these blocks could cause a blockage on the rear conveyor and thus two men were continuously employed breaking up lumps of rock on the rear conveyor, using air picks.

5. The author was told that, on occasion, the sandstone bed would hang up. In this case, holes were drilled upwards between the rear of the supports and light charges used to induce caving.

4.5 Manpower Distribution

Manpower distribution on the face and panel will vary depending on both mining conditions and on absenteeism. Each shift is 8 hours long, from pit head to pit head; representing approximately 7 hours panel working time when transportation time is taken into account. Morning and afternoon shifts are production shifts and the night shift is a maintenance shift.

4.5.1 Manpower distribution in the panel

Inadvertently a complete breakdown of the manpower distribution in the panel by job was not recorded. However Table 4.1 below gives the number of shifts per day (averaged over a 6 month period) in the panel and indicates the distribution of this labour between the face, development and panel services.

TABLE 4.1

Manpower Distribution in the Panel

Region of mine	Total shifts per day
On the face	78.8 (average over 3 months)
In development headings	29.5 (average over 3 months)
On panel services	64.3 (average over 3 months)
Total	180.9 (average over 6 months)

4.5.2 Manpower distribution in the face

Table 4.2 gives a typical manpower distribution on face S3b, by job description for each shift. These figures were recorded during study shifts on this face.

TABLE 4.2

Typical Manpower Distribution on Face S3b

Job description	Shift 1	Shift 2	Shift 3	Total
Advancing of supports	6	6	-	12
Caving and drawing	5	5	-	10
Machine operators (shearer)	2	2	-	4
Snaking of front conveyor	2	2	-	4
Snaking of rear conveyor	2	2	-	4
Conveyor operators	2	2	1	5
Formation of headgate niche	1	2	-	3
Formation of tailgate niche	2	1	-	3
Shot firers	1	1	-	2
Hydraulic technicians	3	3	-	6
Headgate - resetting of timbers	2	2	-	4
Tailgate - resetting of timbers	2	2	-	4
Mechanics, moving conveyor motors, short caving tailgate conveyor	-	-	3	3

TABLE 4.2 (continued)

Job description	Shift 1	Shift 2	Shift 3	Total
Supply men	-	-	3	3
Bolting the face (wooden bolts)	-	-	2	2
General maintenance	-	-	5	5
Machine maintenance (shearer)	-	-	1	1
Shift firemen	2	2	2	6
TOTAL	32	32	17	81

Appendix 9 gives the average number of face workers for a period of 3 months; this average of 78.8 shifts per day on the face is slightly lower than that for the particular shifts studied above. However the two figures are sufficiently close to indicate that the job description distribution given above may be taken as typical.

4.6 Production, Productivity and Operating Costs for an Average Face and Panel

Appendix 9 gives production, productivity, operating costs and development statistics for the panel 3 containing face S3b in the Rozelay mine over a period of 6 months. From this data the requisite statistics for an "average panel" have been derived and are given in Table 4.3 overleaf.

The following important points are also shown in the analysis given in Appendix 9.

(a) The percentage seam recovery from the panel in this seam averages 67.1%. This figure is considerably less than that in the slightly thinner Darcy seam (95%). This difference is attributed to the presence of the sandstone bed in the middle of the Rozelay seam. In the volume calculations this bed has been counted as coal; in practice a large amount of the rock volume is not drawn from behind the face as its higher density tends to drop it preferentially to the bottom of the caved coal; thus careful drawing can prevent the drawing of a great deal of this rock. However, in doing this a certain amount of coal also gets left behind. In the author's opinion, if this sandstone bed were not present in the middle of the coal seam then a percentage recovery approaching the 95% achieved in Darcy mine would also be achieved here.

(b) There is an excellent correlation between productivity and the rate of face advance (unlike the Darcy mine case). This correlation in this case is attributed to the fact that the rate of face advance depends on the number of passes along the face made by the machine; this in turn depends primarily on conditions and not so much on labour; whereas in the Darcy mine an increased face advance rate (if possible) requires the direct application of more face labour to hand mine the face.

TABLE 4.3

Statistics for an Average Face and Panel in Rozelay Mine

		FACE		PANEL	
A	DIMENSIONS				
1	Face length - metres	109			
2	Face height - metres	2.8			
3	Seam thickness - metres	12.25			
4	Monthly face advance - metres	20.3			
5	Mean daily face advance	1.05			
6	No. passes/day of shearer along face	1.92			
B	PRODUCTION AND PRODUCTIVITY				
7	Percentage seam recovery	67.1		67.1	
8	Tons net/tons gross, %	75.7		75.7	
9	Monthly production, tons/net	20,180		21,829	
10	Mean daily production, tons/net	1,035		1,110	
11	Productivity, tons net/man shift	13.0		6.13	
12	Monthly production, tons gross	26,720		28,941	
13	Mean daily production, tons gross	1,368		1,471	
14	Productivity, tons gross/man shift	16.7		8.13	
C	LABOUR				
15	Face labour	Shifts/1000 t net	Shifts/day	Shifts/1000 t net	Shifts/day
	Advancing supports		12		
	Caving and drawing		10		
	Machine operators		4		
	Snaking front & rear conveyors		8		
	Conveyor operators		5		
	Headgate & tailgate niche formation		6		
	Shot firers		2		
	Hydraulic technicians		3		

TABLE 4.3 (continued)

		FACE		PANEL	
19	Supply costs	F/ton net	$/ton net	F/ton net	$/ton net
	Timber	0.87	0.17	1.25	0.25
	Supports (arches & friction props)	-	-	0.45	0.09
	Walking supports	1.64	0.33	1.64	0.33
	Explosives	-	-	0.06	0.01
	Dismantling and loading	0.77	0.15	0.77	0.15
	Conveyors, etc.	0.36	0.07	0.72	0.14
	Monorail	-	-	0.02	0.004
	Electrical	0.23	0.05	0.46	0.092
	Others	0.64	0.13	1.28	0.25
	Total supply costs	4.51	0.90	6.65	1.33
20	Rental costs				
	Walking props	5.89	1.18	5.89	1.18
	Dismantling & loading	1.93	0.39	1.93	0.39
	Conveyors	0.60	0.12	1.20	0.24
	Monorail	-	-	0.06	0.01
	Electrical	0.37	0.07	0.74	0.15
	Others	0.10	0.02	0.21	0.04
	Total rentals	8.89	1.18	10.03	2.00
21	Maintenance costs				
	Supports (arches & friction props)	-	-	0.04	0.008
	Walking props	0.03	0.006	0.03	0.006
	Dismantling & loading	0.01	0.002	0.02	0.004
	Conveyors	0.03	0.006	0.05	0.010
	Monorail	-	-	-	-
	Electrical	0.06	0.012	0.13	0.026
	Other	0.04	0.008	0.08	0.016
	Total maintenance	0.17	0.034	0.35	0.070
22	Total operating costs	26.50	5.30	46.67	9.33
23	% labour costs of total costs	48.8%		63.5%	

TABLE 4.3 (continued)

		FACE		PANEL	
		Shifts/1000 t net	Shifts/day	Shifts/1000 t net	Shifts/day
	Headgate & tailgate timbering Mechanics Supply men Face bolting Machine maintenance Shift foreman		8 3 3 2 1 6		
	Total face labour		81		
16	Panel labour (a) on face (b) development (c) panel services		Not quite compatible due to different data sources.	84.9 31.5 69.8	78.8 29.5 64.3
	Total panel labour			163.0	180.9
17	Ratio face labour/panel labour	.436		.436	
D	COSTS	F/ton net	$/ton net	F/ton net	$/ton net
18	Labour costs Salaries underground workers Additional emoluments Bonus in results Fringe benefits Injuries & absenteeism	 5.31 0.65 1.11 5.52 0.34	 1.06 0.13 0.22 1.10 0.07	 12.18 1.48 2.56 12.67 0.79	 2.44 0.29 0.51 2.53 0.16
	Total labour costs	12.93	2.59	29.64	5.93

In the Rozelay mine there was no apparent correlation between productivity and seam thickness, whereas there was a very good correlation in the case of the Darcy mine. This was surprising as there appears to be no basic differences between the two mines which would account for this difference. However, the data upon which the attempted correlation was based for the Rozelay mine, in fact, contained little variation in seam thickness; this possibly accounts for why a correlation was not established in this case. Intuitively it would be expected that the productivity would be strongly dependent on the seam thickness (as was shown for the Darcy mine) and in the author's opinion such a correlation probably also exists in Rozelay but the lack of variation in the available date precluded establishing this beyond doubt.

(c) The average face is 109 metres long, the average seam thickness is 12.25 metres. The machine averaged 1.92 passes/day along the face of mean height 2.8 m and mean thickness of cut of 0.55 m. The average daily production was 1035 tons net (1110 tons gross) with a face productivity of 13.0 tons net/man shift (16.7 tons gross/man shift and a panel productivity of 6.13 tons net/man shift (8.13 tons gross/man shift).

(d) The total face operating costs are 26.5 F/ton net ($5.30/ton net) and the panel operating costs are 46.67 F/ton net ($9.33/ton net).

(e) Labour accounts for 48.8% of the face operating costs and 63.5% of the panel costs.

It is interesting to note that the face productivity and face costs/ton net are almost identical to those of the Darcy mine where the face is advanced by hand. Since Rozelay is more mechanized and thus intuitively a better productivity and lower cost per ton might be anticipated, it must raise the question as to whether the shearer is being used on this face to its maximum efficiency. This will be discussed in section 4.8. The panel productivity and costs for the Rozelay mine are much better than for the Darcy mine due mainly to the much lower requirement in Rozelay for panel service personnel; the reason for this lower demand for panel services is possibly due, at least in part, to a greater concentration of operations in the Rozelay mine.

4.7 Major Capital Costs for an Average Face and Panel

4.7.1 Face

Mean length 109 m, requiring 70 self advancing supports; assume face equipped with SMF supports.

1.	70 supports at F43,960	3,077,200 F	$615,440
2.	2 face conveyors at F450,000	900,000 F	$180,000
3.	Double drum shearer	600,000 F	$120,000
	Total face	4,577,200 F	$915,440

4.7.2 Panel

4.	2 gate road chain conveyors 50 m long. 100 m at F350/m	350,000 F	$ 70,000
5.	425 metres belt conveyors in gate roads at F500/m	212,500	$ 42,500
6.	1185 m of developed roadway; steel arches 0.5 m spacing 2370 arches at F450 each	1,066,500 F	$ 213,300
	Total panel (including face)	6,206,200 F	$1,241,240

Capital cost/annual ton net/panel (1035 t net/day, 250 days) is 23.98 F/annual ton net ($4.79/annual ton net).

4.8 Analysis of the Mining Cycle

Figure 4.8 shows a breakdown of the working cycle for face advance during two shifts studied by the mine; Figure A11.1 in Appendix 1 shows a similar breakdown of the working cycle based on study shifts carried out by the author. In Appendix 11 the time spent on various face jobs during these shifts has been analyzed and the following important points become apparent.

(i) The shearer is only operating for between 20 - 27% of the available working time; the remainder of this time is mostly taken up with the machine sitting idle waiting for the crews advancing the supports to catch up, i.e. the machine is capable of cutting the face at a much faster rate than the support crews (3 two-man crews) are able to advance the supports behind the shearer. On average it takes about 7 minutes for a two-man crew to advance each support.

(ii) It was also observed that at no time was the caving and drawing of the coal unfinished at the end of the two production shifts.

The "bottleneck" is not the drawing system but in the rate at which the powered supports can be advanced. This is contrary to what has been previously reported (3).

The analysis of the actual operating shifts shows that a face advance of 2.4 machine passes per day (0.55 m advance per pass) was achieved; this is higher than the 1.92 passes per day averaged over six months at the Rozelay mine, indicating that the shifts studied were somewhat more efficient than the average. Based on this analysis an attempt has been made in Appendix 11 to estimate the maximum rate of face advance for a 100 m face. This was done by assuming that additional prop advance crews were put on to the face so that the shearer would be in use 100% of the available time (less turn round time at the end of the face). Additional personnel was also added to the face and panel crews to cope with the extra caving and drawing, hydraulic maintenance and supplies, etc. required to deal with this extra face advance rate. On this basis a maximum of 4.75 passes per day was calculated. This figure was then scaled down by the ratio of $\frac{1.92}{2.4}$ of the observed and current average face advance rates to yield an "achievable" number of passes per day of 3.80. This corresponds to a shearer availability of approximately 75% which does not seem an unreasonable figure to expect.

The current average advance rate is thus 1.92 passes/day and it is believed that a rate of 3.8 passes/day could be achieved.

Appendix 11 then continues the analysis by calculating the production, the productivity and costs/gross ton for both the face and the panel for these current and achievable advance rates, taking into account the fact that extra personnel is required to reach the achievable situation and that other operating costs are also increased. It is also assumed that the production will be directly proportional to the seam thickness and the face advance, and that the number of men on the face is little influenced by the seam thickness. Using these assumptions the current and achievable daily production, productivity and costs for both the face and the panel have been calculated and related to seam thickness. The detailed calculations are given in Appendix 11. Figures 4.11 and 4.12 show the results of these calculations.

These graphs apply to a 100 m face under Rozelay mine conditions with 67.1% extraction and a clean/raw coal fraction of 0.757.

On both of these graphs a limit level of production has been placed. This assumes that with the face advance "bottleneck" removed, the next bottleneck is in the caving and drawing system and is dictated by the rate at which the rear conveyor can remove coal from the face. It is assumed that the rear conveyor has a 50% availability, i.e. that it cannot remove more than 250 tons/hour off the face (rated capacity 500 t/hour), i.e. 3500 tons per day with two production and one maintenance shift per day. This limit could be raised to 4375 tons per day, without changing the equipment, if it were feasible to cave and draw coal during the maintenance shift.

In the author's opinion it is doubtful if a higher production rate than 4000 tons per day from any one face would be advisable; consequently the extrapolation carried out in these graphs to seam thicknesses of up to 40 m is probably taking this analysis too far. However, it is thought that these graphs do indicate the possibility of this method and the range of production, productivity and cost that might be expected.

For example: From these graphs, the current figures for the 12.25 m seam at Rozelay mine are 1370 tons gross/day, 16.7 tons gross/man shift at a cost of 20.1 F/ton gross (26.5 F/net ton) for the face. It is thought that by increasing the face labour as indicated in Appendix 11 the following figures might be achieved for this face: 2700 tons gross/day, 24.6 tons gross/man shift at a cost of 13.8 F/ton gross (18.3 F/net ton).

The above figures apply to the face. The equivalent figures for the panel are:-

Current	1370 tons gross/day;	7.6 tons gross/man shift;	35.30 F/ton gross
Achievable	2700 tons gross/day;	10.8 tons gross/man shift;	24.7 F/ton gross

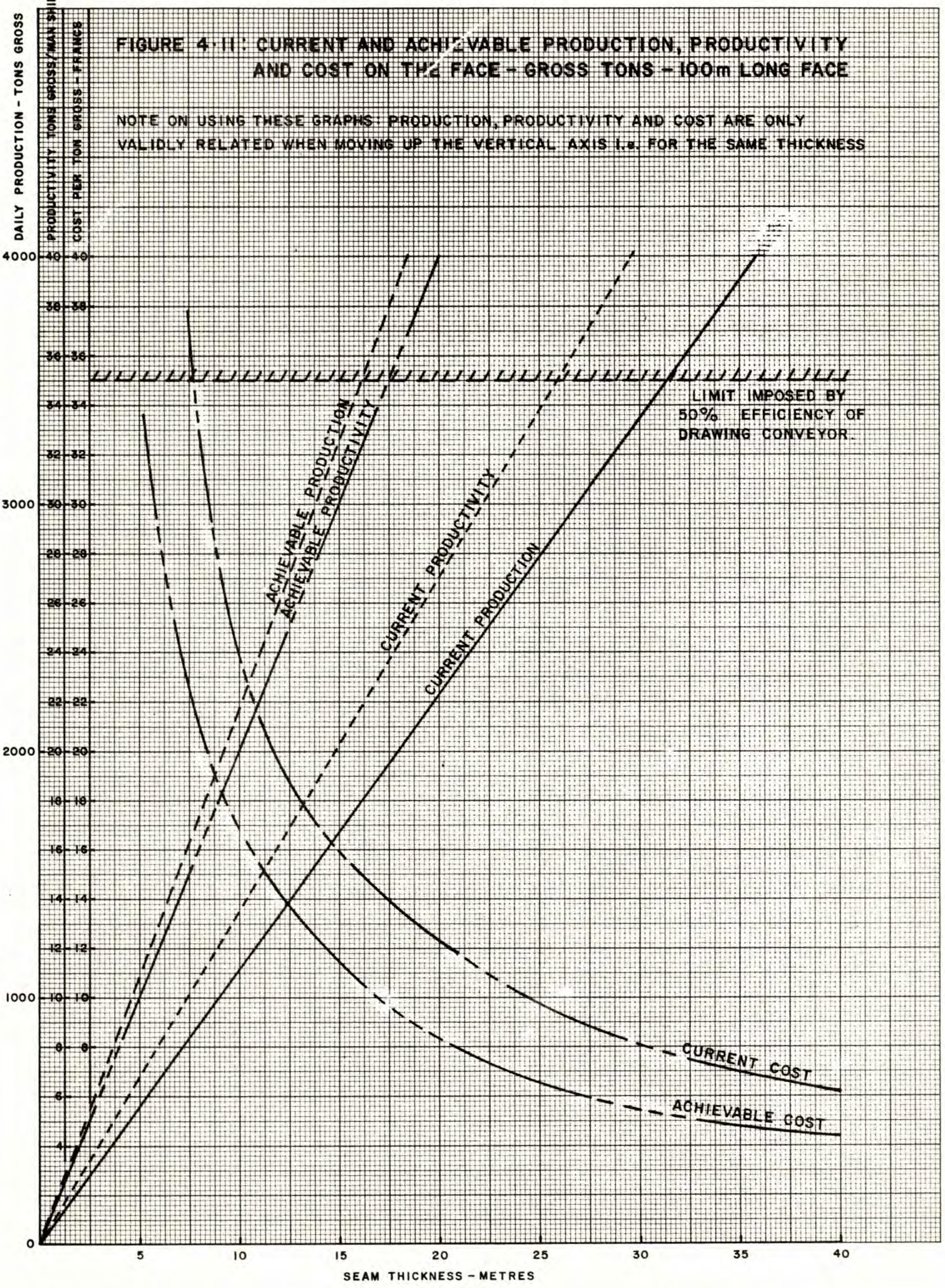


FIGURE 4·11: CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST ON THE FACE - GROSS TONS - 100m LONG FACE
NOTE ON USING THESE GRAPHS: PRODUCTION, PRODUCTIVITY AND COST ARE ONLY VALIDLY RELATED WHEN MOVING UP THE VERTICAL AXIS i.e. FOR THE SAME THICKNESS
DAILY PRODUCTION - TONS GROSS
PRODUCTIVITY TONS GROSS/MAN SHIFT
COST PER TON GROSS - FRANCS
LIMIT IMPOSED BY 50% EFFICIENCY OF DRAWING CONVEYOR.
ACHIEVABLE PRODUCTION
ACHIEVABLE PRODUCTIVITY
CURRENT PRODUCTIVITY
CURRENT PRODUCTION
CURRENT COST
ACHIEVABLE COST
SEAM THICKNESS - METRES
0
1000
2000
3000
4000
5
10
15
20
25
30
35
40

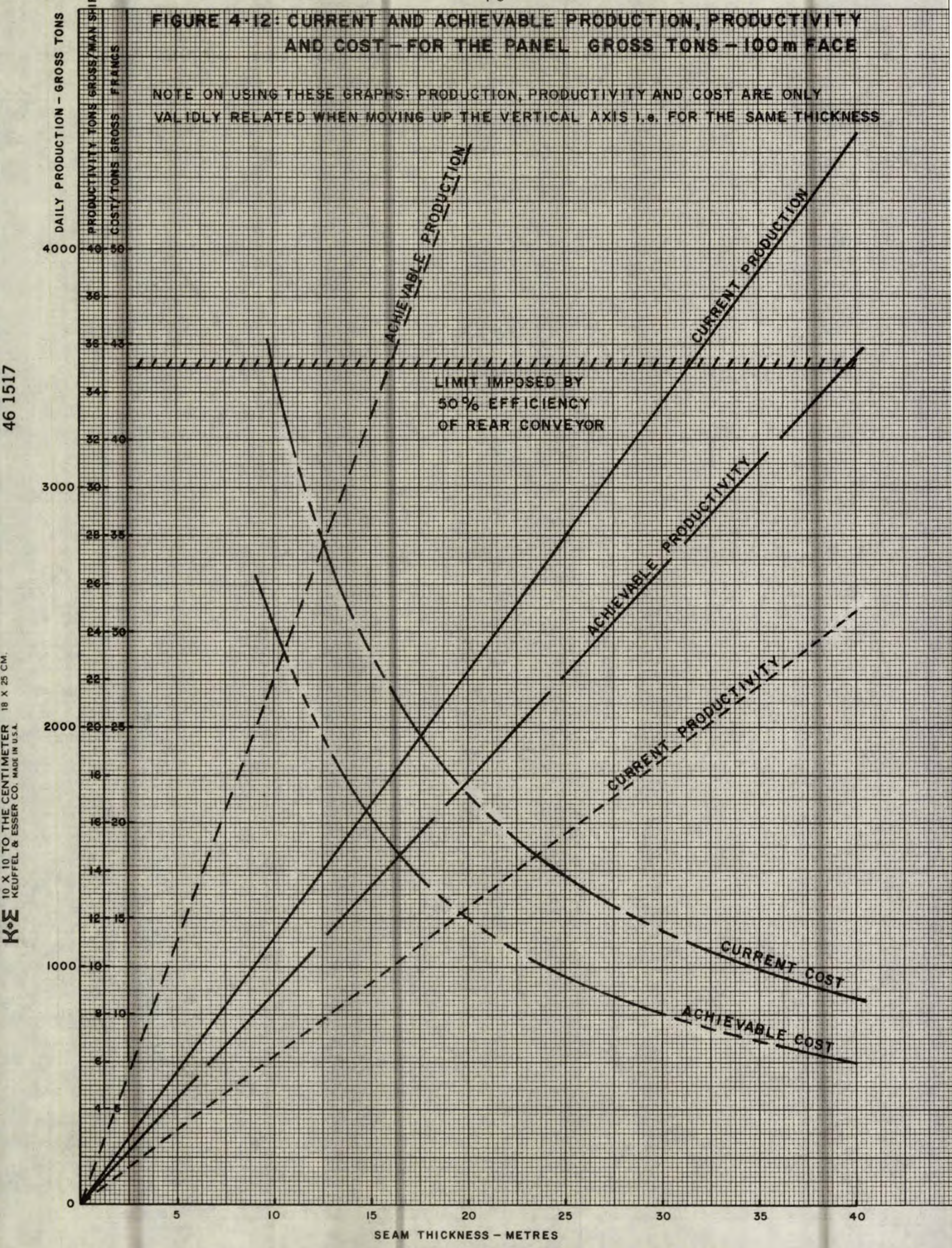


FIGURE 4-12: CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST – FOR THE PANEL GROSS TONS – 100m FACE
NOTE ON USING THESE GRAPHS: PRODUCTION, PRODUCTIVITY AND COST ARE ONLY VALIDLY RELATED WHEN MOVING UP THE VERTICAL AXIS i.e. FOR THE SAME THICKNESS
DAILY PRODUCTION – GROSS TONS
PRODUCTIVITY TONS GROSS/MAN SHIFT
COST/TONS GROSS FRANCS
ACHIEVABLE PRODUCTION
CURRENT PRODUCTION
LIMIT IMPOSED BY 50% EFFICIENCY OF REAR CONVEYOR
ACHIEVABLE PRODUCTIVITY
CURRENT PRODUCTIVITY
CURRENT COST
ACHIEVABLE COST
SEAM THICKNESS – METRES
0
1000
2000
3000
4000
5
10
15
20
25
30
35
40
K+E 10 X 10 TO THE CENTIMETER 18 X 25 CM. KEUFFEL & ESSER CO. MADE IN U.S.A.
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CHAPTER 5. GROUND CONTROL STUDIES AT BLANZY COLLIERIES

A number of ground control studies have been carried out at the Blanzy collieries; the results of these studies are summarised and discussed in this chapter.

5.1 The Behaviour of Powered Supports on a Longwall Face with Caving and Drawing

A study was carried out (4) on the behaviour of Westfalia type 68 powered supports to see if the support characteristics (setting and yield loads) were suited to the mining conditions. These studies were carried out in Panel I of the Darcy mine, at a depth of cover of 860 m, on a 100 m face which was about 3 m high.

Measurements were made of the internal pressure in the rams, the supply reservoir pressure at the prop and of the yield movement of the 3 rams of one side of one support in the centre of the face.

The specified support characteristics (see Appendix 4 for full specifications) were:

Setting pressure of props: 325 bars (50 tons load)
Yield pressure of props: 420 bars (65 tons load)

Pressure transmitted to the floor, at yield load = 23 bars.

Pressure on roof, at yield load, 260 tons over approximately 5.3 m^2 which is approximately 5 bars.

The main function of the supports is to support the face; a secondary function is to apply alternating pressures, via the banana prop, to the roof coal to accelerate caving. Theoretically the internal ram pressure should build up from the setting pressure until yield load is reached, the pressure is then maintained at this level until the face is advanced. At the same time the convergence will slowly increase until the yield pressure is reached, then drop rapidly in proportion to the deformation between the footwall and hangingwall.

In practice the cycle is significantly different; because of the mechanical linkage between the canopy and the banana prop, loading of the coal by the banana prop and the ensuing unloading due to caving result in large amplitude movements of the canopy with a corresponding reaction in the ram pressures. The behaviour cycle of these powered supports is therefore significantly different from that in a conventional retreating or advancing longwall face.

Pressure measurements were recorded on one support for 57 cycles on the front ram and for 67 cycles on the rear ram (the difference in number of cycles recorded was due to equipment breakdown). For each observation cycle 4 pressure values were recorded; the setting pressure, the release pressure, and the maximum and minimum pressures. Table 5.1 gives the results.

TABLE 5.1

Ram Pressures on Powered Supports

	Front prop		Rear prop	
	Pressure (bars)	Standard deviation	Pressure (bars)	Standard deviation
Number of cycles	57		67	
Setting pressure (bars)	213.7	57.8	179	74.5
Release pressure (bars)	207.5	128	236	101
Minimum pressure (bars)	147.5	92.5	146	83.8
Maximum pressure (bars)	271	57.8	268	72
Setting efficiency	0.67		0.56	
Release efficiency	0.49		0.56	

The setting efficiency is defined as the average actual setting pressure divided by the theoretical setting pressure.

The release efficiency is defined as the average release pressure divided by the specified yield pressure.

The following conclusions were drawn from these studies:-

1. The behaviour of the powered supports on a longwall caving and drawing operation is significantly different from that observed on conventional longwall faces. These differences were attributed to:-

   - the different constraint on the roof coal in the thick seam compared with the effects on a thin seam.
   - the mechanical liaison between the banana prop and the main props via the canopy.

   These differences produce large amplitude fluctuations in the yield movement, in both directions, and the bearing capacity can reduce significantly during the cycle; e.g. on average the minimum pressure on the front ram is only 69% of the setting pressure whilst that for the rear ram is 81%.

2. In normal use the support operates largely under its nominal capacity. The front and rear props on average operate in the pressure range 140 - 270 bars which is less than the specified nominal setting pressure (320 bars). The corresponding loads are 22 - 42 tons per ram. Less than 25% of the rams reach the nominal setting pressure at the moment of yield. For example the nominal yield load corresponds to 65 tons per ram but in practice the props yield at an average of about 35 tons.

3. The setting pressure efficiency is better on the front prop than on the rear prop; in contrast, the ultimate release efficiency of the front prop is less satisfactory than the rear prop. In fact, in a large number of observations (44%) the front ram has a yield pressure less than the actual setting pressure.

5.2 Gate Road Deformation under the Influence of the Longwall Face with Caving and Drawing

A series of studies on gate road deformations were carried out in various panels of the Darcy mine (5, 6). The gate roads were all supported by TH steel arches, as described in section 3.3.2, which were on occasion reinforced by trapezoidal sections of timber; in particular, the length of the gate road from the face to about 12 m ahead of the face was systematically brushed and rings were replaced by trapezoidal timber sections as shown in Figure 3.11. Measurements were made in the gate roads of panels I, C, D and Ga; this included the gate roads of single faces (I, Ga), a double face (C, D); roads adjacent to narrow barrier pillars and immediately adjacent to extracted areas; it also included gate roads underlying previous mining in the seam above.

Measurements of convergence (closure of roof to floor) and rib closure in the gate roads were made and records were kept of the maintenance (brushing) required in the roads relative to the face position.

The following conclusions were drawn from these studies:-

1. The gate roads are influenced by the face much farther ahead of the face than is the case in thin seams. For the gate roads of a single face the distance is approximately 100 m ahead of the face; for the gate roads of a double face the distance is from 100 - 150 m.

2. More than half the convergence measured during the gate road life was due to the influence of time (and not of the face). Consequently the more quickly the panels are developed and then extracted the less is the overall closure and the less maintenance is required.

3. A gate road developed alongside a narrow pillar (~ 10 m) was subject to much higher convergence than other gate roads and required twice the maintenance.

4. The presence of an overlying mined out area appears to have a beneficial effect on the gate roads; however results were insufficiently accurate to establish this conclusion categorically. On the other hand, when the gate road passes beneath the exploitation limit of the overlying area there may be more severe convergence, which may become increasingly serious with time.

5. The support trials were not conclusive but the section of trapezoidal timber supports apparently tended to limit the convergence to lesser amounts.

6. The brushing of the floor required during maintenance was almost twice the measured absolute movement of the floor.

5.3 Effect of Mining a Diminishing Pillar

Panel I in the Darcy mine (see Figure 3.1) was mined in two stages, as shown in Figure 5.1. The region I_1 was first mined out; the region I_2 was then mined in the opposite direction so that there was a steadily diminishing remnant pillar between the two gob areas. Measurements were made to assess the effects of this diminishing remnant pillar on the adjacent and underlying roadways and to assess the competency of this pillar (6). The following summarizes the main conclusions from these studies:-

1. Previous work in thin seams (< 3 m) has shown that a coal pillar had completely failed after its width was less than 20 m. The disturbance in the neighbouring areas caused by pillar failure can be very important. The object of this study was to specify these "pillar effects" for a thick seam with a longwall caving and drawing panel.

2. The measurements indicated that the coal pillar had completely failed before mining was completed. However, its effective width was sufficiently large (> 50 m) at failure that it was unable to act

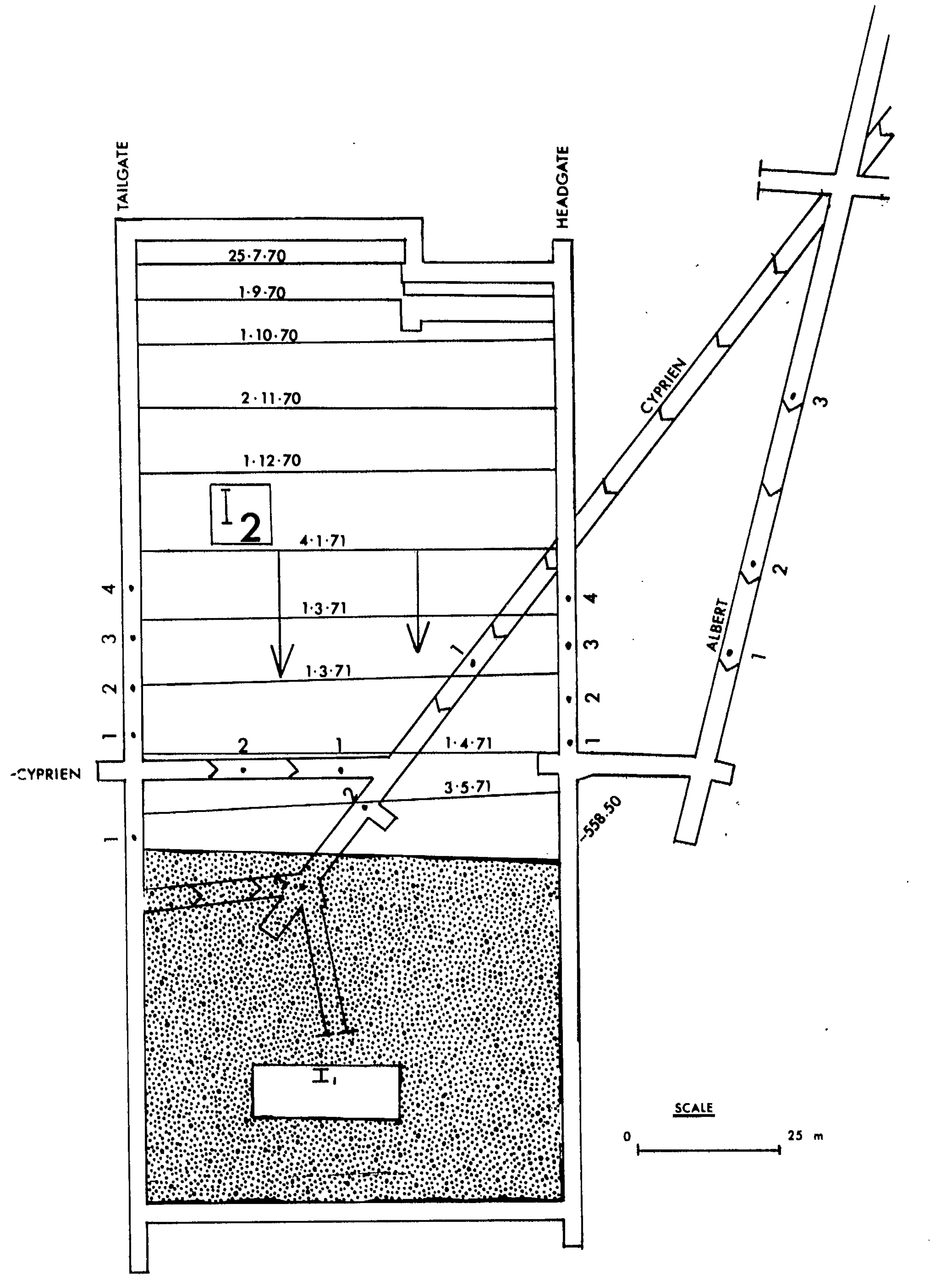


FIGURE 5.1: MINING SEQUENCE - PANEL I

as a "punch" on the adjacent ground and by doing so damage the workings in these areas. This could be seen from the observed stability in the underlying haulageway and adjacent gate roads.

3. It would be desirable to be able to specify more closely the effective pillar width at failure in order to apply the results elsewhere. A sufficiently detailed knowledge of the behaviour of the pillar according to its width would allow the specification of the dimensions of harmful pillar remnants (in thin seams a completely failed pillar no longer represents a remnant dangerous to adjacent openings). Such a study would however require a greater density of instrumentation that was used in this study.

5.4 Ground Behaviour in a Thick Seam Mined by a Retreating Longwall Face with Caving and Drawing

Comprehensive ground control studies were carried out (7, 8) in both the Darcy and Rozelay mines to determine and compare the ground movements and support behaviour during longwall mining with caving and drawing. These studies and their results are summarized below:-

5.4.1 Studies in the Darcy mine (7)

The broad objective of these studies was to obtain a knowledge of the ground behaviour associated with this mining method and to attempt to use this information to assess the possibility of mechanizing the face advance in such panels. In particular it was desired to obtain a knowledge of the deformations and movements which affect the mass of coal around the workings. By comparison with flat seams mined in thin beds, the presence of a thick coal roof and of caving at the rear of the face may considerably modify the factors influencing ground behaviour. The study of deformations was supplemented by a study of the face support behaviour, one of the important points being to determine the support loading most suitable for this ground. It should also be noted that the supports themselves are an active factor in the deformation process as is, of course, the mining phase of advancing or of caving.

The measurements in the Darcy mine were carried out on face D at a depth of cover of 797 m, seam thickness 12.35 m, face length 90 m and face height 2.8 m. Westfalia K2 type 68 supports were used on this face. The measuring section, illustrated in Figure 5.2, comprised a length of 8 supports (Nos. 28 - 35). The reference support was No. 31 and the main measurements were carried out between supports No. 31 and No. 32, as shown in Figure 5.3.

Three types of measurement were made:-

(i) Deformation measurements

- face convergence on two parallel lines between the front and the rear of the face, and measurements of the horizontal displacement between the roof and floor anchors.

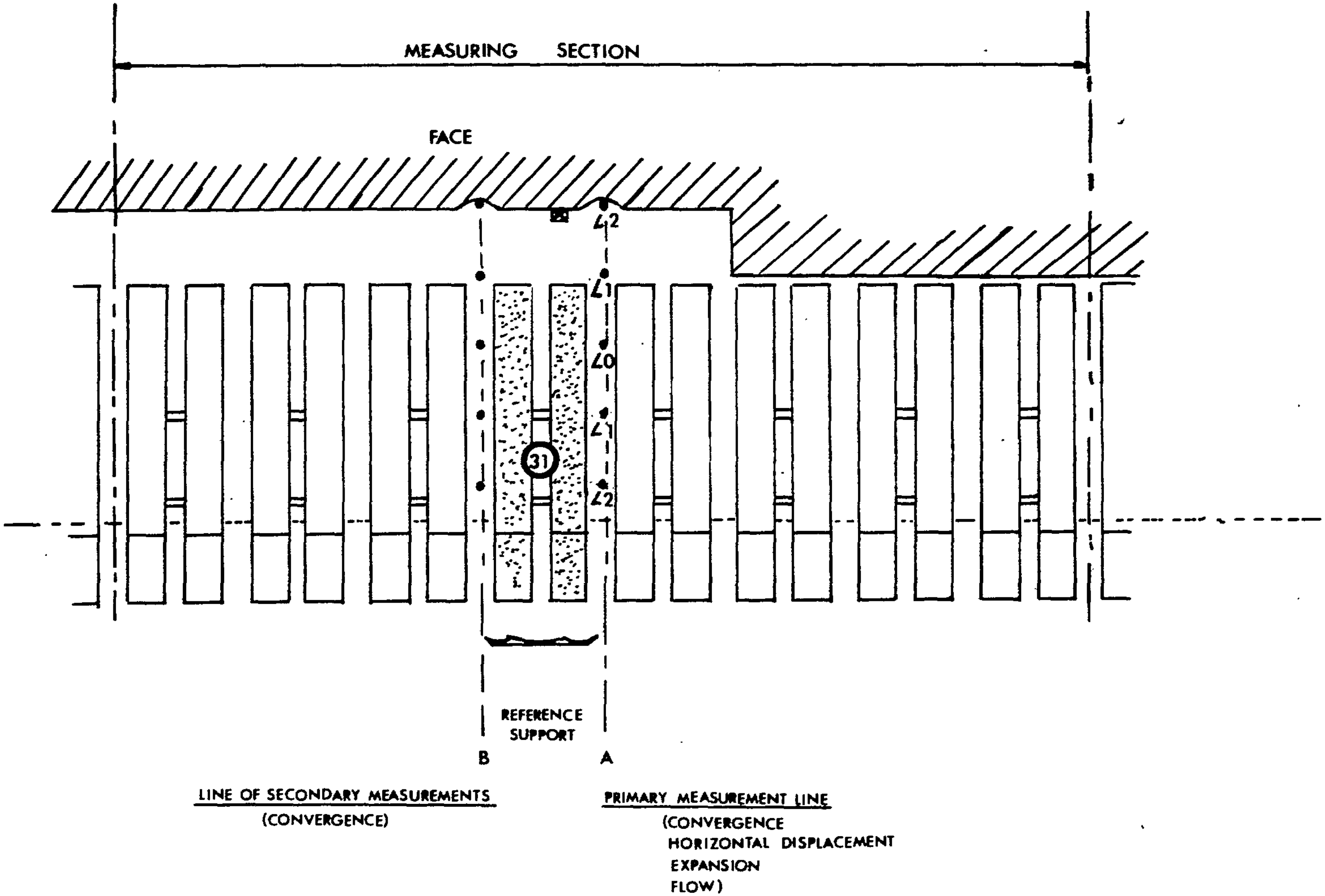


FIGURE 5.2: MEASURING SCHEME, FACE D

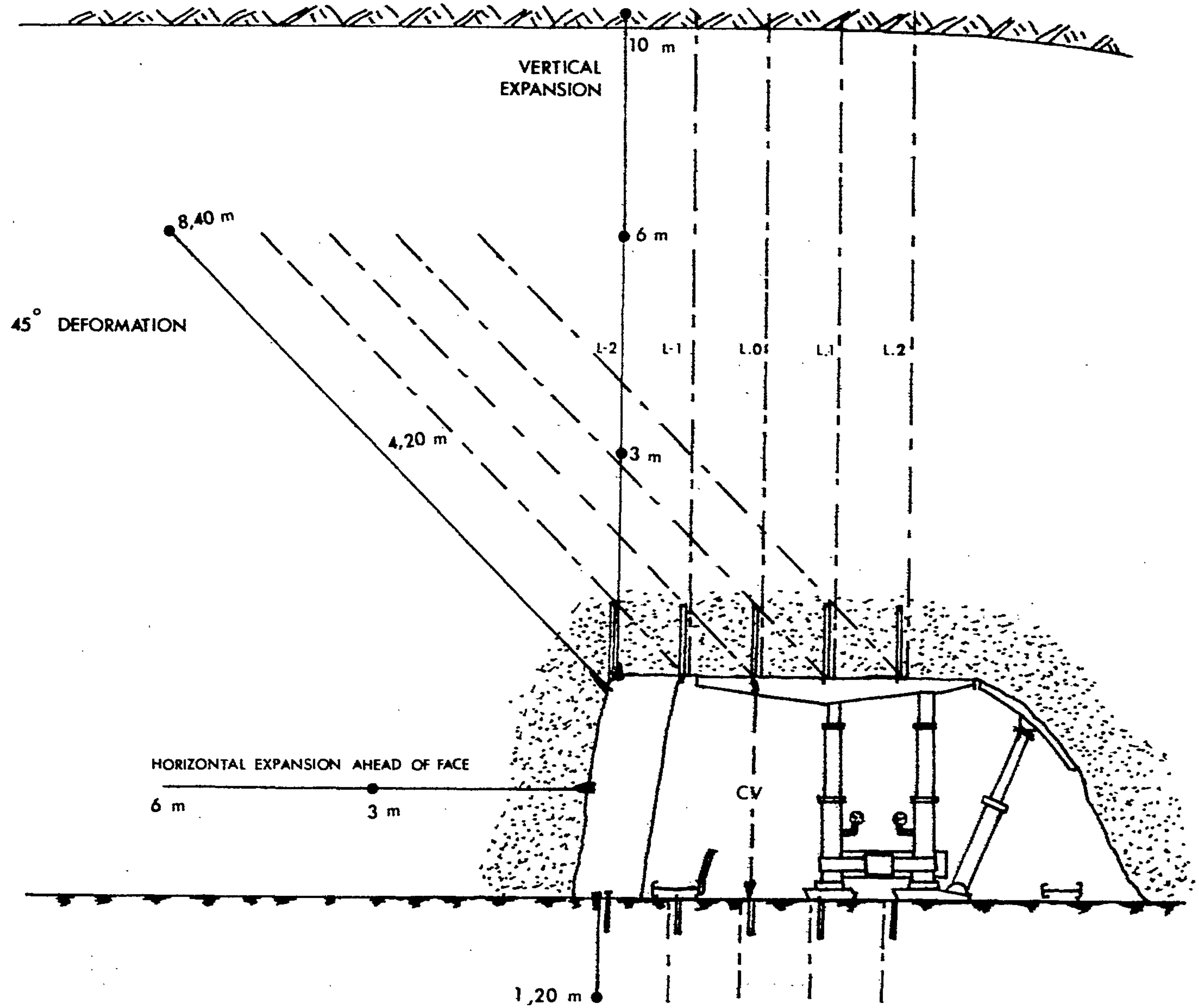


FIGURE 5.3: SECTION THROUGH MEASURING STATIONS — FACE D

- deformation in the overlying coal; vertical expansion between the roof of the face and three horizons (the true roof, the 6 m and the 3 m levels).
- deformation, at 45°, between the roof of the face and points at 45° corresponding to the 3 m and 6 m horizons.
- horizontal expansion ahead of the face, between the face and points 3 m and 6 m ahead of the face.
- apparent expansion between the face floor and the true floor (which was 0.8 - 1 m below the face floor).

(ii) Support measurements

- on the reference support, (No. 31), ram pressures, yield and prop inclination were measured.
- on all eight supports comprising the measuring section, the pressures before and after advance were recorded.
- measurements were also made of the force on, and the yield of the friction props placed against the face (see Figure 3.10 for their positions).

(iii) Reference data

Complete reference data concerning the various phases of mining (advance, caving, drawing, etc.) with time and width of face ahead of the powered supports at any time were also recorded.

5.4.2 Results from the Darcy mine studies

(i) Convergence

The total mean convergence on the face was 1059 mm which corresponds to a value of 233 mm/metre of face advance.

Now it has been shown from previous work on longwall faces in thin seams (based on a statistical analysis of 140 faces) that the convergence per metre of face advance, Cv, expressed in m/m is given by:

$$Cv = 0.2\ \{qW\}^{0.75}\ H^{-0.25}$$

where W is the seam thickness in metres (equals the face height for thin seam), H is the depth below surface in metres and q is a parameter which depends on the treatment of the gob behind the face. For caving q ~ 1, for well compacted backfill in the gob, q ~ 0.5.

Using this formula and taking q = 1, H = 797 m and W = the face height 2.8 m then:-

$$Cv = .085\ m/m = 85\ mm/m$$

This value is much less than the observed value of 233 mm/m.

However, if W is taken as the total seam thickness, 12.25 m, rather than as the face height, then:-

$$Cv = 0.245 = 245 \text{ mm/m}$$

which corresponds closely to the measured value.

This leads to the important conclusion that, for thick seams, the total face convergence depends not on the face height but on the seam thickness between the true roof and true floor, i.e. the general statistically derived formula for thin seams, given above, is also applicable to thick seams, provided that W in both cases is taken as the seam thickness (and not as the face height).

It was also shown that the rate of convergence over the roof span held by the powered supports is less than that ahead of the supports. This indicates the possibility that convergence may be reduced by increasing the bearing capacity of the supports (but this reduction would probably not exceed 10 - 15%).

The convergence undergoes three surges:-

(a) during the advance of the supports (when in fact the roof is temporarily unsupported)

(b) during mining of the face (because the nature of the roof support is weakened from being part of the seam to that of a wooden truss with friction props)

(c) during caving; the cause of this is not clear, it might be due to the fact that the face advance phase is not usually completely separated from the caving phase.

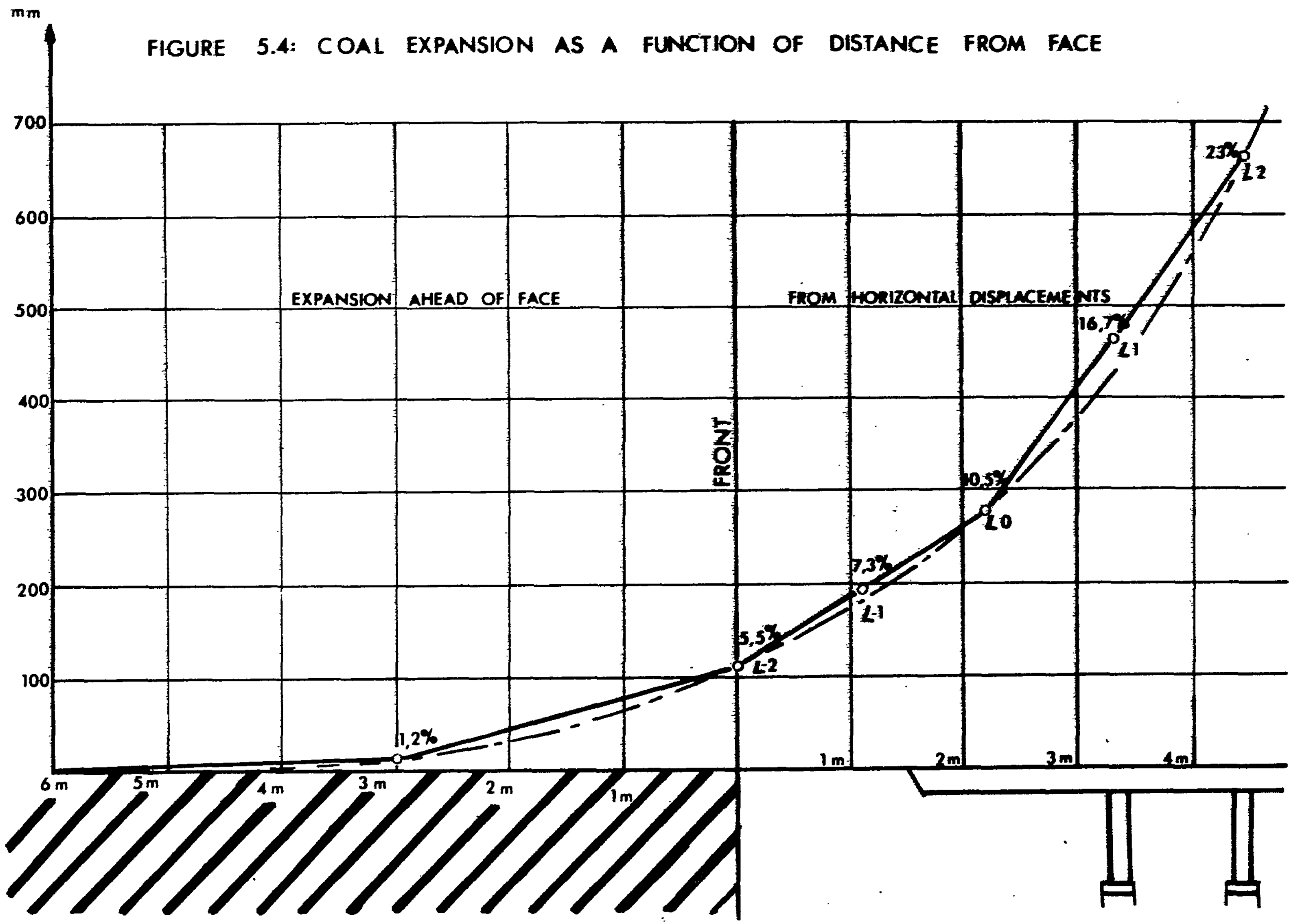
The convergence is also, of course, a function of time. It is known that increasing the speed of face advance will reduce the convergence, but the range of this reduction is probably quite small (10%?).

(ii) Horizontal displacements

The point 6 m ahead of the face was taken as the reference point; the relative expansion at the 3 m point was 1.2% (the coal is perhaps still coherent) and at the face the expansion was 5.5% (the coal is unconsolidated). It is clear that since mining the coal face has undergone a significant horizontal expansion; this explains its usual tendency to slab off.

The horizontal displacements in the vicinity of the canopies of the powered supports, measured by the relative displacement between the roof and floor anchors, show that the coal overhead flows continuously towards the caved area; the mean overall displacement between the face and the caved area reached 550 mm. This horizontal displacement accelerates during the advance of the powered supports. Figure 5.4 summarizes the percentage horizontal expansion determined from these tests.

FIGURE 5.4: COAL EXPANSION AS A FUNCTION OF DISTANCE FROM FACE



The coal above the supports is largely unconsolidated; the need for the wire mesh above the supports is thus emphasised.

Overall these results indicate that the conditions for eventual mechanization of these workings are not favourable because:

- the unconsolidated top of the face needs only to slab off and the equally unconsolidated mass above is exposed and may cave, creating caving ahead of the supports.
- even if the step of the face advance is reduced by half (from ~ 1 m to ~ ½ m) when a shearer is introduced on the face, there may still be serious problems in supporting this region if support is not placed immediately under this unconsolidated roof.

(iii) Expansion of the coal mass above the face

From the measurements in the 45° and vertical holes, Figure 5.5 has been constructed. Although construction of this figure involves some assumptions, nonetheless it gives a good general picture of the flow of coal towards the caved area. These expansion measurements clearly indicate that the degree of unconsolidation is such that caving will usually take place without the need for shot firing.

(iv) Support behaviour

The nominal support characteristics were:-

Yield pressure: 480 bars
Setting pressure: 240 bars (Note: this was limited by the hydraulic reservoir capacity; the specified pressure should be 325 bars).

Table 5.2 gives the recorded mean ram pressures and their efficiencies (as defined in section 5.1).

These results are very dispersed, especially at unloading where 20% of the front rams and 30% of the rear rams had reached yield pressure.

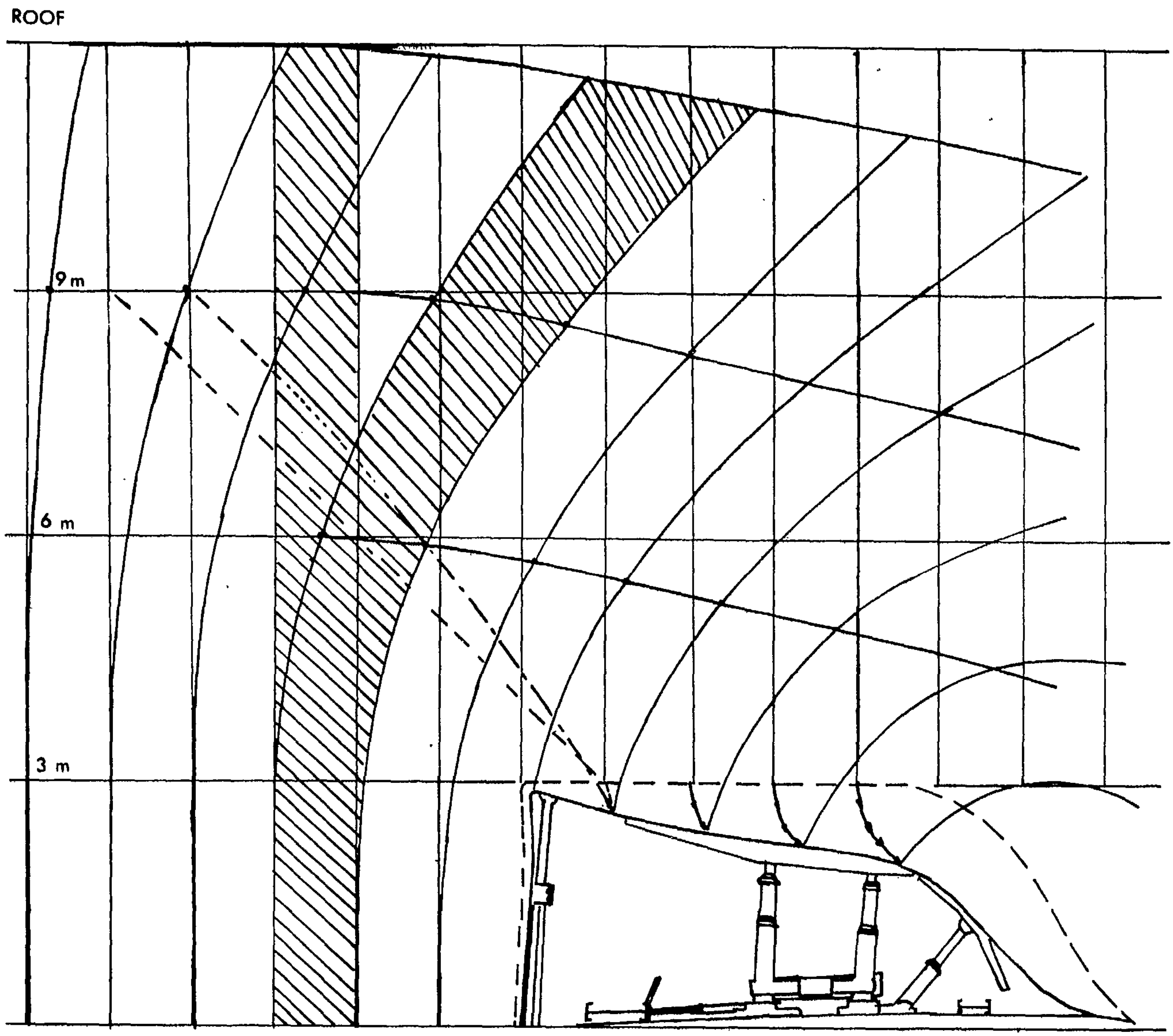
It is evident that, on average, the supports are being used well below their actual capabilities. Improved performance (without increasing the overall setting pressure) can only be achieved by systematically pumping up those rams in which the pressure is low; this would achieve a significant gain in the setting pressure and efficiency but a greater reduction after advance.

(v) Bearing capacities and the load per metre of face

The total load, before advance of the supports, is about 200 tons (190 tons from the powered support and 10 tons for the two friction props at the face); after advancing the supports it reduces to about 138 tons.

After support advance the bearing capacity per m^2 of exposed roof reaches 2.8 tons and the load per metre of face is 135 tons/m.

FIGURE 5.5: DEFORMATION AND FLOW OF ROOF COAL



• DISPLACEMENT MEASURING POINTS

DISPLACEMENT OF A COAL SLICE

TABLE 5.2

Ram Pressures and Efficiencies - Darcy Mine, Face D

		Setting pressure	Unloading pressure
Front ram	Mean	171.5 bars	325.6 bars
	Standard deviation	65.0 bars	124.9 bars
	Dispersion %	37%	39%
	Efficiency	0.71	0.67
Rear ram	Mean	218 bars	308.6 bars
	Standard deviation	96 bars	167.3 bars
	Dispersion %	44%	55%
	Efficiency	0.89	0.64

It should be noted that these forces are much higher than the weight of the coal overlying the supports which is approximately 65 t/m.

(vi) Yield and inclination of the rams

The yield movement of the rams is only a small portion of the total convergence (~ 20%). During caving there is a negative yield movement (i.e. expansion) of the rams.

Although when originally set the rams are inclined, on average, at 1^{o} towards the solid, on unloading they are tilted 3^{o} - 4^{o} towards the gob. This change in ram inclination corresponds closely to the roof displacement mentioned above. This push towards the rear is an unfavourable element as far as holding the roof at the face is concerned; a counteracting ram might seem desirable but with the roof above relaxed by 5% it is doubtful whether the beneficial effect would be significant.

5.4.3 Studies in the Rozelay mine (8)

A ground control study, similar to that described above for the Darcy mine, was carried out in the Rozelay mine with the object of:-

(a) comparing the results with those from the Darcy mine

(b) examining the effects of the sandstone bed in the middle of the No. 2 seam at Rozelay

(c) distinguishing, if possible, the reasons which have allowed the mechanical shearer to be used successfully at Rozelay but which in a short trial at Darcy produced unsuccussful results.

The measurements were carried out on face S3b in the No. 2 seam at the Rozelay mine during the period 15 May to 9 June 1972. The face was 310 m below surface, face length 109 m descending at between 6^{o} and 8^{o}. S.M.F. powered supports were used on the face and the DTS 300 double drum shearer was used for face advance. Other face and seam characteristics were:-

Face height	2.4 m
Thickness of sandstone bed in the roof -	2.17 m
Seam thickness, including the sandstone bed -	12 m
Coal thickness	9.85 m
Thickness of coal below the sandstone bed	5.2 m
Thickness of coal in the floor	0.48 m

5.4.4 Results from the Rozelay studies

(i) Convergence

The total face convergence was 660 mm, corresponding to a value of 160 mm/m of face advance. Using the theoretical formula $Cv = 0.2 \{qW\}^{0.75} H^{-0.25}$ and assuming (a) qW = 2.4 (W = face height 2.4 m, q ~ 1) and (b) qW = 12 (W = seam thickness, q ~ 1) then the calculated convergence Cv is, respectively, 92 mm/m and 300 mm/m.

The actual convergence measured, 160 mm/m, lies between these two values. Thus, contrary to the case at the Darcy mine, the convergence predicted by this formula (using W = 12 m) is considerably greater than the measured value. In fact calculations show that a theoretical convergence of 160 mm/m corresponds to a value of qW = 5.3 m (which for W = 12 m would make q = 0.45). However it is interesting to note that the value qW = 5.3 m corresponds very closely with the total thickness of coal beneath the sandstone bed (5.2 m).

The convergence undergoes three surges:-

- at the time of support advance, when the roof is unsupported; the convergence rate is 19.2 mm/hr.
- at the time of shearing the face, removal of the support of the roof, temporary reduction in the bearing capacity; the convergence rate is 7.9 mm/hr.
- at the time of caving; readjustment of the overlying coal with shifting of the beds; the speed of convergence is 7.9 mm/hr.

(ii) Horizontal displacements and roof displacements

The results on face S3b showed that the mean expansion of coal between the face and 6 m ahead of the face was 2.1%. The percentage expansion 3 m ahead of the face was 0.3% whilst that at the face itself was 4%. The expansion above the supports, at the rear caving line is ~ 9% compared to 26% at Darcy.

The horizontal displacement measurements and the vertical displacement measurements indicated a distinct influence of the sandstone bed in the middle of the seam. It appears that the presence of the sandstone bed reinforces the seam in front of the face and limits the expansion of the unconsolidated zone ahead of the face. Similarly, in the roof there is good support closer to the face, i.e. the unconsolidated zone stabilizes more rapidly.

(iii) Support behaviour

The behaviour of the powered supports showed:

(a) The front ram efficiencies were 0.69 for the setting pressures and 0.74 for the unloading pressures. The efficiencies of the rear rams were lower, 0.48 for both setting and unloading.

(b) The ram yield in this case clearly represented a larger fraction of the total convergence than in the Darcy case (Cl/Cv = 0.57 as opposed to 0.2 for Darcy). This indicates a better roof control, less punching into the roof and less unconsolidation of the overlying coal.

(c) The tilting of the rams was compatible with the observed horizontal displacements.

5.4.5 Comparison and conclusions

Table 5.3 shows a comparison of the more important factors derived

TABLE 5.3

Comparison of Results from Darcy and Rozelay Mines

	Darcy - Face D	Rozelay - Face S3b
Face advance by	hand	drum shearer
Face rear	caving and drawing	caving and drawing
Face supports	Westfalia K2, 68	S.M.F.
Depth of cover	797 m	305 m
Seam thickness	12.3 m	12 m
Face height	2.8 m	2.4 m
Total convergence	1060 mm	660 mm
Convergence per metre (mm per m from face)	233 mm/m	160 mm/m
qW - theoretical	11.3 m	5.3 m
Floor expansion	168 mm	250 mm
Horizontal displacement		
- towards the rear	552 mm	280 mm
- to the left	118 mm	178 mm
- linear	565 mm	330 mm
Mean horizontal expansion of coal 0 - 6 m in front of the face	3.8%	2.1%
Expansion of coal		
to 3 m	1.2%	0.3%
at the face	5.5%	4%
towards the caving area	23%	9%
Depth of relaxed zone	~ 3.5 m	~ 2 m
Mean ram pressures (and efficiency compared to theoretical)		
front rams - setting pressure	171 bars (0.71)	220 bars (0.69)
rear rams - setting pressure	218 bars (0.89)	153 bars (0.48)
front rams - unloading pressure	325 bars (0.67)	276 bars (0.74)
rear rams - unloading pressure	308 bars (0.64)	180 bars (0.48)
Actual load/unit area	2.8 bars	2.5 bars
Mean yield - front rams	23.4 mm	32.5 mm
- rear rams	65 mm	24 mm
Mean tilting - towards the rear	3.54°	1.11°
- towards the left	0.86°	0.45°

from these ground control studies at Darcy and at Rozelay. There are some significant and important differences. Both seams were of similar thickness (12.3 m compared with 12 m); the face heights were also fairly close (2.8 m compared with 2.4 m). The main differences in the seams were (a) the overburden depth was much greater at Darcy than at Rozelay (797 m compared with 305 m) and (b) the seam at Rozelay contained a strong sandstone bed in its centre. The following conclusions may be drawn from the results.

1. The empirically derived equation, $Cv = 0.2 \{qW\}^{0.75} H^{-0.25}$, developed for describing the convergence per metre of face advance for thin seams is also applicable to thick seams provided that W is taken as the seam thickness and not as the face height. The agreement with the Darcy results is excellent. In the Rozelay seam the same equation applies but in this case the influence of the sandstone bed is evident in that the equation only applies if the seam thickness is taken as the thickness of coal below the sandstone bed.

2. In both cases there is an unconsolidated zone of coal ahead of the face and overlying the supports. However the unconsolidated zone is less in the Rozelay situation than at Darcy; this is probably due to two causes:

   (a) The sandstone bed appears to give a reinforcement to the seam and tends to prevent growth of the unconsolidated zones.

   (b) The large difference in overburden pressures also probably accounts, at least in part, for the fact that the Rozelay coal is more stable than that at Darcy.

3. This difference in overall stability of the coal immediately ahead of and above the face is probably a key factor in determining the successful use of a drum shearer at Rozelay compared with an unsuccessful trial at Darcy. One would anticipate that there would be a much greater tendency for the face to slab off and for the overlying coal ahead of the supports to cave in the Darcy conditions than at Rozelay. This is a very important factor in considering the potential application of this method in Canada. In the author's opinion, based on a subjective judgement only, the coal seams in the Western Canadian Rocky mountains and foothills are much more highly sheared and friable than either coal seams at Darcy or at Rozelay. Unfortunately it was not possible to quantify this subjective judgement; a very indirect measure might be a comparison of the percentage fines coming from the mine; at Blanzy approximately 60% of the coal was less than 8 mm, at one Western Canadian mine 82% of the coal output is less than 9.5 mm (3/8"). Whilst these figures are not really comparable and the percentaged fines can only be an indirect indicator of the relative competency of the coals, it is thought that they tend to support the author's subjective judgement.

4. The fact that the yield of the supports at Rozelay was a greater percentage of the overall convergence than at Darcy also indicates that there was better roof control in Rozelay and that there was less tendency for the supports to punch into the roof.

5. These studies showed that, in general, the powered supports in use perform well below their rated capacities. This would seem to be a fact of life with this method but should perhaps be borne in mind when designing face support; it might be advisable to use a higher rated capacity support than theoretically would be recommended so that the gap between actual performance, in terms of bearing capacity, and desired performance be narrowed.

6. Methods of attempting to reduce the liability of the face to slab and the roof ahead of the supports to cave should be examined. The following factors, while certainly not eliminating the problem, do contribute to better face conditions:-

   (i) The time of exposure of the unsupported roof ahead of the supports should be reduced to a minimum. Consideration might be given to the design or modification of supports to achieve this end.

   (ii) A face descending down dip is obviously a desirable feature.

   (iii) The use of wooden or fibreglass bolts to help consolidate the face is desirable.

CHAPTER 6. ENVIRONMENTAL CONTROL

6.1 Dust Control

It will be appreciated that, without some form of control, the caving and drawing of the overlying coal will produce severe dust conditions on the face. Two very effective dust control methods are therefore used:

(i) Water infusion
(ii) Plastic sheeting over the supports (attached to the wire mesh).

6.1.1 Water infusion

The object of infusing the seam with water is to increase the natural moisture content of the coal and to wet the coal before caving so that the dust produced during mining and particularly during caving and drawing is reduced to acceptable levels. In general the infusion holes are drilled in accordance with the principles shown in Figure 6.1. The holes are drilled from both gate roads in inclined planes such as P_1 and P_2 in this figure. The orientation in these planes is such that the holes are uniformly distributed across the line C, D, where the holes meet the roof. Hole 5 is drilled along the axis of the gate road and a sixth hole will be drilled over the adjacent panel. As the face advances it passes below the line CD before reaching AB, so that the caved coal is wetted until almost the moment of caving. The exact location of the holes is, in any particular case, decided by the seam geometry, in general, for a 100 m face, a total of 12 holes per infusion section are drilled (6 from each gate road).

The holes are drilled with either a Meudon 4CV or a Turmag PIII - 4 drill. The hole size is 42 mm diameter. After drilling a 13 mm diameter plastic pipe is inserted in the hole for a length of about 20 m and is grouted in position with cement grout. It is necessary to use about 20 m of pipe to prevent leakage of water, via fissures, back into the gateroad.

Two days after the cement has set, water infusion begins; a water pressure of about 10 - 20 bars (150 - 300 psi) is used. The flow of water is regulated to be about 1.5 litres/minute for each hole, which is approximately 2 m^3/day. A flowmeter is attached to each hole and read daily. The aim in general is to infuse about 15% by volume of water into the volume of the slice between P_1 and P_2.

Water infusion is carried out on working days only but during all three shifts. Eight men per shift per panel are used for this water infusion program, plus 2 men for supplies and a foreman.

Figure 6.2 shows an actual layout of water infusion holes used in panel T during the month June - July 1972. This was a trial section in which the holes were drilled in vertical planes rather than in planes inclined towards the face; the trial yielded no conclusive results and work later reverted to the inclined planes. At the start of this month, sections F, G, G_b, H, I, J, K, L, M and N comprising 53 holes in the headgate and sections 7, 7b,

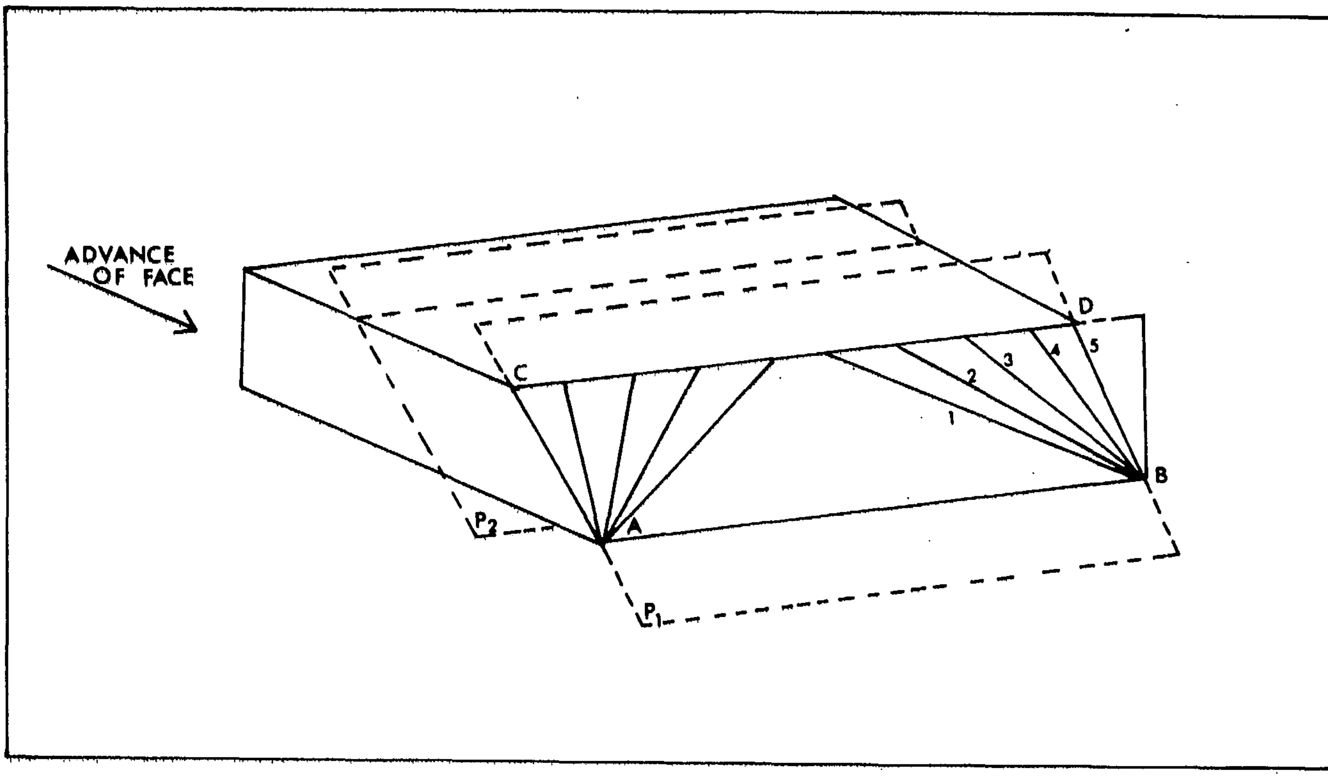
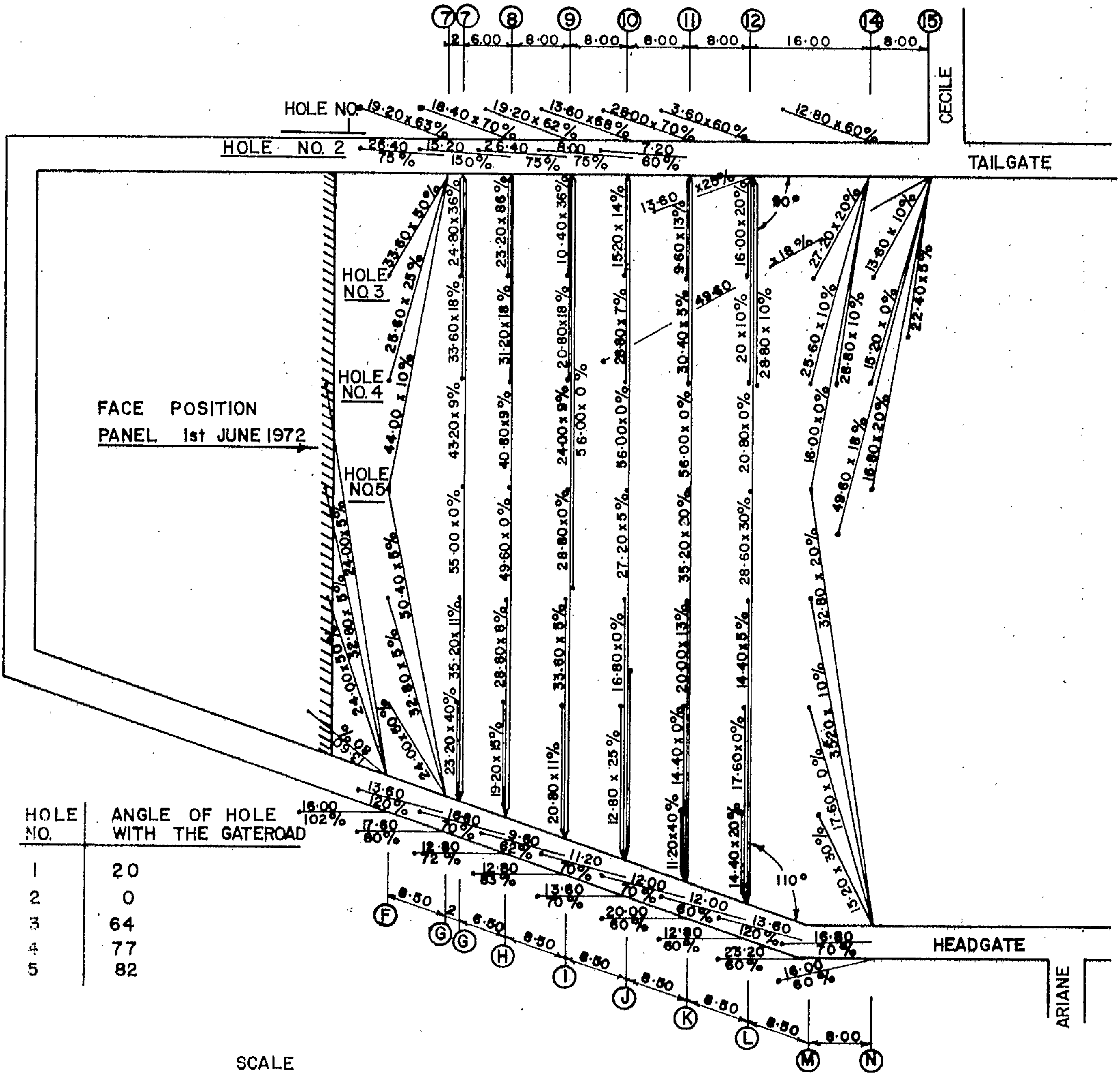


FIGURE 6.1: BASIC PRINCIPLE OF HOLE LAYOUT FOR WATER INFUSION

FIGURE 6.2: WATER INFUSION PANEL 'T'
1st JUNE 1972



HOLE NO.	ANGLE OF HOLE WITH THE GATEROAD
1	20
2	0
3	64
4	77
5	82

8, 9, 10, 11, 12, 14, 15 comprising 47 holes in the tailgate were in use for infusion, The dispersion of the holes in sections 14 and 15 was due to a pinch in the seam. During the month new sections, 0 (5 holes) in the headgate and 16 (4 holes) in the tailgate were brought into service; while sections F, G, G_b, H and 7, 7b, 8 were overtaken by the face advance. The total volume of water infused in this period was 7220 m^3; the face production was 30,414 tons net which corresponds to 34,600 tons in place.

The natural humidity of the coal was 2.26%; the effect of infusion was to increase this to 4.82%. The mean dust counts on the face, given in number of respirable particles ($< 5\mu$) per cubic centimetre were:-

Shift 1: 1st 15 days	1302	Shift 1: 2nd 15 days	977
Shift 2:	1011	Shift 2:	1813
Shift 3:	1377	Shift 3:	1607

6.1.2 Plastic sheet over the supports

The unconsolidation and break up of the coal overlying the supports produces a lot of fine coal and dust which filters down through the mesh overlying the supports; when the supports are advanced this is disturbed and dust conditions in the face deteriorate. This was particularly noticed by the author on face C/D, where due to the hand advance mining cycle, the supports were only advanced on the night shift. The dust conditions, subjectively observed, on the face during the first two shifts seemed remarkedly good; however on the night shift, when all the supports were advanced, dust conditions became quite unpleasant. In the Rozelay mine, where the supports are advanced behind the shearer, this effect was averaged out over the shifts.

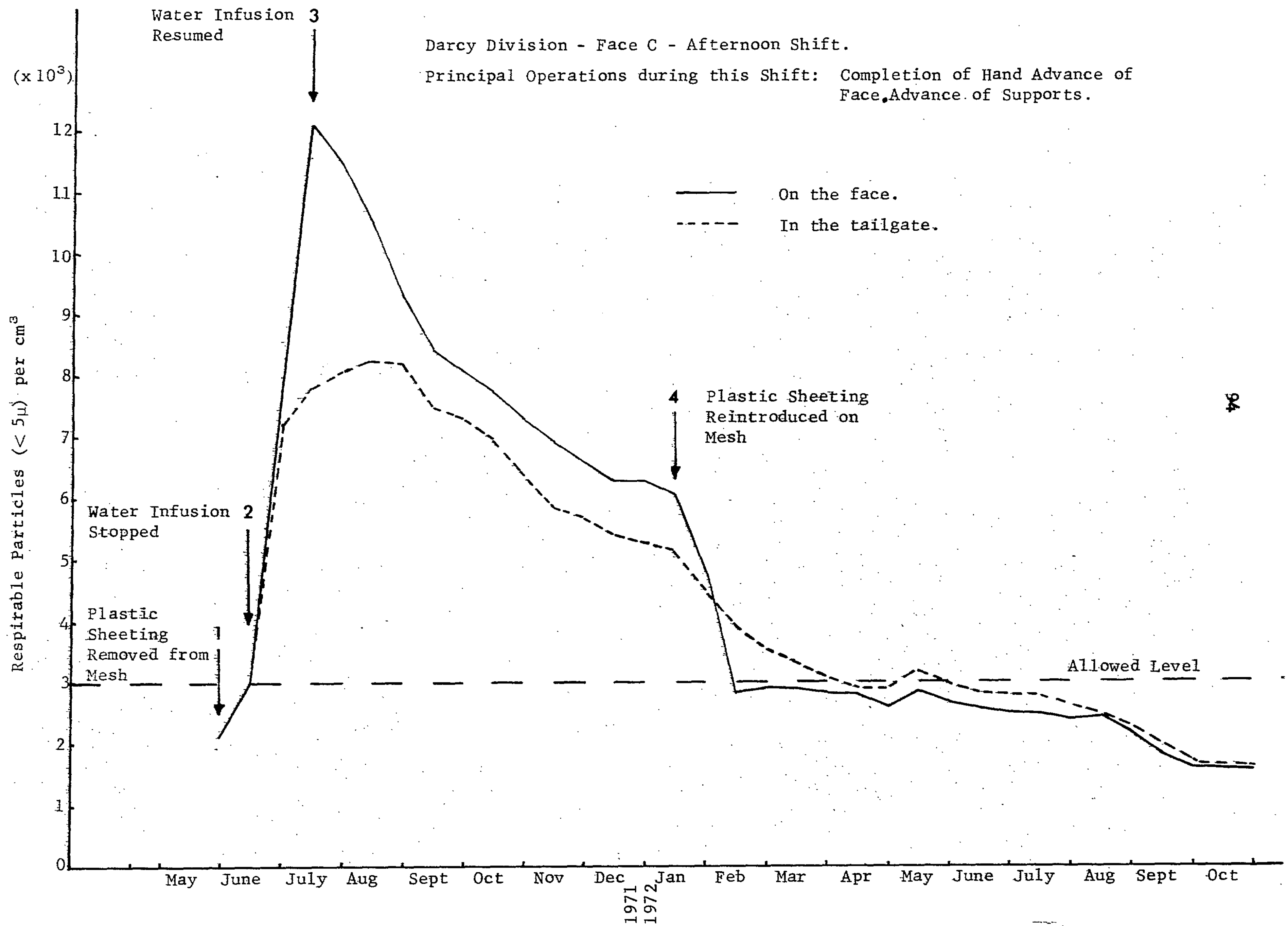
A lightweight plastic sheet is therefore now attached to the wire mesh and passes over the supports with this mesh. Although, of course, this sheet gets cut and damaged over the supports nonetheless it has a very beneficial effect in preventing dust seeping onto the face from above.

Figure 6.3 shows the results of a trial on face C/D in which water infusion was stopped and the plastic sheet was removed, and then the water infusion was restarted and later again the plastic sheet was reintroduced. Dramatic changes in the respirable dust count are seen at each of these stages and this figure clearly indicates both the value of water infusion and the plastic sheet in reducing the dust count on the face to tolerable and legally acceptable levels.

6.2 Spontaneous Combustion

In the past, spontaneous combustion has been a major problem in the Blanzy collieries; the coal has a relatively high susceptibility to spontaneous combustion. These problems were particularly noticeable during the period in which the mining method of single descending slices was used. This method demanded that panel ventilation be kept up around the gob area so that the next slice could be started, etc. This led to ventilation leakages through the gob which induced fires in the gob areas. One of the main attractions for introducing the footwall slice, caving and drawing system was to reduce this

FIGURE 6.3: Effect of Water Infusion and Plastic Sheet on Respirable Dust Count



hazard. This method extracts the whole seam in one pass and, when coupled with retreat mining, means that the gob area is abandoned completely and air leakages through the gob are minimized. The last gob fire at Blanzy occurred in 1966, and since the introduction of this mining system no gob fires have occurred.

The current retreat mining system does demand that the complete panel be developed before extraction is commenced. There is therefore some danger of spontaneous heating occurring in the gate roads which are exposed to the ventilation for some considerable time. If spontaneous heating is detected along a gate road the section of the gate road walls is carefully sealed and a mixture of water and schist (0 - 4 mm) is injected behind the sealed area; this is then followed by fly ash mixed in water. Continuous monitoring of CO content of the mine air is carried out, from points throughout the panel and the mine to allow the early detection of spontaneous heating.

In summary, the operation of a closed panel system (in this case longwall caving and drawing on retreat) has reduced dramatically the dangers of spontaneous combustion in the Blanzy coal fields, an area previously plagued by mine fires. It is essential that this method be operated on retreat, otherwise ventilation leakages between the gate roads through the gob can occur, and the advantages of the closed panel system will be nullified.

6.3 Outbursts and Methane Problems

No outbursts have occurred in the Blanzy coal fields; neither has methane emission been a problem in the Darcy and Rozelay seams; the methane emission is very low at approximately 2 m^3 per ton mined. This is very low in comparison with some Western Canadian coal seams where the methane emission can be of the order of 57 - 92 m^3 per ton mined (2000 to 3250 cu ft/ton). It will be realised then that in such a thick seam in Canada, this longwall caving and drawing method could release considerable quantities of methane rapidly into the mine causing very serious ventilation problems.

6.4 Accident Statistics

Table 6.1 gives a breakdown of accident statistics from the Darcy and Rozelay mines; as a comparison, similar statistics for underground coal mining accidents in Alberta are also given where comparative figures could be derived.

According to these figures the accident rate in Darcy and Rozelay is some 8 - 9 times higher than that in Alberta underground coal mining. However, these figures may be deceptive because the basis on which an accident is defined as "reportable for statistical purposes" differs between France and Alberta.

TABLE 6.1

Accident Statistics - 1971

	Darcy	Rozelay	Alberta (underground only)
1. Absenteeism	3.2%	3.26%	-
2. Reported accidents			
Total number	268	114	35
Accidents/million tons	353	315	13
Accidents/million shifts	841	952	108.3
3. Breakdown of accidents/million shifts			
Hands	191	259	-
Lower limbs (except feet)	204	217	-
Feet	88	84	-
Body	138	117	-
Others	220	275	-
Total	841	952	-
Fractures	191	209	-
4. Breakdown of accidents by causes (per million shifts)			
Rock fall	85	200	52.6
Machines	151	58	34.0
Handling material	289	360	-
Falls of victims, traffic	182	175	21.7
Others	134	159	-
Total	841	952	108.3

CHAPTER 7. CAN THIS METHOD BE USED IN CANADA?

The preceding chapters have dealt in detail with the mining techniques and problems and the current and achievable costs in a French environment. It is necessary now to examine whether or not this method could be used in Canada, and if so under what circumstances. To do this the constraints imposed on the method by Canadian economics, Canadian geological and geographical conditions, mining conditions and mine environment must be considered.

7.1 Economic Constraints

The production, productivity and costs for a mechanized longwall face at the Rozelay mine were presented in Chapter 4 and Appendix 11. In Appendix 12 these figures have been re-calculated with Canadian conditions in mind; the following assumptions were made:-

1. It was assumed that 90% panel extraction could be achieved in Canada.

2. Canadian labour rates were taken as $50 per man shift.

3. It was assumed that in Canada the mean number of man shifts per day on the face could be reduced from 110 to 92 and, for the panel (including the face) from 248.8 shifts/day to 203 shifts/day (as detailed in Appendix 12).

4. It was assumed that capital investment costs for the panel remained the same.

On the basis of these assumptions the face and panel production, productivity and operating costs and their variation with seam thickness were re-calculated and Figures 7.1 and 7.2 show the results for the current and achievable conditions. It should be remembered that the "current" figures refer to the current face advance rates in France whereas the "achievable" figures refer to the advance rates that it has been estimated could at best be achieved.

From these figures it is therefore possible to estimate, for any seam thickness, the face and panel operating costs, production and productivity.

Allowable operating costs for any Canadian mine will vary from site to site depending on many factors. However, as an example let us consider a hypothetical mine located in the Alberta foothills, mining high grade metallurgical coal. Let us assume that this prospect is 5 miles from the railhead, necessitating a 5-mile truck haulage to the plant. The plant is 800 miles from Vancouver. A minimum coal potential for an underground mine of 1 million short tons of raw coal per year (~ 3000 tons/day) for a period of 20 years is deemed necessary to meet contract obligations. Thus 20 million short tons of recoverable reserves are needed which at 50% extraction from the seam (but 90% extraction from the panels) requires 40 million short tons of raw coal reserves to be proven. Assume that the plant recovery is 75% and the sale price is $20.00 per long ton clean coal F.O.B. Vancouver. What does

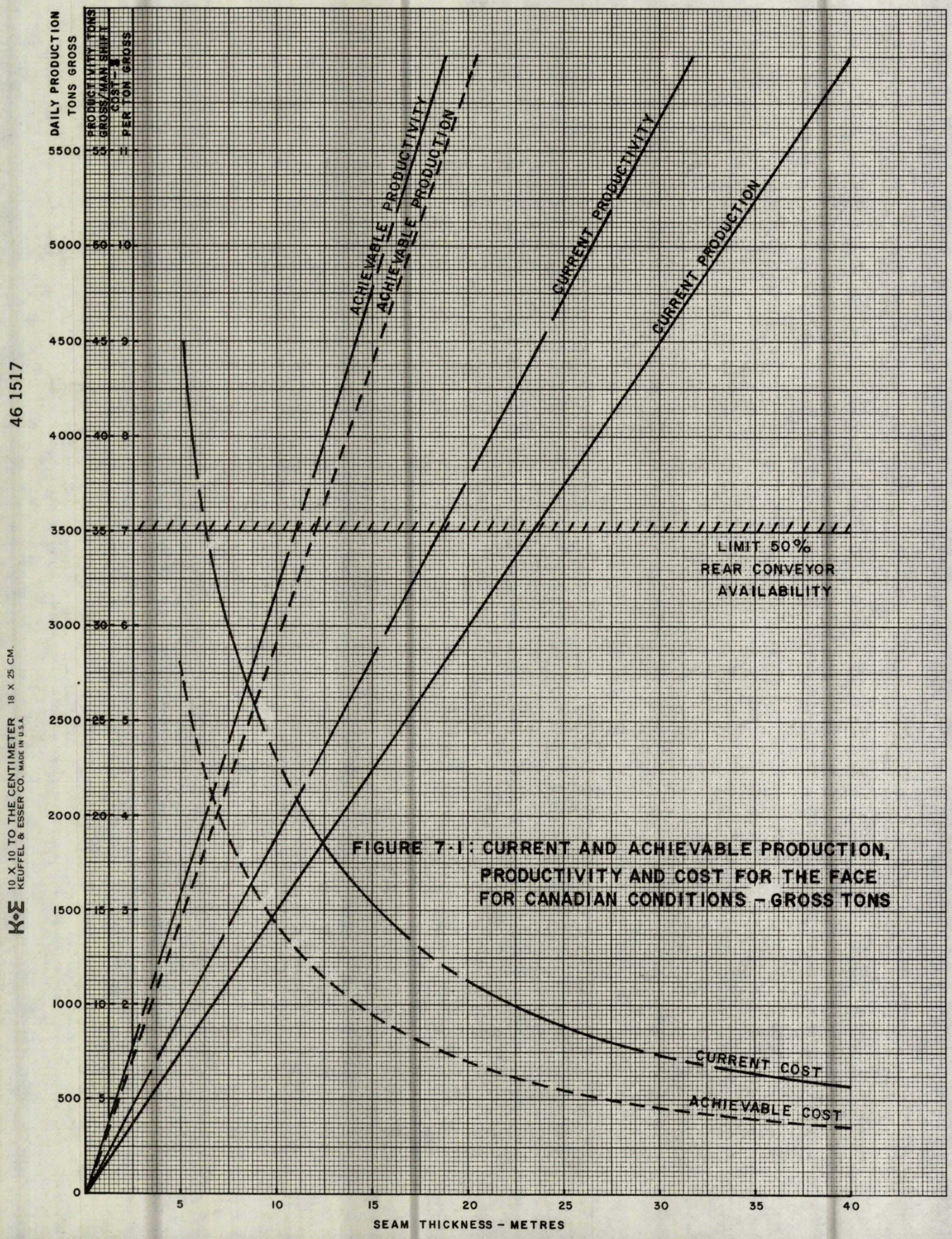


FIGURE 7-1: CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST FOR THE FACE FOR CANADIAN CONDITIONS - GROSS TONS

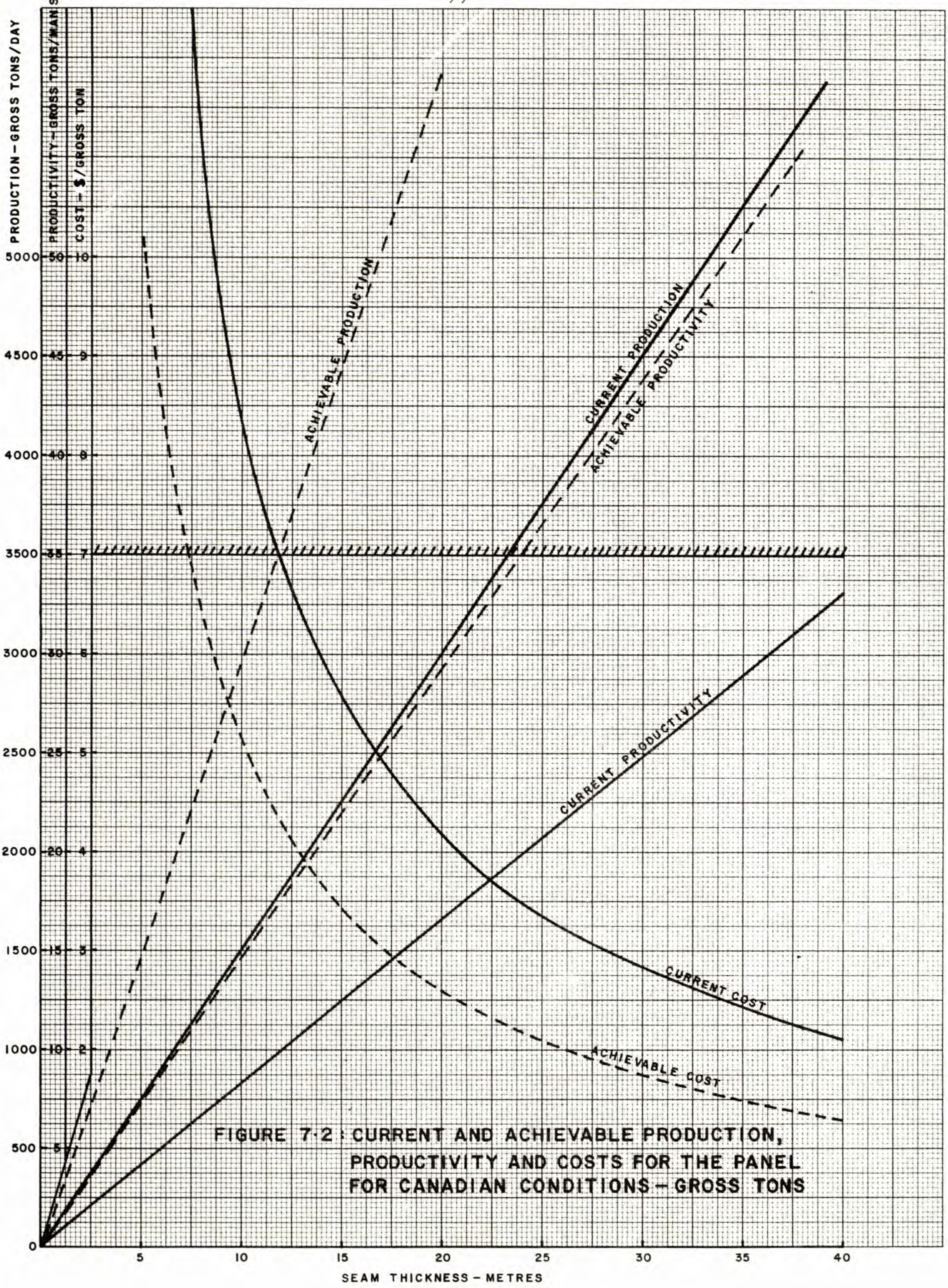


FIGURE 7·2 : CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COSTS FOR THE PANEL FOR CANADIAN CONDITIONS – GROSS TONS this mean in terms of allowable operating costs? Table 7.1 gives an estimated breakdown of costs. Approximately 26% goes to terminal, freight and royalty costs; 11.6% to cleaning plant costs and 18.3% to non operating mining costs. Thus in order to "break even" (no profit) the operating cost should not exceed about \$6.00 per short ton of raw coal, which corresponds to \$6.60 per metric ton raw coal.

In Canada the cost of getting the coal from the panel to the portal is relatively low since, unlike France where shafts must be sunk and extensive haulageways constructed, entry into the seam is invariably via an incline from the outcrop. Let us allow \$0.50 short ton raw for this cost. Hence the allowable panel operating cost for this hypothetical Canadian mine is \$5.50/short ton raw or \$6.05/metric ton raw coal. Using this cost figure and Figures 7.1 and 7.2 we can therefore determine the minimum seam thickness required to meet this economic break even point, and also determine the corresponding panel and face production, and productivity for the current and achievable face advance rates. Table 7.2 summarizes these results.

From this table it is apparent that for the "current" case a minimum seam thickness of 45.4 ft (13.75 metres) is required to break even, at this seam thickness the production would be 2280 short tons/day raw coal at a panel productivity of 12.65 short tons raw coal per man shift, a face productivity of 28.3 short tons raw coal/man shift and a face operating cost of \$3.03/short ton raw coal. For the achievable case the minimum seam thickness is 28 ft (8.5 metres) and the daily production would be 2750 short tons/day at a panel productivity of 13.45 short tons/man shift, a face productivity of 29.5 short tons/man shift and a face cost of \$3.03/short ton raw coal. For both cases 2 faces would have to be operated to meet the required production of at least 3000 short tons/day.

For this hypothetical mine then, it may be said that, from the point of view of economics only, this method is most certainly not suitable for seams less than 28 ft thick. It should most certainly be profitable for seams greater than 45 ft thick. For seams between 28 - 45 ft thick a more detailed feasibility study of the operations would be desirable, but it would be worthwhile considering this method as a potentially profitable mining method.

Figures 7.1 and 7.2 allow a similar quick economic assessment to be made for any given mine site at which the allowable operating cost can be estimated. Alternatively the mining costs for any given seam thickness can be determined, together with production and productivity, using these figures if it is desired to assess the economics for any particular seam.

7.2 Geologic Constraints

This mining method is only suitable for relatively flat lying thick coal seams. The dip should not exceed a maximum of 20° and preferably the dip should average no more than 15°. The coking coal seams of Western Canada lie in the foothills and front ranges of the Rocky Mountains; the region has been severely distorted geologically; the structure is characterized by numerous major folds and low angle thrust faults. Seams change in dip and

TABLE 7.1

Estimate of Allowable Operating Costs

	Item	$ per long ton clean		$ per short ton raw		%
1	Contract price		20.00		13.39	100 %
2	Transportation costs, etc.					
	Terminal cost	0.75		0.50		
	Freight	4.00		2.68		
	Royalty	0.29		0.19		
	UMWA	0.10		0.07		
	Total transportation, etc.	5.14		3.44		25.7%
3	Value at Railhead (1 - 2)		14.86		9.95	74.3%
4	Coal preparation costs					
	Direct operating cost	1.16		0.78		
	Indirect	0.18		0.12		
	Taxes, insurance	0.09		0.06		
	Plant amortization (6%)	0.90		0.60		
	Total coal preparation cost	2.33		1.56		11.6%
5	Maximum mining cost (3 - 4)		12.53		8.39	62.7%
6	Non operating mining cost					
	Taxes, insurance	0.30		0.20		
	Tracking pit head to plant	1.12		0.75		
	Amortization of capital	2.24		1.50		
	Total non operating mining costs	3.66		2.45		18.3%
7	Maximum allowable operating cost (5 - 6)		8.87		5.94	44.4%

TABLE 7.2

Minimum Seam Thickness, Production and Productivity for Allowable Operating Costs

ITEM	CURRENT	ACHIEVABLE
Allowable operating cost	$6.00/short ton raw $6.60/metric ton raw	
Allowable panel operating cost Assuming $0.50/short ton raw panel to portal	$5.50/short ton raw $6.05/metric ton raw	
Minimum seam thickness to meet allowable panel operating cost	13.75 metres 45.4 ft	8.4 metres 28 ft
Daily raw coal production	2075 metric tons/day 2280 short tons/day	2500 metric tons/day 2750 short tons/day
Raw coal panel productivity	11.5 metric tons/ms 12.65 short tons/ms	12.25 metric tons/ms 18.45 short tons/ms
Raw coal face productivity	25.7 metric tons/ms 28.3 short tons/ms	26.8 metric tons/ms 29.5 short tons/ms
Face operating cost	3.33 $/metric ton 3.03 $/short ton	3.33 $/metric ton 3.03 $/short ton

are normally quite steeply inclined; thickness variations can be extreme, the seams thinning in the limbs of folds and thickening in the hinge regions. The coal is intensely sheared and friable. By virtue of the overall thrust movement of the Rocky Mountains the continuity of the seams down dip can be very poor, although that along strike is often quite good.

In these types of conditions, there will be relatively few areas where the seams are sufficiently flat lying to allow this method to be used. These will mostly occur at the top of synclines and at the bottom of anticlines. In such areas it will be essential to conduct intensive exploration programs to establish the seam thickness and dip variations and to ascertain whether or not potential mining panels will be faulted out.

7.3 Mining Constraints

Given an area which satisfies the above economic and geologic constraints then there is no reason why, in principle, the mining method described here should not be applicable. The mining equipment and methods used in France should be suitable. There is, however, one key item to which an answer cannot be given here. This concerns the friability of the coal and the ability to be able to support the coal face and roof ahead of the powered supports. The face advance rates, both for the current and achievable cases used in the economic analysis, demand that few problems be encountered with the face advance. If caving occurs on the face ahead of the supports then there will be considerable delays and the required face advance rates will not be achieved. The ground control studies indicated that this was one of the main factors allowing the mechanization of the Rozelay mine and preventing so far, the mechanization at the Darcy mine. The ability to maintain good stability ahead of the powered supports will primarily depend on the natural friability of the coal and on the overburden thickness. A quantitative comparison of the coal friability in France and in Canada has not been obtained; however, there is some indirect evidence and the author's subjective opinion to suggest that, in general, the coking coals in the Rocky mountains and foothills are more friable than the French coals. If this is so, then in Western Canada, the potential use of this method could be seriously handicapped by caving on the face.

In the author's opinion no indirect method of assessing friability can give a satisfactory answer to this question of whether or not caving ahead of the supports will be a major problem; the only satisfactory answer would be obtained from an experimental coal face. It would therefore be vital to carry out studies on an experimental face before committing this method to production to meet contract needs.

7.4 Environmental Constraints

With regard to spontaneous combustion, it would appear that the Blanzy coals are considerably more susceptible to spontaneous heating than the Western Canadian coking coals; thus, in view of the good record of this method in the Blanzy coal fields, it is thought that this method should present little or no spontaneous heating problems in Western Canada, provided that the faces are operated on retreat.

The methane content of Canadian coal seams varies considerably; in seams containing little methane there should, as at Blanzy, be few ventilation problems. However, in seams which contain large quantities of methane (perhaps as much as 3000 cu ft/ton mined), this method in which large volumes of coal are caved will produce large quantities of gas in the gob and on the face which will probably pose very serious ventilation problems and may well prevent the use of this method in very gassy seams. Working with a methane filled gob may be a serious restriction and it is essential that detection systems be placed on all potential ignition sources, with power shut-off controls automatically activated if the methane concentration exceed specified limits.

It is anticipated that the dust problems on a Canadian face would be similar to those at Blanzy; i.e. if water infusion and plastic sheeting over the supports are used,then the dust level should be kept to tolerable levels. In this respect, the friable nature of Western Canadian coals may make water infusion even more effective than at Blanzy.

7.5 Conclusions

This method of longwall, bottom slice, retreat mining with caving and drawing has some potential use in Canada; this use however will be restricted to a very limited number of flat lying thick coal seams. The following restrictions are envisaged:-

(i) The method is confined to seams which dip at less than 20°.

(ii) At best, under current economic constraints, the method is restricted to seams which are at least 28 ft thick.

(iii) The faces must be operated on retreat and down dip.

(iv) Serious problems with regard to ventilation are envisaged in seams with high methane content; the method is probably restricted to low methane content seams.

(v) The friability of Western Canadian coking coals is such that caving on the face, ahead of the powered supports, could become a very serious problem; this could be a sufficiently serious problem to nullify its potential use in Western Canada.

(vi) In view of the above restrictions it is deemed essential that an experimental face be operated by this method before this method be considered as one suitable for use in Western Canada.

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8.2 Acknowledgements

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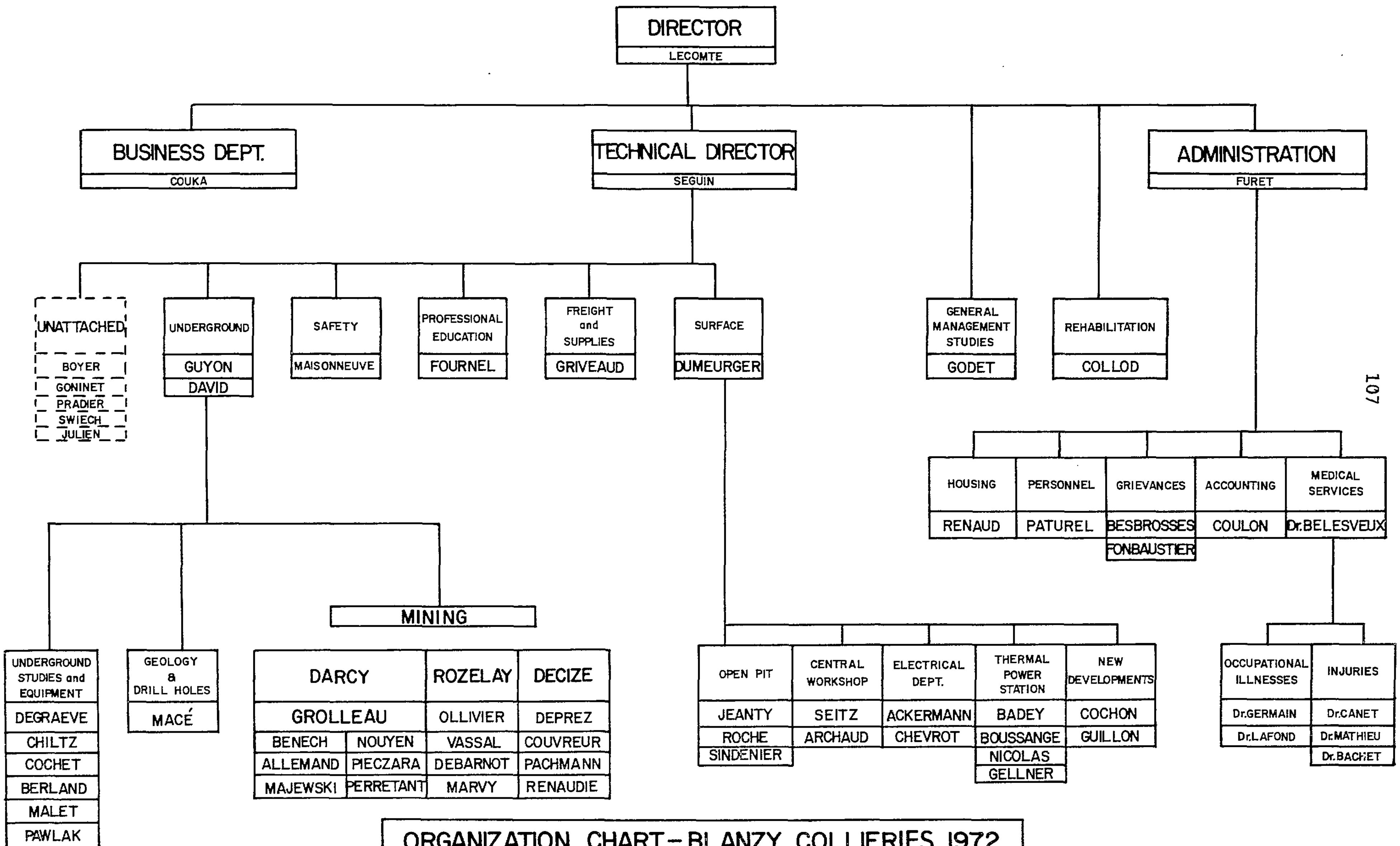
APPENDICES



ORGANIZATION CHART – BLANZY COLLIERIES, 1972

APPENDIX I

APPENDIX 2. A BRIEF HISTORY OF MINING METHODS IN THE BLANZY COAL FIELDS

Exploitation in the Blanzy coal fields first became important about 1832, at which time mining was carried out by room and pillar operations. In 1863, with the introduction of fill techniques, the 'Blanzy method' was first started; this was basically a method of horizontal slices. Variations and improvements to this 'Blanzy method' were carried out until quite recently when the method of bottom slicing with caving and drawing was introduced.

Most of the coal seams mined in the Blanzy coal fields are thick and relatively flat lying ($< 30^o$); they are sometimes gassy and can be susceptible to spontaneous combustion. The method of ascending horizontal slices allowed rapid excavation of coal and minimized the risk of spontaneous combustion.

A2.1 Extraction of Thick Seams by Ascending Horizontal Slices with Descending Sub levels

This method is illustrated in Figure A2.1. Each seam was split into descending sub levels; each sub level was mined by ascending horizontal slices. A roadway, along strike, was driven in the centre of the coal seam. Coal was won using air hammers and explosives, and was transported in 700-litre wagons. The supports were entirely wooden and the gob was filled pneumatically with schist material. Within each horizontal slice the faces were developed either parallel or perpendicular to the strike roadway.

(i) Face perpendicular to the strike roadway (Figure A2.2)

Horizontal crosscuts to the hangingwall, or the footwall, were driven perpendicular to the strike roadway at spacings of approximately 15 m apart. These crosscuts formed the working face. Each face had three working aisles; one in the process of being filled, one for transportation and one for face advance.

(ii) Face parallel to the strike roadway (Figure A2.3)

In this case two horizontal crosscuts were driven to the hangingwall (or to the footwall) and were connected by a gallery, parallel to the strike roadway, which formed the face. The face was retreated towards the strike roadway with fill being placed behind it. This method allowed more efficient transportation of coal from the face since a one-way traffic flow was created, likewise ventilation of the face was improved.

A2.2 Extraction of Relatively Thin Seams (Figure A2.4)

This method was used only in seams less than 3 m thick and at shallow depths. The face ran between the bottoms of two inclined roadways (one for intake air, the other for return air). Usually a one-way traffic system was used with empty cars and fill arriving via one incline and coal cars exiting via the other incline.

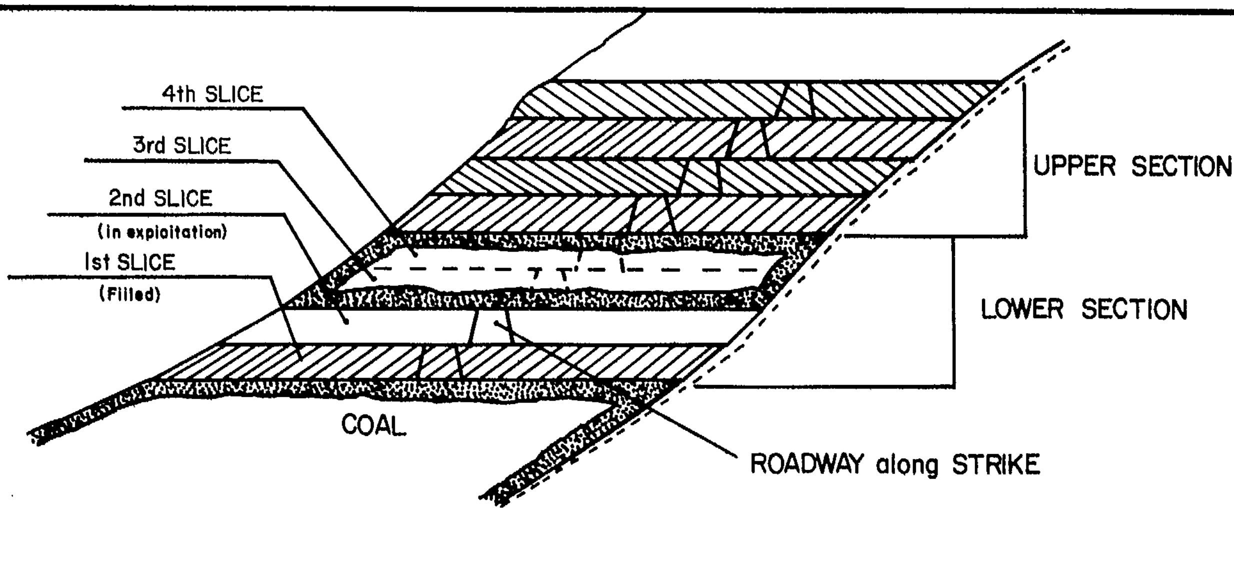


FIGURE A2.1: THICK SEAM MINING-DESCENDING SECTIONS with ASCENDING SLICES

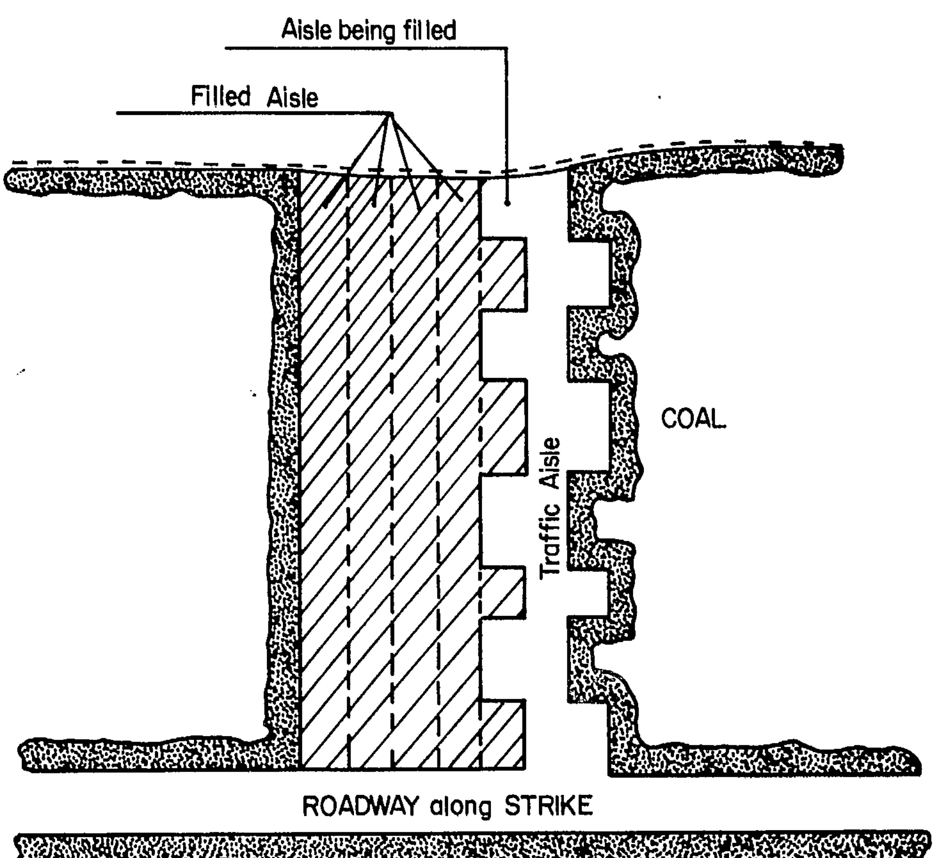


FIGURE A 2.2: METHOD USING FACE PERPENDICULAR to the STRIKE ROADWAY

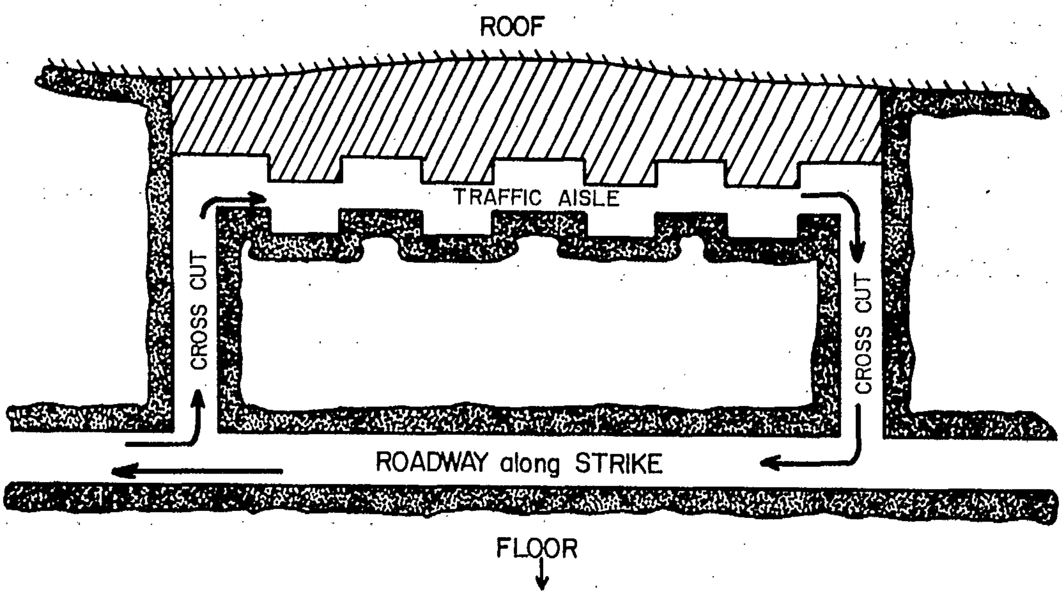


FIGURE A2.3: METHOD USING FACE PARALLEL to STRIKE ROADWAY

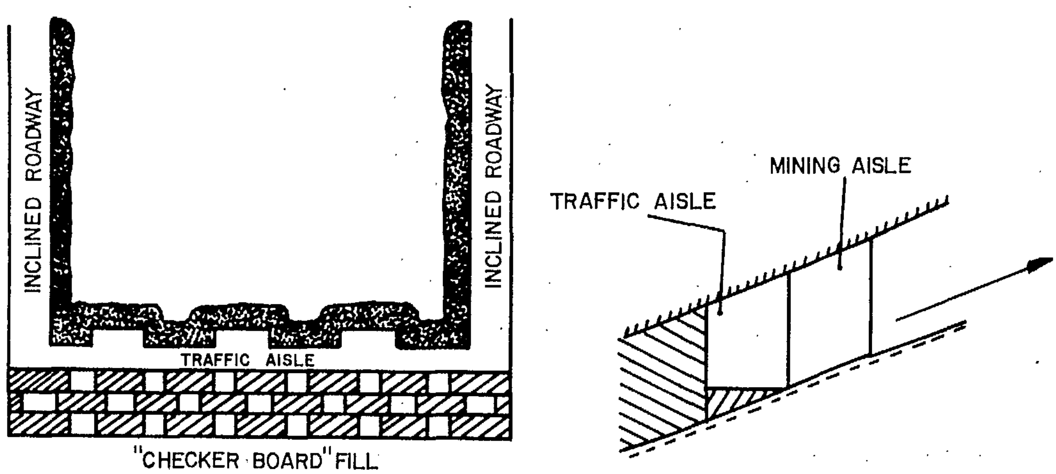


FIGURE A2.4: THIN SEAM EXCAVATION - USING RISING FACES (between roof & floor)

The face was retreated to the rise, Complete filling of the gob was not practised. A length of gob of about 6 m was filled and a gap of 4 m was left. In the next filling aisle the 6 m length of fill was placed opposite the 4 m void in the preceeding aisle; thus a "checkerboard" of fill in the gob was created.

A2.3 Mechanization of the 'Blanzy Method'

The traditional Blanzy method presented three major inconveniences: it subjected the miner to intense physical effort; with a method perfected over many generations it was difficult to make new innovations; furthermore mechanization was incompatible with the hand filling methods that were practised. Mechanization was therefore introduced to increase productivity, although the basic mining method remained the same.

For the fill operations there were two alternatives - either eliminate the fill or mechanize its transportation and placement. In the first case the gob area was allowed to cave. In the second case either compressed air was used to transport and place the fill pneumatically or the fill was mechanically placed by "slinging" into the gob. Coal transportation was improved by replacing coal cars with chain conveyors on the face and belt conveyors in the roads.

(i) Caving in thin seams (Figure A2.5)

An incline was developed in the seam and two horizontal roadways, 100 m apart, were driven to form the head and tailgates of the face. The face connected these two gate roads and was retreated towards the incline; the roof was allowed to cave in the gob.

(ii) Caving in thick seams (Figure A2.6)

For thick seams a method of descending horizontal slices was used. A floor of boards and mesh was laid during the advance of the face; face supports were retrieved by winches and the roof was allowed to cave behind the face. The mat on the floor formed the roof for mining the next slice under the caved roof.

(iii) Ascending slices with pneumatic fill (Figure A2.7)

As with the traditional Blanzy method, the seam was split into descending sub levels, each sub level was mined by a large number of ascending horizontal slices (15 - 20 slices per section instead of 4 - 5 previously). Fill was delivered pneumatically to the face and projected into the gob. A mesh and brattice cloth curtain retained the fill in position and prevented the fill spilling into the working aisle.

(iv) Descending slices with pneumatic fill (Figure A2.8)

This method was used only in flat seams at shallow depths. The descending slices were started immediately under the roof. A mat, comprising a triple row of boards covered with mesh, was laid on the floor of the slice. This served to contain the fill during the mining of the slice underneath.

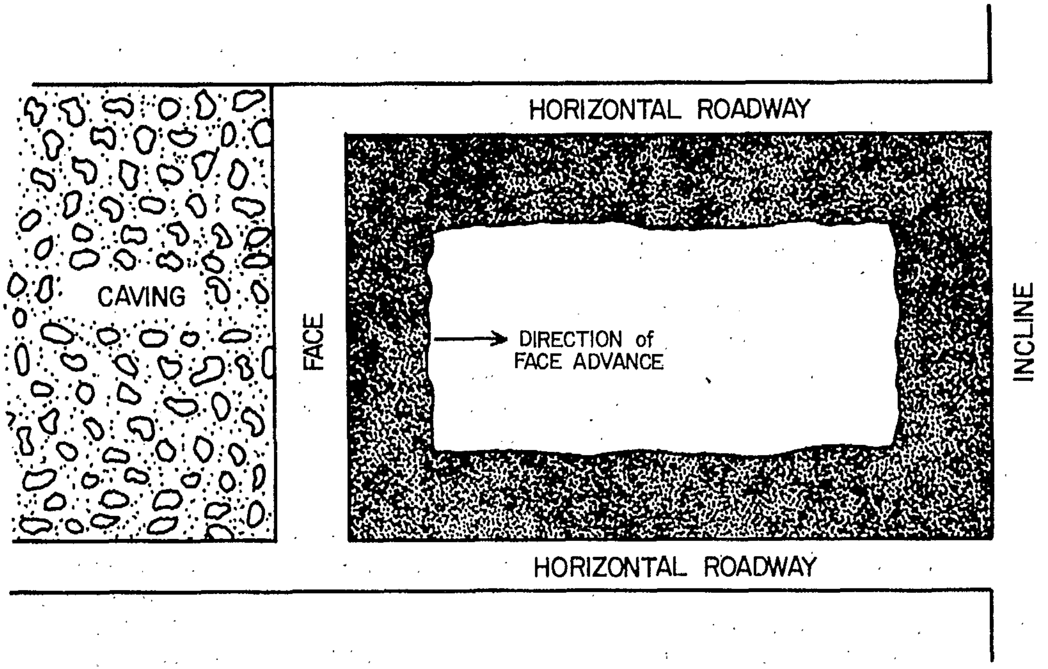


FIGURE A2.5 : MINING of THIN SEAMS with CAVING in the GOB

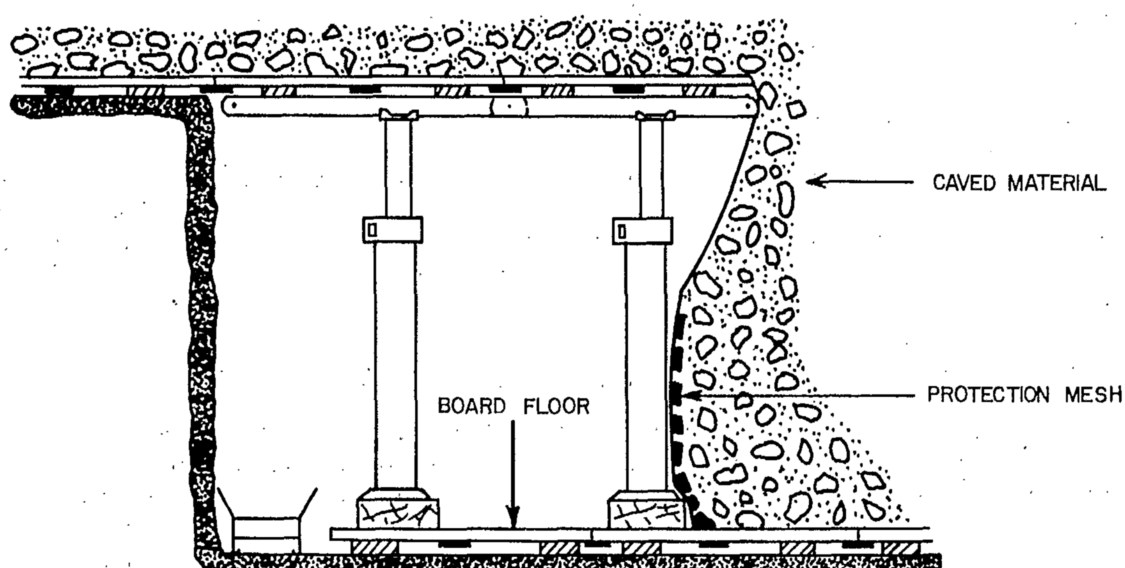


FIGURE A2.6 : MINING of THICK SEAMS USING DESCENDING HORIZONTAL SLICES & CAVING

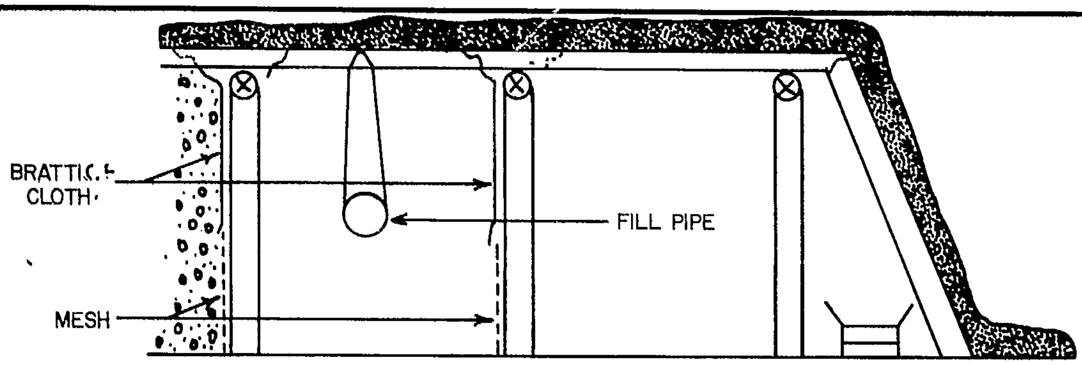


FIGURE A2.7: FACE with PNEUMATIC FILL PLACEMENT

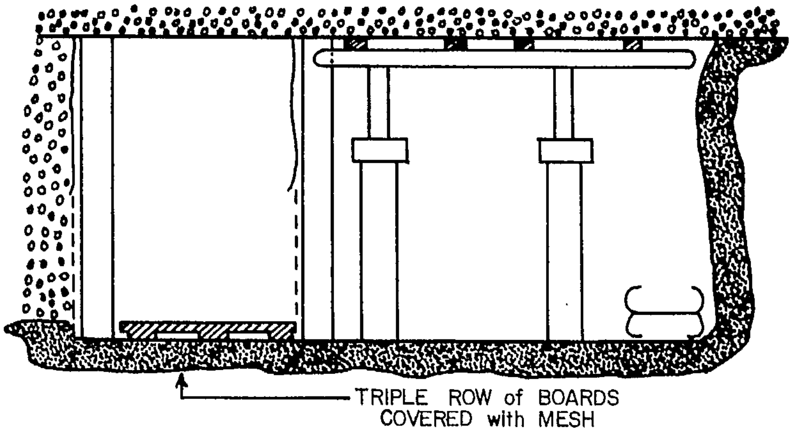


FIGURE A2.8: METHOD of DESCENDING SLICES with PNEUMATIC BACK FILL

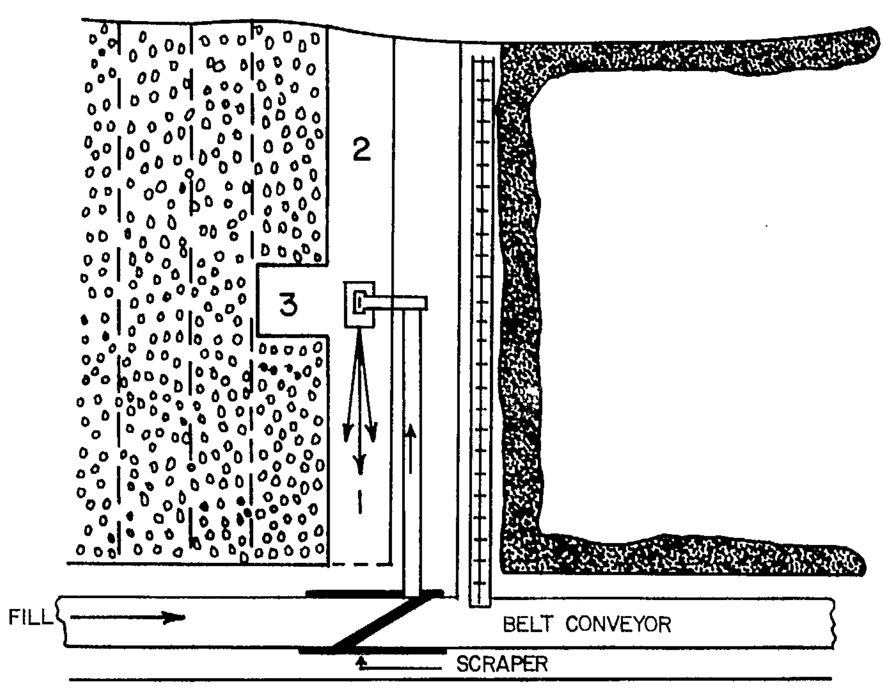


FIGURE A 2.9: PLACING of FILL MECHANICALLY

(v) Mechanical fill

Placing of fill mechanically was used when the limit of the air compressors was reached for the pneumatic placing of fill. The method of mining was again one of descending sub levels with ascending horizontal slices. Belt conveyors in the roadways for the transport of coal were reversed to bring fill to the face; it was thus only possible to place fill during the non production night shift. A scraper at the junction with the face transferred the fill to a face conveyor which then delivered it to the mechanical fill placer, as shown in Figure A2.9.

A2.4 The Introduction of Caving and Drawing Methods

In 1964 the longwall caving and drawing method was introduced for the first time in this area. Because the coal was relatively hard it was initially thought that, in thick seams (> 9 m), it would be necessary to mine in two slices as illustrated in Figure A2.10 (a). The first longwall slice was retreated about 3 m above the footwall to induce the overlying coal to cave; a floor of wood and mesh was put down to form the 'roof' for the second slice. The second footwall slice was mined at about 30 m behind the first face. The caved coal was drawn, through windows cut in the mesh, onto a rear conveyor on the second face. At this stage of development the face was supported by both hydraulic and friction props as shown in Figure 2.10 c, d, e and f; these figures also illustrate the mining cycle. Only one conveyor was used on the face, the face advance and the drawing being distinctly different elements of the mining cycle.

For coal seams less than 9 m thick the method of using two slices, each 3 m thick, was not practical and for seams between 3 and 9 m it was decided to try a single footwall slice, as shown in Figure A2.10 (b); both caving and drawing were achieved by the one face. The face support system and the mining cycle were the same as above except that, in this case, a mesh was placed over the top of the supports sagging to the footwall after passage of the supports, to prevent the caved coal spilling onto the face. As before, the caved coal was drawn through windows cut in the mesh, onto the conveyor.

Experience with this method indicated that, in the Blanzy conditions, even in very thick seams it was not necessary to use two slices; the single footwall slice was adequate to ensure caving of the overlying coal, although, on occasion, it was necessary to induce caving by shot firing.

The current mining methods, described in detail in the text, are merely improved mechanized versions of this longwall, bottom slice, caving and drawing method.

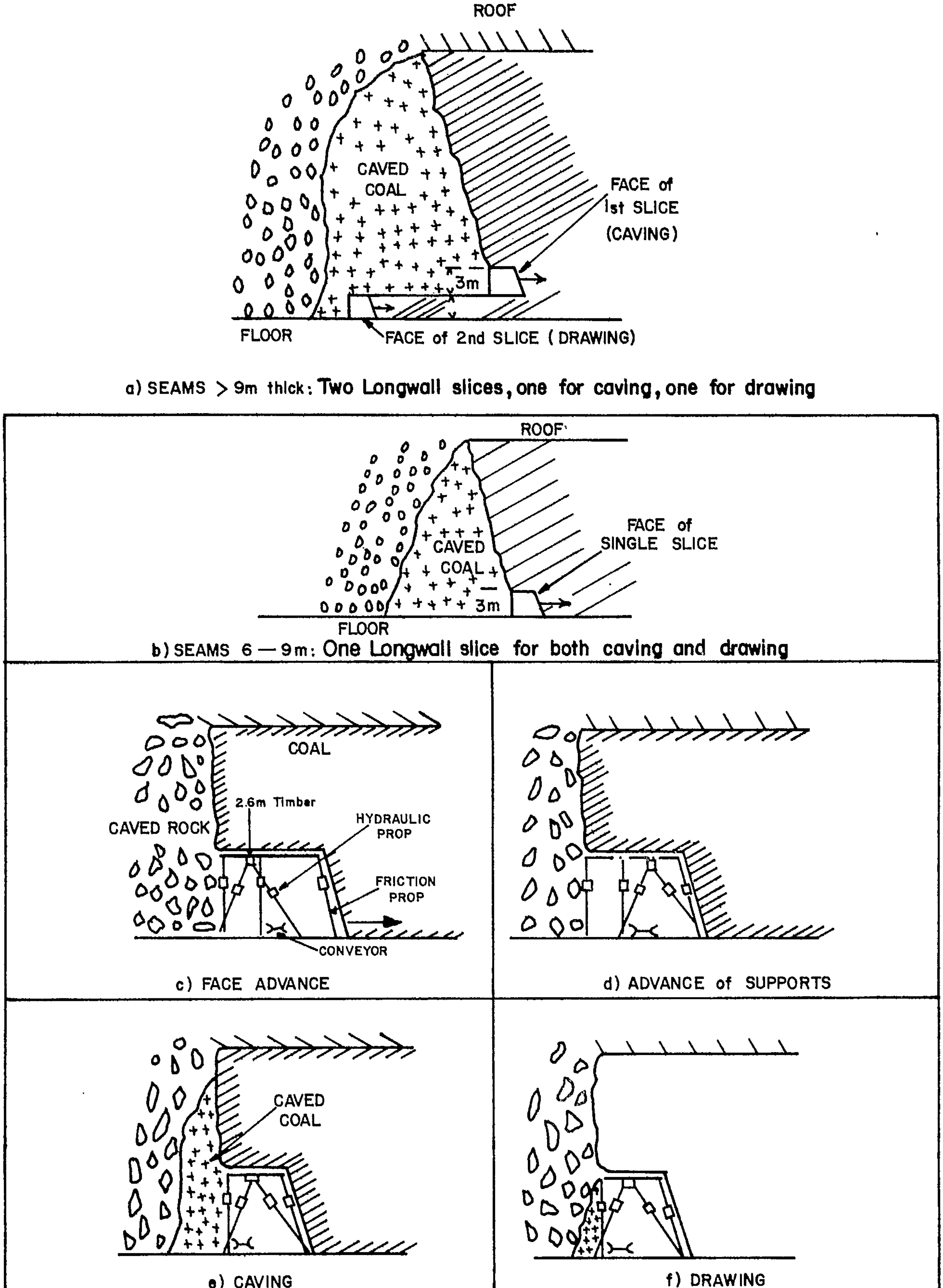


FIGURE A2.10: LAYOUT and MINING CYCLE for EARLY CAVING and DRAWING SYSTEMS

APPENDIX 3. LA STEPHANOISE DIESEL POWERED MONORAIL FOR TRANSPORT OF SUPPLIES

A3.1 Description

Both the Darcy and Rozelay mines make use of diesel powered monorails for the transport of supplies from the shaft loading areas to within 50 metres of the face, unloading in the headgate. This appeared to the author to be an exceedingly efficient means for supply transportation. Since diesel powered monorails are not currently in use in Canada a brief description and performance characteristics are given below.

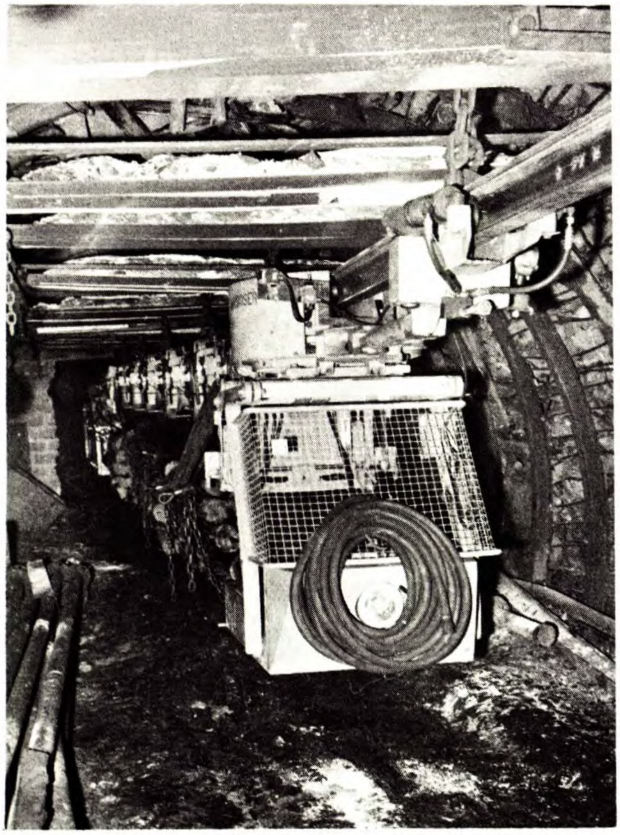
Figure A3.1 (a) shows a photograph of the Stephanoise type 830 diesel powered monorail in use. Power is provided by a 50-kW diesel engine directly coupled to a hydrostatic pump. The output of this pump drives the motors onto which are connected the driving wheels. These wheels are spring loaded into the web of the monorail. Depending on the loads to be towed, the slope inclinations and the speeds required, the locomotive can be built with 4, 6 or 8 driving wheels. As the frame for all three is the same, changing from one to the other is easily carried out.

The system has two drivers' cabs, one attached to the diesel unit, and the other may be separated from the locomotive and placed at the end of the train. The train is therefore never "backed", as the driver is always in the cab facing the travel direction. In a case where the rear cab of the train has to pass over a load which has only just been discharged from the trolleys and not yet moved from the passage of the train, it is possible to fold up the rear cab, using a compressed air piston, to within 700 mm of the rail. Figure 3.1 (c) shows a cab in this folded up position.

A variable number of trolley units are generally inserted between the diesel unit and one of the cabs (see Figure 3.1 (b)). These carriages can be manually or pneumatically loaded. Loading can be completely mechanized by the use of hydraulically winch loaded carriages (Figure 3.1 (b)) which are fed through an auxiliary pump driven by the diesel. The lifting capacity of each hydraulic carriage is 2 tons.

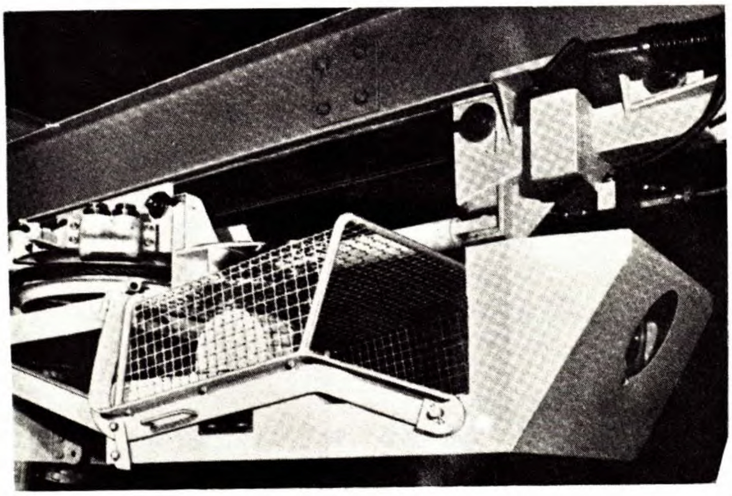
A3.2 Technical Specifications

1. 3860 cu cm diesel motor producing 50 kW at 2,200 rpm;
2. Pump, 186 cu cm/R displacement, i.e. 410 litres/minute;
3. 8 hydraulic motors each of 314 cu cm displacement;
4. Maximum speed 11.6 km/hour;
5. At maximum slope of 30%, load towed is 8 tons;
6. Width at rail level - 81 cm;
7. Width of diesel frame unit - 90 cm;
8. Ground to rail height - 136 cm;
9. Minimum curving radius in the vertical plane, 7 m;
10. Minimum curving radius in the horizontal plane, 4 m;
11. Total weight 5.8 tons (without loading carriages);
12. Total weight, of 6 carriage train, 7.4 tons (no pay load).



(a)

(b)



(c)

FIGURE A3.1 : La Stephanoise monorail for transport of equipment

A3.3 Braking Systems

Three braking systems have been fitted to the locomotive to ensure safety.

1. Hydraulic braking on the motors by reducing or cutting completely the rate of flow from the pump by means of the manual control. Experience has shown that the locomotive with its load can easily be handled and kept stationary on a 30% slope.

2. A heavy duty caliper brake acting on the web of the monorail. This brake consists of two jaws acting in the web of the monorail, it is capable of a braking effort of 5 tons on dry rail. This allows a train of 15 tons to be slowed down and stopped. The brake is applied manually, or automatically if the train should trip through overspeed.

3. An emergency braking system is fitted at each end of the train and works automatically when the other two systems have failed. The brake consists of two carriages; the first of these has two cams with teeth that are brought in contact with the web of the monorail by means of two springs, should the train trip through overspeed. The second carriage, which continues to move with the train, comprises a winch onto which is wound a 25-mm cable attached to the first carriage. This winch has a permanently applied disc brake. When the train passes overspeed, the first carriage grips the rail, allowing the winch on the second carriage to unwind. The disc brake creates a torque sufficient to stop the train. This braking system is independent of the condition of the rail and the braking effort increases with each revolution that the winch unwinds. A load of 13 tons descending a slope of 30% is stopped in 7 metres.

A3.4 Performance

Figure A3.2 shows manufacturers performance specifications for the type 830 unit in terms of total weight (unit + pay load) against speed for different slope inclinations.

The table below indicates some average performance figures obtained from the Rozelay and Darcy mines for two months of 1972.

Month	Rozelay		Darcy	
	Aver. No. hours worked per working day (3 locomotives)	Ton kilometres per man shift	Aver. No. hours worked per working day	Ton kilometres per man shift
Sept. - 1st 15 days	34.9	4.55	20 (3 loco.)	3.18
- 2nd 15 days	28.5	4.29	4.4 (1 loco.)	4.05
Oct. - 1st 15 days	29.6	4.06	20 (3 loco.)	3.36
- 2nd 15 days	26.4	4.35	4.5 (1 loco.)	4.95

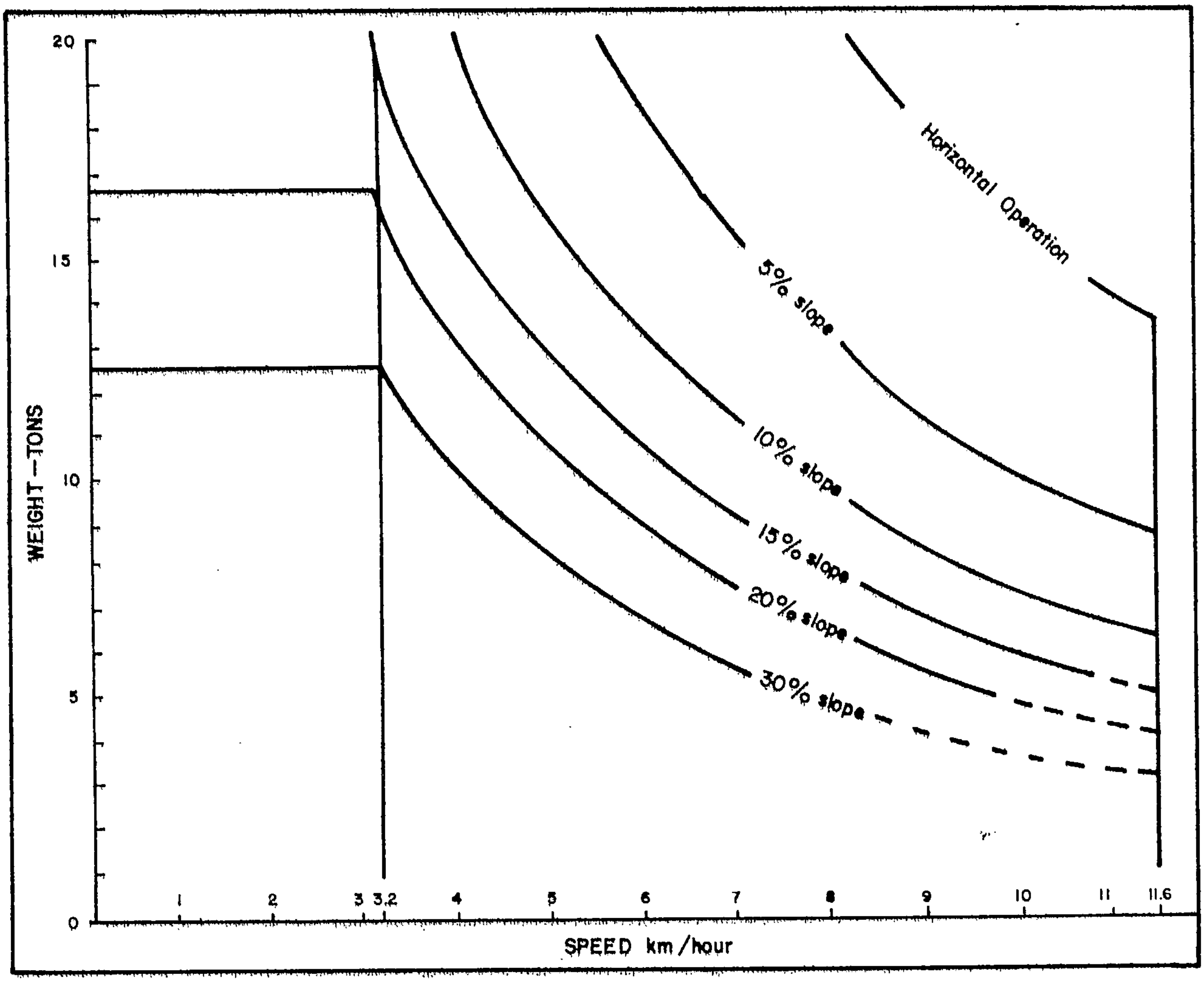


FIGURE A3.2: PERFORMANCE of STEPHANOISE DIESEL POWERED MONORAIL TYPE 830

A3.5 Capital Costs

	Item	Weight	Price Francs	Price ~ $
1.	1 power unit with 8 motors of 400 cc, without cabin, with parking brakes.	4.35 t	203,600	40,720
2.	1 folding cabin with pneumatic control.	0.27 t	16,750	3,350
3.	1 folding cabin without pneumatic control.	0.23 t	13,560	2,712
4.	6 loading carriages with hydraulically operated winches, including brakes (unit mass 0.215 t, unit price 10,900 F)	1.290 t	65,400	13,080
5.	Complete auxiliary equipment for 1 train, comprising - 25 mm diameter cable and attachments, for connecting the train elements. - coupling rods with pulleys and axles. - hydraulic lines for operation of parking brakes and 6 loading carriages.	0.60 t	8,210	1,642
6.	2 cable brakes with: - 8 brake shoes. - disc brake. - release mechanism for overspeed trip.	1.20 Kg	33,800	6,760
7.	1 hand pump for re-setting emergency brake.		500	100
	Total	7.4 t	341,820	68,364

APPENDIX 4. SPECIFICATIONS, PERFORMANCE AND COSTS OF POWERED SUPPORTS

Four types of walking supports are used in Blanzy collieries; these are: S.M.F., Westfalia type 68, Westfalia high bearing capacity and Westfalia very high bearing capacity; these are shown in Figures A4.1, A4.2, A4.3 and A4.4. All use double action props with monolithic roof bars; the props are either fixed in a rigid frame or set on individual pads. An additional jack for each line of props activates the articulated second roof bar, or "banana", which provides protection to the rear conveyor. The distance between the centres of the props rows, parallel to the face, varies from 70 - 80 cm depending on the manufacturer. Double action pistons link the two rows of any one support to allow the props to be straightened to the vertical position. Advancement of the front and rear conveyors is done by jacks connected to the base of the units.

The Westfalia high bearing capacity support has also extra jacks in the roof bars which allow an extension to be advanced so that an additional roof span may be supported without advancing the support (Figure A4.3).

Hydraulic power is supplied to the Westfalia supports by a group of two WOHA pumps of 60 litres/minute capacity each. For the SMF supports two PMH 12 pumps of an individual 80 litres/minute capacity are used. The hydraulic fluid is 5% water emulsion with Castrol KS oil.

In all cases the prop setting load and the nominal yield load are close together. Table A4.1 gives the complete specifications of each of the support types.

Table A4.2 gives capital costs, maintenance costs and performance data for each of these support types.

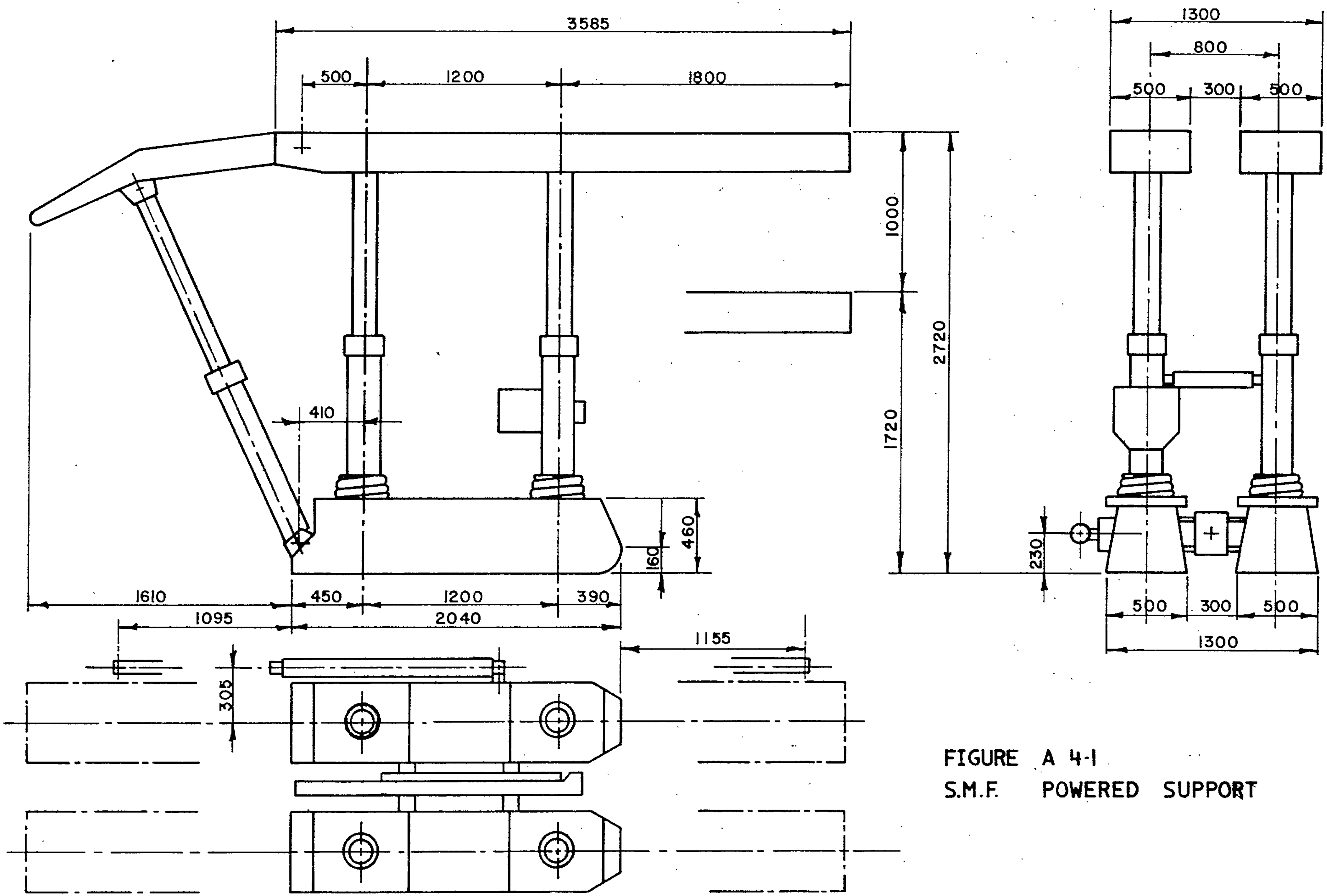


FIGURE A 4-1
S.M.F. POWERED SUPPORT

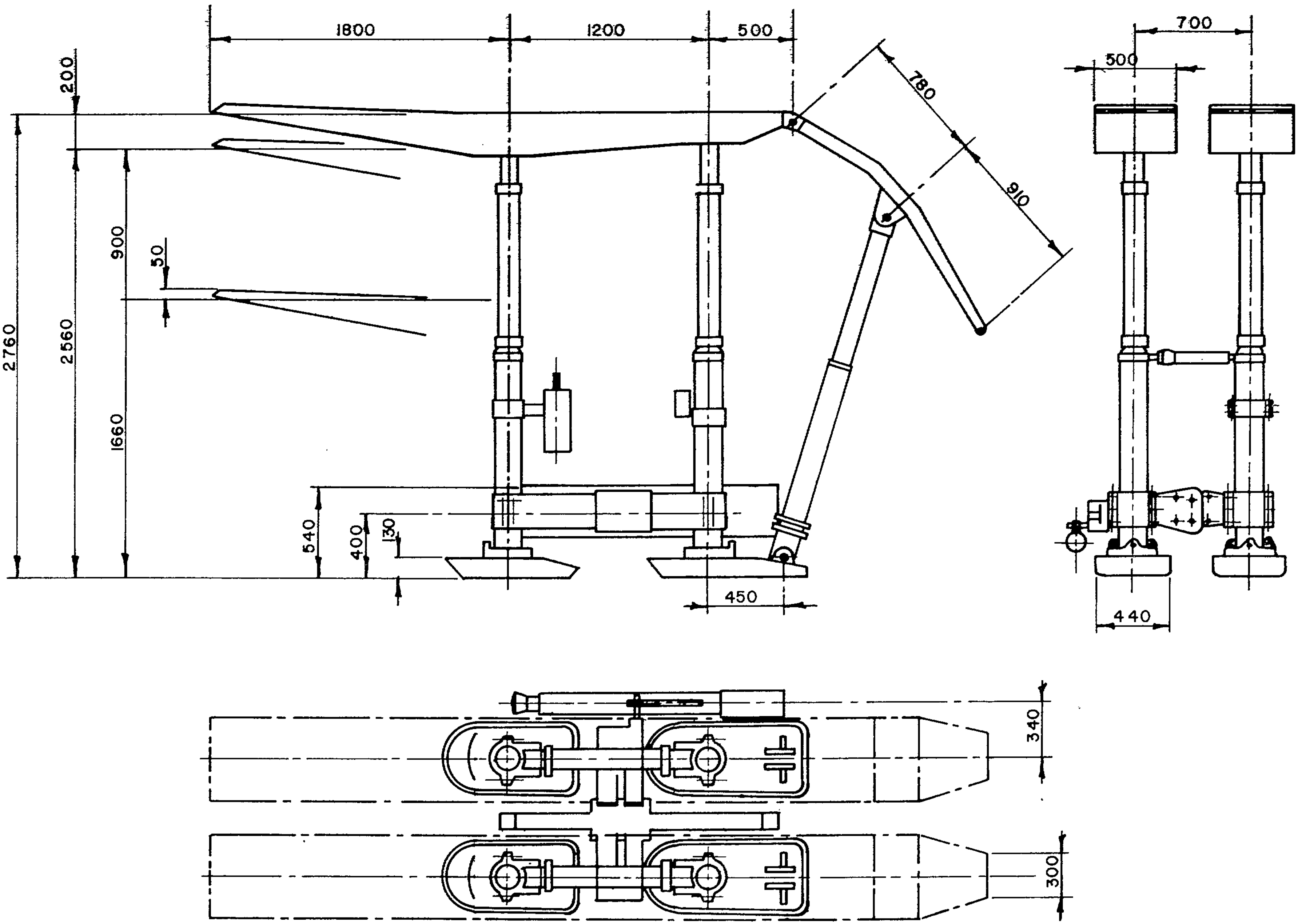
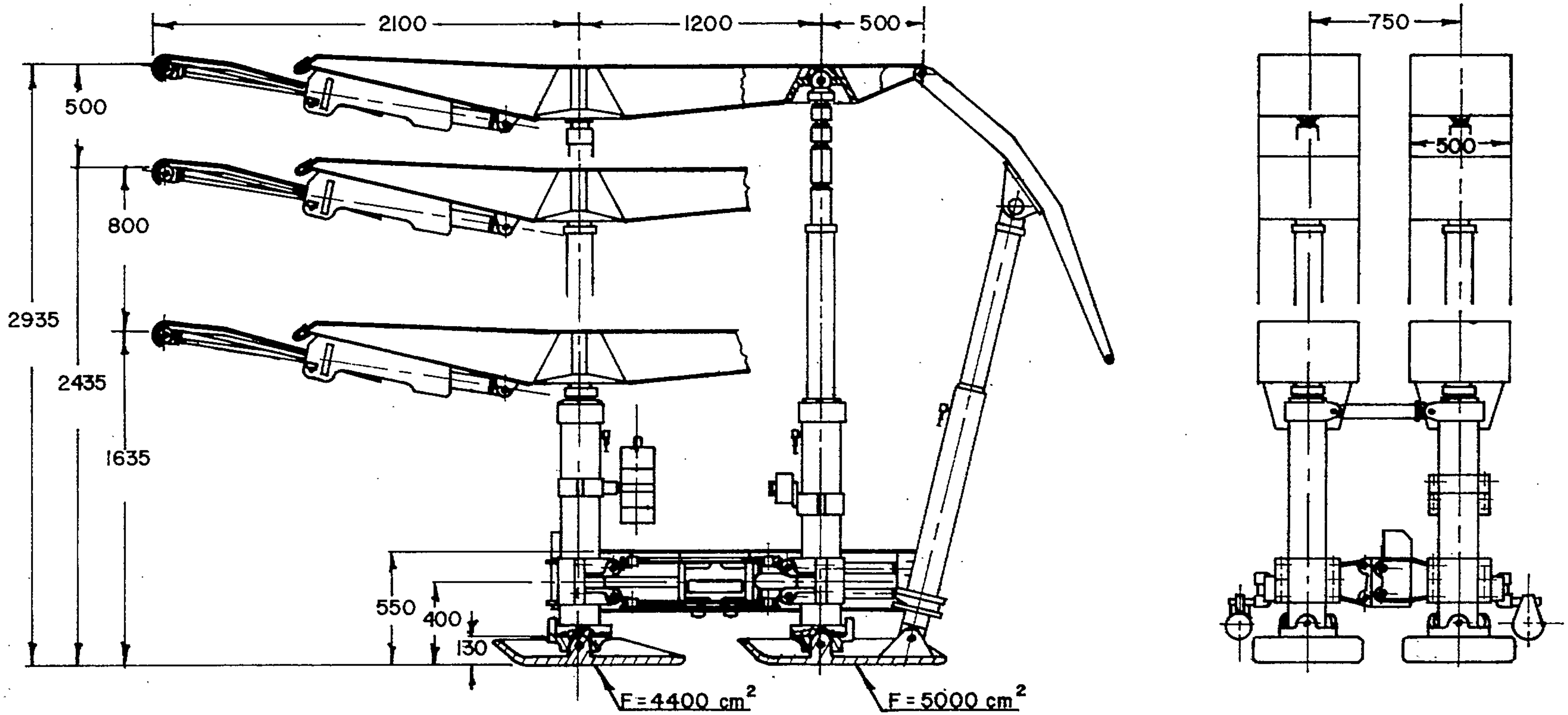


FIGURE A4·2 : WESTFALIA POWERED SUPPORT TYPE 68



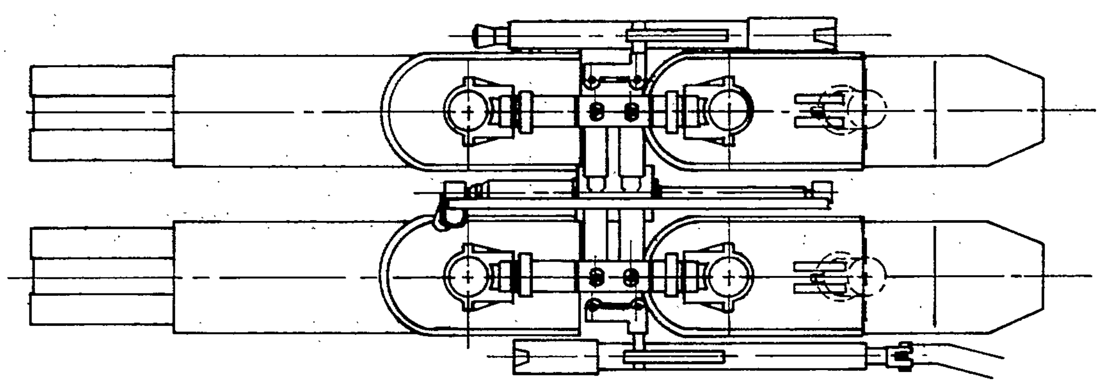


FIGURE A 4·3
WESTFALIA POWERED SUPPORT
HIGH BEARING CAPACITY

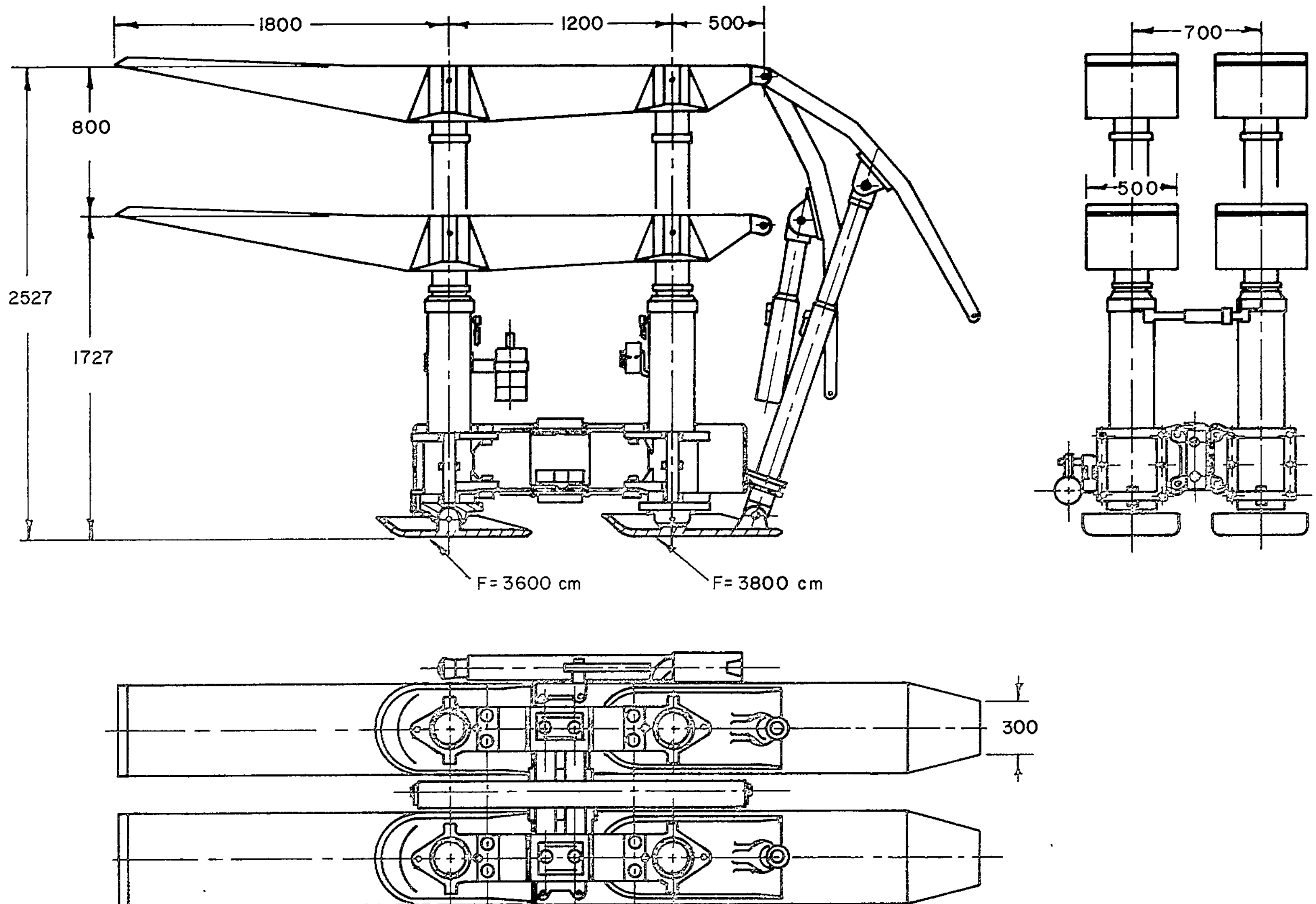


FIGURE A 4·4 : WESTFALIA POWERED SUPPORT - VERY HIGH BEARING CAPACITY

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TABLE A4.1

Comparison of Specifications of Self Advancing Supports

	Item \ Support Type		S.M.F.	Westfalia Type 68	Westfalia High Bearing Capacity	Westfalia Very High Bearing Capacity
1	THE SUPPORT					
	1.1 Height of support	closed open open with extension	1720 mm 2720 mm	1660 mm 2560 mm 2760 mm	1635 mm 2435 mm 2935 mm	1727 mm 2527 mm
	1.2 Distance between legs	front to rear between rows	1200 mm 800 mm	1200 mm 700 mm	1200 mm 750 mm	1200 mm 700 mm
	1.3 Width of support		1300 mm	1200 mm	1250 mm	1200 mm
	1.4 Total weight of support		6450 Kg	4500 Kg	5250 Kg	5900 Kg
	1.5 Weight of 1 canopy (*with extension)		1000 Kg	600 Kg	780 Kg*	710 Kg
	1.6 Weight of drawing assembly (Banana prop + piston)		225 Kg	360 Kg	290 Kg	300 Kg
	1.7 Advance step of support		650 mm			
	1.8 Allowable tilt	to front to rear	10° 10°			
2	THE PROPS					
	2.1 Double action type	- diameter - stroke - weight	135/155 mm 1000 mm	125/140 mm 900 mm	160/140 mm 800 mm 250 Kg	200/180 mm 800 mm 387 Kg
	2.2 Section area		144-189 cm^2	31-154 cm^2	201-154 cm^2	314-60 cm^2
	2.3 Setting load applied with 320 bars (4800 psi) pressure		60 t	50 t (325 bars)	64.3 t	100.5 t
	2.4 Nominal release load (under x bars)		70 t (320 bars)	65 t (420 bars)	96.5 t (480 bars)	109.9 t 350 bars
	2.5 Inclination of props	to the front to the rear	10° 10°			
	2.6 Total load applied by the support at the setting pressure (bars)		240 t (320 bars)	200 t (325 bars)	257.2 t (320 bars)	402 t (320 bars)

TABLE A4.1 Continued

	Item \ Support Type		S.M.F.	Westfalia Type 68	Westfalia High Bearing Capacity	Westfalia Very High Bearing Capacity
	2.7 Total load applied by the support at the release pressure (bars)		280 t (375 bars)	260 t (420 bars)	388 t (480 bars)	439.6 t (350 bars)
	2.8 Bearing on the base at release load		18 bars 20 t	23 bars	21.8 bars 12 t	30.5 bars 12 t
3	(A) SUPPORT MOVEMENT GIVEN BY PISTON					
	3.1 Double action piston	diameter stroke weight	75/110 mm 650 mm	75/60 mm 650 mm	90/70 mm 700 mm 76.4 Kg	90/70 mm 700 mm 76.4 Kg
	3.2 Face applied by piston at 320 bars differential pressure		16 t	5.2 t (325 bars)	8.2 t	8.2 t
	(B) SNAKING OF CHAIN CONVEYOR BY DIFFERENTIAL PISTON					
	3.3 Double action piston	diameter stroke weight	50/70 mm 1000 mm	63/80 mm 1000 mm	80/63 mm 1000 mm 63 Kg	30/63 mm 1000 mm 63 Kg
	3.4 Force applied by ram at 320 bars differential pressure		12 t	16 t (325 bars)	16 t	16 t
4	ACCESSORIES					
	4.1 Realignment of legs provided by 2 identical action rams	diameter stroke	50/40 mm 377 mm	50/40 mm 256 mm	63/40 mm 275 mm	63/40 mm 280 mm
	4.2 Modification for drawing					
	4.2.1 Length of Banana		1700 mm	1620 mm	1620 mm	1620 mm
	4.2.2 Double action ram	diameter stroke weight	110/120 mm 970 mm	100/110 mm 750 mm	110/100 mm 750 mm	110/100 mm 750 mm 110 Kg
	4.2.3 Force applied at 320 bars			30 t (325 bars)	30.4 t	30 t
	4.3 Hydraulic tilting given by 2 identical rams	diameter stroke	60/83 mm 172 mm			
	4.4 Hydraulic canopy extension					
	4.4.1 Useable stroke				700 mm	
	4.4.2 Hydraulic piston for extension	diameter stroke			63/50 mm 700 mm	

TABLE A4.2

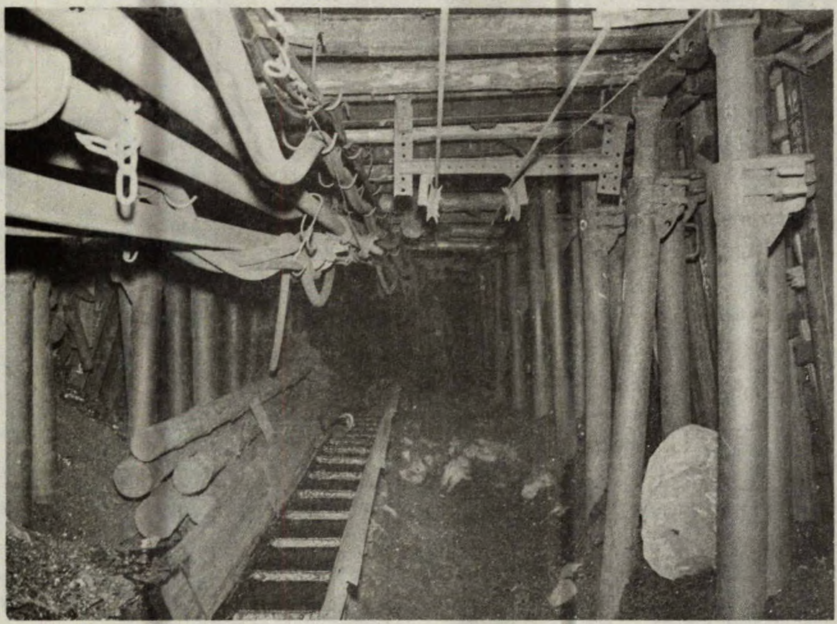
Capital and Maintenance Costs, Performance Data for Various Powered Supports

		Support Type / Item	S.M.F.	Westfalia Type 68	Westfalia High Capacity	Westfalia Very High Capacity
NUMBER AND UTILIZATION	1	Total number of supports (inventory)	171	201.5	60	15
	2	Coefficient of utilization (simple)	0.913	0.877	0.883	0.966
	3	Coefficient of utilization (actual)	0.873	0.864	0.885	0.966
CAPITAL, AMORTIZATION & FINANCIAL CHARGES	4	Capital cost per support	F43,960 $ 8,972	F43,145 $ 8,629	F67,291 $13,458	F57,600 $11,520
	5	Amortization & financial charges per day per support (theoretical)	41.61 F/day 8.32 $/d	40.84 F/day 8.16 $/d	63.70 F/day 12.74 $/d	54.52 F/day 10.90 $/d
	6	Amortization & financial charges per day per support (actual)	47.66 F/day 9.53 $/d	47.26 F/day 9.45 $/d	71.91 F/day 14.38 $/d	56.38 F/day 11.28 $/d
	7	Amortization & financial charges per ton net produced	3.43 F/ton 0.69 $/t	4.36 F/t 0.87 $/t	3.86 F/t 0.77 $/t	4.05 F/t 0.81 $/t
MAINTENANCE COSTS ON TOTAL PROP INVENTORY	8	Labour	F61,050 $12,210	F96,602 $19,320	0 0	F3,945 $ 789
	9	Replacement parts - colliery	F114,606 $ 22,921	F26,159 $ 5,231	0 0	F1,260 $ 252
	10	Replacement parts - outside	F510,008 $102,000	F106,352 $ 21,270	F1,191 $ 238	F9,281 $1,856
	11	Total cost of replacement costs (9 + 10)	F642,615 $128,523	F132,511 $ 26,502	F1,191 $ 238	F10,541 $ 2,108
	12	Total maintenance costs (11 + 8)	F685,666 $137,133	F229,113 $ 45,822	F1,191 $ 238	F14,487 $ 2,897
	13	Labour maintenance cost per day per support (8 ÷ (1 x days in use n)	3.09 F/d/s 0.62 $/d/s	4.20 F/d/s 0.84 $/d/s	0 0	2.06 F/d/s 0.41 $/d/s

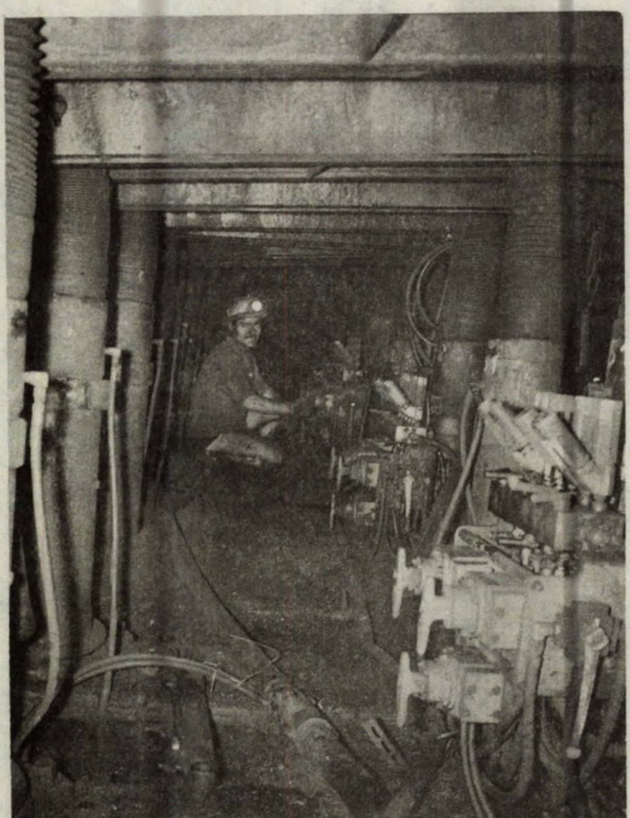
TABLE A4.2 Continued

		Item \ Support Type	S.M.F.	Westfalia Type 68	Westfalia High Capacity	Westfalia Very High Capacity
	14	Replacement parts, cost per day per support (11 ÷ (1 x n))	31.68 F/d/s 6.34 $/d/s	5.76 F/d/s 1.15 $/d/s	0.34 F/d/s 0.07 $/d/s	5.50 F/d/s 1.10 $/d/s
	15	Maintenance costs/day/support (12 ÷ (1 x n))	34.78 F/d/s 6.96 $/d/s	9.96 F/d/s 1.99 $/d/s	0.34 F/d/s 0.07 $/d/s	7.56 F/d/s 1.51 $/d/s
	16	Cost of maintenance per ton produced	2.50 F/t 0.50 $/t	0.92 F/t 0.18 $/t	0.01 F/t 0.002 $/t	0.54 F/t 0.11 $/t
UNDERGROUND SUPPLIES	17	Costs of hydraulic fluid	F64,999 $13,000	F14,061 $ 2,812	F1,570 $ 314	0 0
	18	Cost of hoses, fittings and adaptors	F145,940 $ 29,188	F99,221 $19,844	F784 $159	F3,620 $ 724
	19	Replacement parts	F39,363 $ 7,872	F161,416 $ 32,283	F22,400 $ 4,480	F1,018 $ 204
	20	Total underground supply costs (17 + 18 + 19)	F250,293 $ 50,059	F274,700 $ 54,940	F24,754 $ 4,951	F4,638 $ 928
	21	Supply costs per day per support 20 ÷ (1 x n)	12.69 F/d/s 2.54 $/d/s	11.95 F/d/s 2.39 $/d/s	7.16 F/d/s 1.43 $/d/s	2.42 F/d/s 0.48 $/d/s
	22	Supply costs per ton net produced	0.91 F/t 0.18 $/t	1.10 F/t 0.22 $/t	0.38 F/t 0.08 $/t	0.17 F/t 0.03 $/t
SUMMARY	23	TOTAL COSTS/DAY/SUPPORT (6 + 15 + 21)	95.13 F/d/s 19.03 $/d/s	69.17 F/d/s 13.83 $/d/s	79.41 F/d/s 15.88 $/d/s	66.36 F/d/s 13.27 $/d/s
	24	TOTAL COST PER TON PRODUCED (7 + 16 + 22)	6.84 F/t 1.37 $/t	6.38 F/t 1.28 $/t	4.25 F/t 0.85 $/t	4.76 F/t 0.95 $/t

APPENDIX 5 : PHOTOGRAPHS OF FACE OPERATIONS

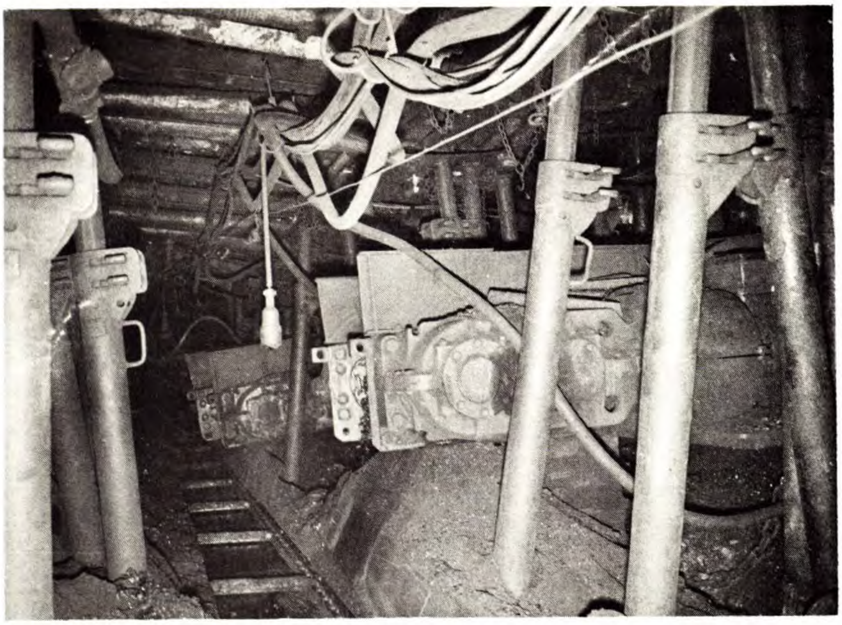


(a) Headgate panel C-D



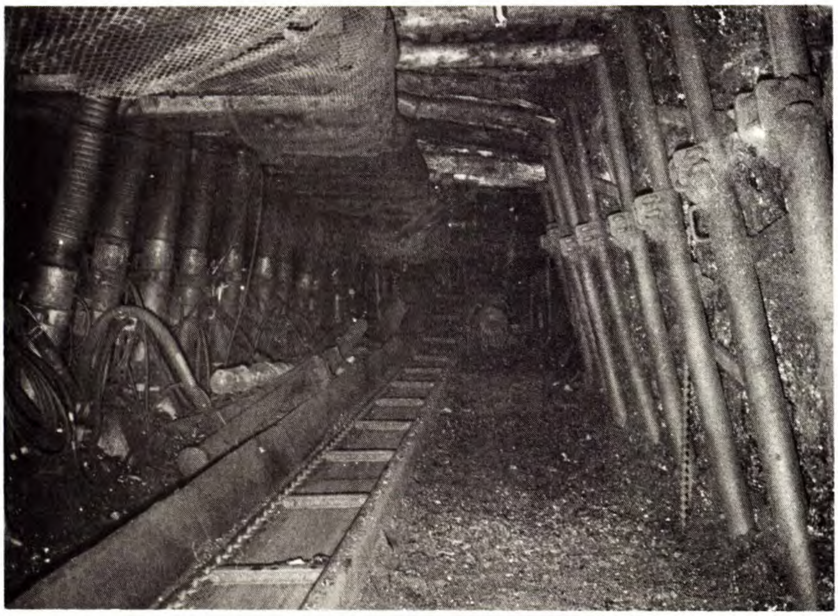
(b) S.M.F. supports - Face C

(c) Westfalia supports - Face D

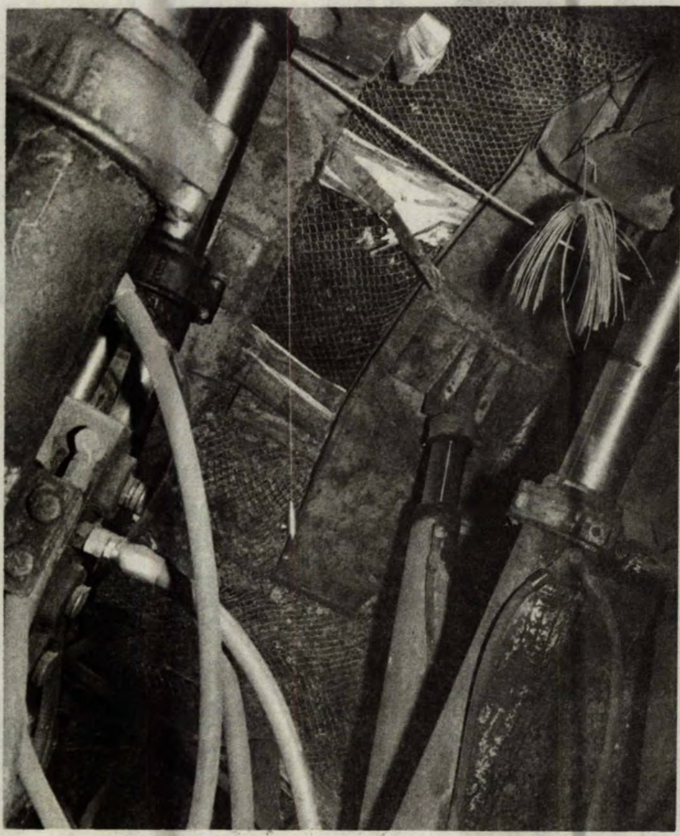


(d) Junction of Tailgate with Face C

(e) Extending the mesh - Face D



(f) Face C, after hand advance, before support advance



(g) "BANANA" props and mesh support

(h) Drawing caved coal - Face D

(i) Drawing caved coal - Face D

APPENDIX 6. FACE, PANEL AND DEVELOPMENT STATISTICS FOR THE DARCY MINE

Table A6.1 gives the face statistics for panels C, D and T for ten months of 1972 together with the average values. The average values for the period May - September are also given to allow comparison with panel statistics over the same period. Table A6.2 gives a breakdown of the average face labour distribution over the same periods.

Figure A6.1 shows that there is a direct linear correlation between the gross tonnage produced from the face and the calculated seam volume (face length x face advance x seam height). This is given by:

$$y = 4.61 + 1.50x$$

where y is the gross tonnage x 10^3 per month; x is the calculated seam volume x 10^3 cu metres/month.

Using this curve the percentage recovery is calculated to be 95.1%.

Table A6.3 shows attempted correlations between the productivity and the face advance rate, the face length and the seam thickness. There is a significant linear correlation between the productivity and the seam thickness (significant at better than a 1% level) (see Figure A6.2). There is no significant correlation with face advance or with face length. The lack of influence of the face advance on the productivity is probably due to the fact that with this hand mining method the face advance remains fairly constant and a significantly higher face advance would probably require significantly more labour to achieve it. Likewise a longer face would require more labour on the face and productivity is not greatly influenced. Certainly it would appear that seam thickness is the prime variable affecting productivity.

Table A6.4 gives the statistics for the panels C, D and T in the Darcy mine, over the 6-month period in which these were the only producing panels; from these statistics mean values for an "average panel" have been derived. Likewise a detailed breakdown of panel labour is also given.

Table A6.5 gives a cost breakdown for panel T over 6 months. From this cost breakdown the cost figures for the "average panel" have been calculated and the proportion of these costs attributable to the face of this average panel have been estimated.

Table A6.6 gives a breakdown of the development carried out in the Darcy mine and mean figures for the development required for an average panel have been derived.

Development

Assume an "average panel" is 400 m long x 86.1 m wide x 10.07 m thick = 3.468×10^5 cu metres.

TABLE A6.1

Face Statistics for 10 Months of 1972 for Faces C, D and T

		ITEM	FACE C										
			J	F	M	A	My	Jn	Jy	A	S	O	MEAN C
DIMENSIONS	1	Mean face length - metres	100	100	98.7	98.3	98.3	98.0	98.0	97.8	97.6	96.8	98.35
	2	Mean face height - metres	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
	3	Mean seam thickness - metres	10.71	10.81	11.44	11.98	11.37	10.72	10.72	10.69	13.60	-	11.34
	4	Mean width of face cut - metres	1.30	1.29	1.23	1.26	1.17	1.30	1.20	1.23	1.23	1.26	1.25
	5	Monthly face advance - metres	16.9	12.3	12.35	12.68	12.9	15.6	2.4	12.4	16.0	17.7	13.1
	6	No. days worked in month	22	22	23	20	23	22	5	21	22	22	20.2
	7	Mean daily face advance - metres	0.77	0.55	0.53	0.63	0.56	0.70	0.48	0.59	0.72	0.80	0.63
PRODUCTION	**8	Percentage recovery %	88	86	85.5	84	92.7	98.8	92.2	104	98.3	90.6	92.0
	9	Tons net/tons gross %	-	-	71.5	74.1	75.1	77.5	75.9	78.3	80.2	76.2	76.2
	10	Monthly production $x 10^3$ tons net	22.6	20.89	18.91	19.75	23.14	24.55	4.28	20.5	25.92	27.54	20.80
	*11	Monthly production $x 10^3$ tons gross	29.69	27.45	26.41	26.65	30.81	31.68	5.64	26.18	32.31	-	27.33
	12	Mean daily production tons net	1027	926	822	987	1006	1116	856	976	1178	1251	1014
	13	Mean daily production tons gross	1349	1217	1148	1331	1339	1440	1127	1246	1468	1641	1332
	14	Coal volume in seam/month $(1 \times 3 \times 5) / \times 10^3$ cu m	21.3	13.29	13.94	14.93	14.42	16.39	2.5	12.96	21.23	-	14.55
LABOUR	***15	Shifts on face per 1000 tons net	80.7	83.7	82.6	70.8	65.2	57.5	78	55.9	57.9	52.2	68.4
	16	Mean man shifts/day on face	82.9	77.5	67.9	69.9	65.6	64.2	66.8	54.5	68.2	65.3	68.3
	17	Productivity tons net/ man shift	12.39	11.94	12.10	14.12	15.33	17.39	12.82	17.88	17.27	19.16	15.04
	18	Productivity tons gross/ man shift	16.28	15.69	16.90	19.05	20.41	22.44	16.89	22.83	21.53	25.14	19.76

Note: 1* Where ratio tons net/tons gross not quoted, average ratio has been used to calculate tons gross.
2** Recovery in panel can exceed 100% due to caving & drawing of coal from above the panel sides.
3 Correlation between seam volume per month (Item 14) and gross tons per month given by:
$y = 4.61 + 1.50x$ ($n = 24$, corr. coeff. 0.925, stand. error 3.41), where $y \times 10^3$ = gross tonnage and x = estimated volume $\times 10^3$ (see Figure A6.1). Slope of graph (Figure) = 1.50 tons/cu metre.
4*** See Table A6.2 for breakdown of labour distribution.

TABLE A6.1 (continued)

	ITEM NO.*	FACE D										
		J	F	M	A	My	Jn	Jy	A	S	O	MEAN D
DIMENSIONS	1	91.5	91.0	88.0	80.9	74.2	65.1	62.7	59.1	51.7	44.1	70.83
	2	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
	3	11.01	10.69	10.39	11.25	11.44	9.52	9.52	9.82	10.36	-	10.44
	4	1.31	1.32	1.34	1.36	1.24	1.40	1.36	1.25	1.31	1.21	1.31
	5	14.5	14.5	15.4	13.3	14.9	16.9	4.1	15.1	17.1	17.0	14.3
	6	22	22	23	20	23	22	5	21	22	22	20.2
	7	0.66	0.65	0.67	0.66	0.65	0.76	0.82	0.71	0.77	0.77	0.71
PRODUCTION	8	88	86	85.5	84	92.7	98.8	92.2	104	98.3	90.6	92.0
	9	-	-	71.6	74.1	75.1	77.5	76.8	78.2	80.2	76.2	76.1
	10	18.10	18.44	18.82	14.91	16.80	14.30	3.45	14.11	14.84	11.97	14.57
	11	23.78	24.23	26.28	20.12	22.37	18.45	4.49	18.04	18.50	-	19.14
	12	825	838	818	745	730	650	689	672	674	544	718
	13	1084	1101	1142	1004	972	838	897	859	840	714	943
	14	14.61	14.10	14.08	12.10	12.67	10.47	2.45	8.76	9.16	-	10.93
LABOUR	15	67.5	76.8	74.6	75.9	72.6	76.1	76.3	92.0	72.8	79.6	76.4
	16	55.7	64.4	61.0	56.5	53.0	49.5	52.6	61.8	49.1	43.3	54.7
	17	14.81	13.02	13.40	13.17	13.77	13.14	13.10	10.87	13.73	12.56	13.16
	18	19.46	17.11	18.71	17.77	18.33	16.95	17.05	17.12	17.12	16.48	17.29

* See first page of Table A6.1 for Item description.

TABLE A6.1 (continued)

	ITEM NO.*	FACE T											OVERALL MEAN	MEAN OVER 5 mo. for which panel statistics available
		J	F	M	A	My	Jn	Jy	A	S	O	MEAN T		
DIMENSIONS	1	-	-	-	72.3	79.5	86.7	90.1	97.1	102.6	103.6	90.27	86.1	84.3
	2	-	-	-	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
	3	-	-	-	6.61	-	11.44	11.44	8.34	4.20	3.57	7.60	10.07	10.22
	4	-	-	-	1.31	1.09	1.13	1.13	1.13	1.14	1.36	1.18	1.25	1.22
	5	-	-	-	13.2	25.2	23.9	5.7	26.0	24.0	28.5	20.9	15.6	15.5
	6	-	-	-	20	23	22	5	21	22	22	20.2	20.2	18.6
	7	-	-	-	0.65	1.09	1.08	1.13	1.24	1.09	1.39	1.09	0.78	0.83
PRODUCTION	8	-	-	-	108	127	93	89.1	92.3	120.8	115	106.5	95.8	99.6
	9	-	-	-	76.8	74.3	76.8	79.2	76.2	75.5	71.2	75.7	76.0	77.1
	10	-	-	-	11.48	22.48	30.41	7.88	30.15	20.41	19.76	20.36	18.38	18.21
	11	-	-	-	14.98	-	39.59	9.45	39.57	27.03	27.75	26.89	24.20	23.15
	12	-	-	-	574	977	1382	1577	1435	927	898	777	930	989.6
	13	-	-	-	747	1315	1799	1991	1883	1227	1388	1026	1223	1279
	14	-	-	-	6.31	-	23.70	5.87	21.05	10.34	10.54	12.97	12.84	12.28
LABOUR	15	-	-	-	98.4	102.3	76.6	111.3	87.4	151.2	142.3	109.9	82.1	82.2
	16	-	-	-	56.5	99.9	106.0	175.5	125.4	140.1	127.8	118.7	76.3	82.1
	17	-	-	-	10.16	9.77	13.03	8.98	11.44	6.61	7.02	9.57	12.92	13.00
	18	-	-	-	13.23	13.15	16.97	11.34	15.01	8.75	9.86	12.64	17.0	16.88

* See first page of Table A6.1 for Item description.

TABLE A6.2

Labour Distribution on Faces C, D and T for 10 Months of 1972 - In Shifts per 1000 tons net[a] (and shifts/day)[b]

		ITEM		FACE C									
			J	F	M	A	My	Jn	Jy	A	S	O	MEAN C
EXCAVATION	HAND WINNING	Headgate, tailgate niche and face advance (a)	33.7	30.9	30.2	26.8	25.7	22.8	26.8	23.7	23.6	23.1	26.7
		(b)	34.6	28.6	24.8	25.9	25.9	25.4	22.9	23.1	27.8	28.9	26.8
	CAVING	Caving & drawing (including tailgate) (a)	11.7	15.9	15.4	13.3	9.1	7.3	7.5	8.3	7.4	6.5	10.2
		(b)	12.0	14.7	12.6	13.1	9.2	8.1	6.4	8.1	8.7	8.1	10.1
	OTHER	Others (a)	0.8	0.8	1.4	0.6	0.6	0.8	3.7	1.5	0.7	0.5	1.1
		(b)	.82	.74	1.15	.59	0.6	.89	3.2	1.5	.8	.6	1.09
OTHER FACE WORK	HAND	Setting props, timbers, etc. (a)	1.7	2.3	2.4	1.9	1.9	1.8	2.3	1.8	1.6	1.1	1.88
		(b)	1.74	2.1	2.0	1.87	1.9	2.0	2.0	1.7	1.8	1.4	1.85
	MECHANISED	Advancing supports and conveyors, maintenance (a)	19.1	20.4	14.3	15.7	16.5	14.4	22.2	11.8	14.7	12.6	16.7
		(b)	19.6	18.9	15.9	15.5	16.6	16.1	19.0	11.5	17.3	15.8	16.6
	SUPPLIES REPAIRS	Transport, repairs timbering (a)	0.4	1.0	0.4	0.8	0.5	0.7	0.9	0.6	0.5	0.7	.65
		(b)	.41	.19	.3	.79	0.5	.78	.77	.58	.59	.87	.65
	OTHERS	Conveyor operators & others (a)	13.3	12.4	13.5	11.8	10.9	9.7	14.5	8.3	9.4	7.6	11.1
		(b)	13.7	11.5	11.1	11.6	11.0	10.8	12.4	8.1	11.1	9.5	11.1
		TOTAL shifts/1000 t net (a)	80.7	83.7	82.6	70.8	65.2	57.5	78	55.9	57.9	52.2	68.4
		TOTAL shifts/day (b)	82.9	77.5	67.9	69.9	65.6	64.2	66.8	54.5	68.2	65.3	68.3

TABLE A6.2 (continued)

		FACE D										
		J	F	M	A	My	Jn	Jy	A	S	O	MEAN D
EXCAVATION	HAND MINING	26.3	29.8	28.9	27.2	26.2	27.4	24.1	27.6	22.0	24.5	26.4
		21.7	24.9	23.9	20.2	19.1	17.8	16.6	18.5	14.8	13.3	19.1
	CAVING	8.8	9.7	9.4	10.1	9.7	10.2	9.3	11.5	11.4	11.3	10.1
		7.3	8.1	7.8	7.5	7.1	6.6	6.4	7.7	7.7	6.1	7.2
	OTHER	2.3	1.6	0.9	0.9	0.8	0.8	4.1	1.7	0.2	0.8	1.4
		1.9	1.3	.75	.7	0.6	0.5	2.8	1.1	0.1	0.4	1.0
OTHER FACE WORK	HAND	1.7	2.3	2.1	2.6	2.6	2.6	2.9	2.4	2.6	2.8	2.5
		1.4	1.9	1.7	1.9	1.9	1.7	2.0	1.6	1.7	1.5	1.7
	MECHANISED	16.6	19.1	19.7	19.6	17.5	20.0	21.2	30.7	17.1	18.7	20.0
		13.7	16.0	14.4	14.6	12.8	13.0	14.6	20.6	11.5	10.2	14.1
	SUPPLIES REPAIRS	0.7	0.9	0.1	0.3	0.5	0.4	0.6	0.9	1.1	1.1	0.7
		0.6	0.75	0.1	.22	.4	0.3	.4	.6	.7	0.6	0.5
	OTHERS	11.1	13.5	13.6	15.2	15.4	14.6	14.2	17.1	18.4	20.4	15.4
		9.1	11.3	11.3	11.3	11.2	9.5	9.8	11.5	12.4	11.1	10.8
		67.5	76.8	74.6	75.9	72.6	76.1	76.3	92.0	72.8	79.6	76.4
		55.7	64.4	61.0	56.5	53.0	49.5	52.6	61.8	49.1	43.3	54.7

(See first page of Table A6.2 for Item description)

TABLE A6.2 (continued)

		FACE T											OVERALL	MEAN OVER 5 mo. for which panel statistics available
		J	F	M	A	My	Jn	Jy	A	S	O	MEAN T	MEAN	
EXCAVATION	HAND MINING	-	-	-	33.0 18.9	34.6 33.8	32.1 44.4	51.9 81.8	36.7 52.7	70.9 65.7	74.1 66.5	47.6 51.9	32.0 30.5	31.7 31.4
	CAVING	-	-	-	16.0 9.2	20.0 19.5	13.9 19.3	13.4 21.1	12.1 17.4	11.6 10.7	9.3 8.3	13.7 15.1	11.1 10.3	10.8 10.7
	OTHER	-	-	-	10.1 5.8	5.1 5.0	4.9 6.8	1.1 1.7	2.0 2.9	4.2 3.9	3.1 2.8	4.3 4.1	2.1 1.8	2.1 2.1
OTHER FACE WORK	HAND	-	-	-	1.9 1.1	1.9 1.9	1.8 2.5	5.7 8.9	4.6 6.6	3.9 3.6	3.6 3.2	3.3 4.0	2.5 2.3	2.7 2.7
	MECHANISED	-	-	-	21.2 12.2	23.0 22.4	16.5 22.8	23.1 36.4	19.0 27.3	36.3 33.6	34.6 31.1	24.3 26.5	20.0 18.2	20.3 20.1
	SUPPLIES REPAIRS	-	-	-	- -	- -	- -	- -	- -	1.6 1.5	0.7 .6	0.3 0.3	0.6 0.5	0.6 0.6
	OTHERS	-	-	-	16.2 9.3	17.6 17.2	10.2 14.1	16.1 25.4	13.2 18.9	22.7 21.0	16.9 15.2	16.1 17.3	14.0 12.6	14.1 14.0
		-	-	-	98.4	102.3	76.6	111.3	87.4	151.2	142.3	109.9	82.1	82.2
		-	-	-	56.5	99.9	106.0	175.5	125.4	140.1	127.8	118.7	76.3	81.3

(See first page of Table A6.2 for Item description)

FIGURE A61 : CORRELATION BETWEEN TONS GROSS PRODUCED/MONTH AND CALCULATED SEAM VOLUME (FACES C,D AND T)

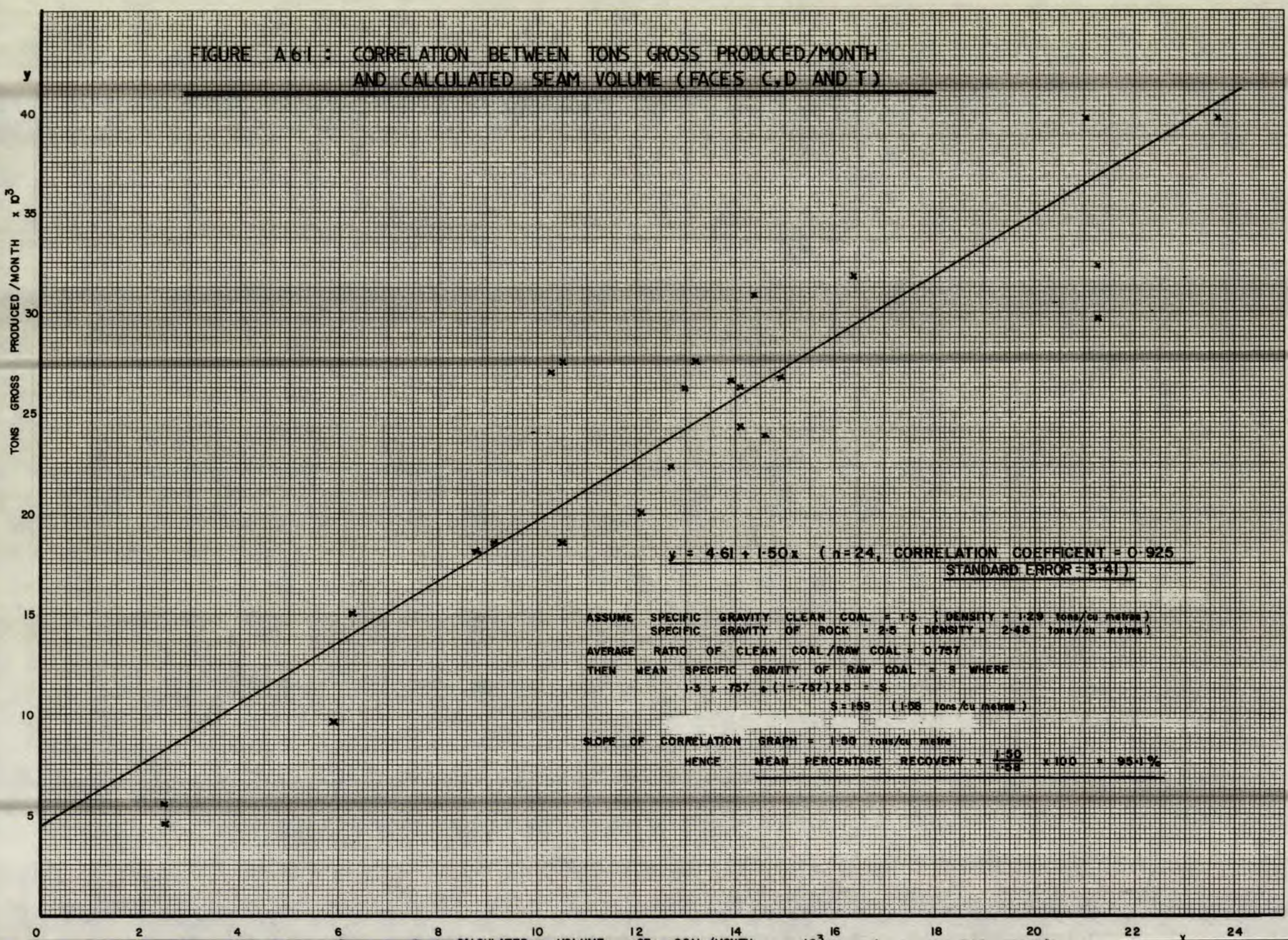


TABLE A6.3

Productivity per Unit Face Advance, per Unit Face Length, per Unit Thickness

P/ms	t metres	length l metres	advance a metres	$\frac{P}{al} = Y_1$	$\frac{P}{at} = Y_2$	$\frac{P}{lt} = Y_3$
16.28	10.71	100	.77	.211	1.974	.0152
15.69	10.81	100	.55	.285	2.638	.0145
16.90	11.44	98.7	.53	.322	2.787	.0149
19.05	11.98	98.3	.63	.308	2.524	.0161
20.41	11.37	98.3	.56	.371	3.205	.0182
22.44	10.72	98	.70	.327	2.990	.0213
16.89	10.72	98	.48	.359	3.264	.0160
22.83	10.69	97.8	.59	.396	3.619	.0218
21.53	13.60	97.6	.72	.306	2.20	.0162
19.46	11.01	91.5	.66	.322	2.678	.0193
17.11	10.69	91.0	.65	.289	2.462	.0175
18.71	10.39	88	.67	.317	2.688	.0204
17.77	11.25	80.9	.66	.332	2.393	.0195
18.32	11.44	74.2	.65	.380	2.465	.0215
16.95	9.52	65.1	.76	.342	2.343	.0273
17.05	9.52	62.7	.82	.332	2.184	.0285
18.90	9.82	59.1	.71	.331	1.994	.0239
17.12	10.36	57.7	.77	.385	2.146	.0286
13.23	6.61	72.3	.65	.281	3.079	.0276
16.97	11.44	86.7	1.08	.181	1.373	.0171
11.34	11.44	90.1	1.13	.111	0.877	.0110
15.01	8.34	97.1	1.24	.124	1.451	.0185
8.75	4.20	102.6	1.09	.078	1.911	.0203
9.86	3.57	103.6	1.39	.068	1.987	.0266

P = Productivity tons gross/man shift
t = seam thickness, metres
l = face length, metres
a = face advance, metres

1. $Y_1 = .0144 + .0265t$ (r = 0.623, n = 24, standard error .075)
   r significant to better than 1% level
   i.e. Unit Productivity is a linear function of seam thickness.

2. $Y_2 = 1.893 + .0056\ l$ (r = 0.13, n = 24, standard error = .615)
   No significance (worse than 50% level)

3. $Y_3 = .0183 + .0023\ a$ (r = 0.114, n = 24, standard error .0047)
   No significance
   i.e. Productivity is a linear function of seam thickness; no correlation to face length or advance.

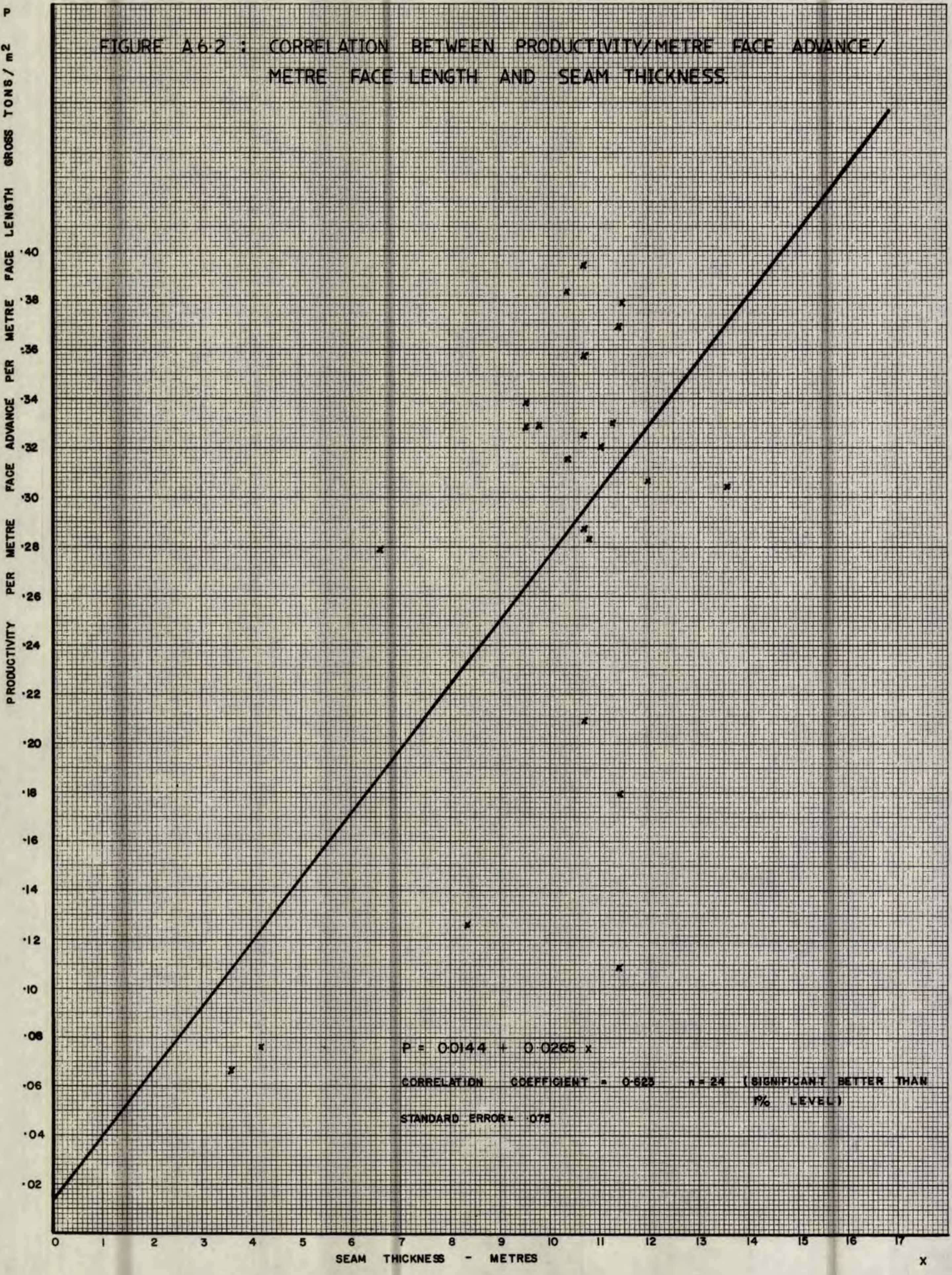


FIGURE A6·2 : CORRELATION BETWEEN PRODUCTIVITY/METRE FACE ADVANCE/ METRE FACE LENGTH AND SEAM THICKNESS.

TABLE A6.4

Panel Statistics - Totals and Averages from Panels C, D and T - Darcy Mine

	ITEM	MAY	JUNE	JULY	AUG	SEPT	MEAN	MEAN FOR AVERAGE PANEL
1	No. days worked	23	22	5	21	22	18.6	18.6
2	Total production from 3 panels tons net	61,170	64,760	15,610	69,260	62,420	54,644	18,214 t per panel
3	Mean daily production from all panels - tons net	2,659	2,943	3,122	3,298	2,837	2,972	990.6 t/panel
4	Tons net/tons gross %	74.8	77.2	78.0	77.3	78.5	77.2	77.2 %
5	Total production from 3 panels - tons gross	81,788	83,886	20,870	89,560	79,515	71,121	23,707 t/panel
6	Mean daily production from all panels - tons gross	3,555	3,813	4,173	4,266	3,614	3,884	1,294 t/panel
7	Shifts per 1000 tons net							
	(a) on face	83.1	71.5	95.1	75.2	90.3	83.0	83.0
	(b) development	69.8	53.4	65.9	45.7	59.4	58.8	58.8
	(c) services	120.9	97.5	147.3	97.7	113.4	115.4	115.4
	Total shifts/1000 tons net on panel	273.8	222.4	308.3	218.6	263.2	257.2	257.2
8	Shifts per day							
	(a) on face	73.6	70.1	98.9	82.7	85.4	82.2	82.2
	(b) development	61.9	52.4	68.6	50.2	56.2	57.9	57.9
	(c) panel services	107.1	95.6	153.3	107.4	107.3	114.1	114.1
	Total shifts/day on panel	242.6	218.1	327.2	240.3	248.8	254.2	254.2
9	Productivity tons net/ man shift	3.65	4.50	3.18	4.57	3.80	3.89	3.89
10	Productivity tons gross/ man shift	4.88	5.82	4.10	5.92	4.84	5.04	5.04

TABLE A6.4 (continued) Detailed Labour Breakdown

ITEM	May	June	July	Aug	Sept	Mean	"Average panel"
Shifts per 1000 tons							
(a) on face	83.1	71.5	95.1	75.2	90.3	83.0	83.0
(b) development	69.8	53.4	65.9	45.7	59.4	58.8	58.8
(c) services							
(i) installation & dismantling	17.2	12.3	10.8	9.4	9.4	11.8	11.8
(ii) transport	19.0	17.7	23.6	20.5	23.8	20.9	20.9
(iii) maintenance work	32.7	29.1	37.1	27.0	30.9	31.4	31.4
(iv) supplies	30.4	25.6	45.8	27.0	32.3	32.2	32.2
(v) safety and other	21.6	12.8	28.9	13.8	17.0	18.8	18.8
Total services	120.9	97.5	147.2	97.7	113.4	115.4	115.4
Total shifts/1000 t net on panel	273.8	222.4	308.3	218.3	263.2	257.2	257.2

TABLE A6.5

Cost Analysis - Panel T - Estimated Face T

ITEM	PANEL: JULY	AUG	SEPT	MEAN YEAR TO DATE	AVERAGE PANEL	ATTRIBUTED TO AVERAGE FACE	AVE FACE
No. days worked in month	5	21	22	19	18.6		
Average no. shifts/day - panel	474	335	377	241.9	254.2		
Production t/net	7,880	30,150	20,410	188,380	18,214		
Daily product/t net	1,577	1,435	927	930	990		
% t n/t g	79.2	76.2	75.5	75.7	77.2		
Productivity t net/m.s	3.53	4.52	2.73	3.42	3.89		
Face length	90.1	97.8	102.6	90.27	84.3 m		
LABOUR COSTS Average panel prorated by $\frac{F}{t}\left(\frac{t}{\text{shifts}}\right)_T\left(\frac{\text{shifts}}{t}\right)_{\text{average}} = 0.99$ F/t							
7 Salaries - underground workers F/ton	23.72	16.94	27.0	21.60	21.4	Prorated by ratio of	6.84
8 Additional emoluments F/ton	5.23	1.55	2.49	2.49	2.47	face to panel labour	0.79
9 Bonus on results F/ton	5.41	3.45	5.51	4.50	4.45	$= \frac{82.1}{257.2} = 0.32$	1.42
10 Fringe benefits F/ton	27.15	17.35	27.66	22.31	22.0		7.04
11 Injuries, absenteeism F/ton	2.08	0.43	0.82	0.73	0.72		0.23
Total Labour F/ton	63.6	39.74	63.5	51.65	51.0		16.32
SUPPLIES F/ton Average panel - assume same as T				(174 F/m.s)			
Timber F/ton	1.09	1.05	2.33	1.53	1.53	70% timber to face	1.09
Supports - metal arches/friction props	-	0.62	0.62	1.03	1.03	assume 0 on face	-
Walking props	0.75	0.84	0.30	0.99	0.99	100% of face	0.99
Explosives	1.50	0.79	0.74	1.04	1.04	30% to face	0.31
Dismantling & loading	0.17	0.01	-	0.04	0.04	100% to face	0.04
Conveyors, etc.	8.93	0.41	5.15	1.75	1.75	50% to face	0.88
Monorail, etc.	-	0.03	0.36	0.18	0.18	zero to face	-
Electrical supplies	0.31	0.11	0.17	0.18	0.18	50% to face	0.09
Others	1.65	2.72	2.80	2.36	2.36	50% to face	1.18
Total Supplies	14.43	6.60	12.51	9.16	9.16		4.58

TABLE A6.5 (continued)

ITEM	JULY	AUG	SEPT	MEAN YEAR TO DATE	AVERAGE PANEL		AVE FACE
RENTALS							
Walking props F/ton	4.02	2.58	3.85	5.65	5.65	100% to face	5.65
Dismantling & loading	0.07	0.06	0.27	0.13	0.13	100% to face	0.13
Conveyors	1.85	1.77	2.61	2.42	2.42	50% to face	1.12
Monorail	0.05	0.05	0.07	0.08	0.08	zero to face	-
Special electric machines	0.72	0.65	1.01	1.18	1.18	50% to face	0.59
Others	0.19	0.22	0.44	0.36	0.36	50% to face	0.18
Total Rentals	6.93	5.37	8.28	9.84	9.84		7.67
MAINTENANCE							
Machine shop							
Walking supports	0.02	0.19	0.35	0.10	0.10	100% to face	0.10
Conveyors	0.13	0.06	0.03	0.08	0.08	50% to face	0.08
Other	0.09	0.04	0.04	0.15	0.15	50% to face	0.08
Contracted maintenance	0.09	-	-	0.02	0.02	50% to face	0.01
Total Maintenance	0.34	0.30	0.43	0.37	0.37		0.27
TOTAL COST/TON	85.33	52.02	84.74	71.04	70.37		28.84 F/ton net

TABLE A6.6

Development Statistics for the Darcy Mine

	ITEM	J	F	M	A	My	Jn	Jy	A	S	MEAN	"AVERAGE PANEL"
1	Metres/1000 tons net panel preparation	2.97	2.49	2.38	2.39	3.56	2.80	1.55	1.43	1.05	2.29	2.29
2	Total production t/net x 10^3	75.6	73.88	70.12	65.35	68.24	75.05	16.58	68.70	65.80	64.36	18.2
3	Descents/100 tons	36.6	41.2	48.5	42.7	69.8	53.4	65.9	45.7	50.5	49.6	58.8
4	No. working days	22	22	23	20	23	22	5	21	22	20.6	18.6
5	Total metres/ month	224.5	183.9	166.9	156.2	242.9	210.1	25.7	98.2	69.0	153.0	41.7
6	Metres advance/ working day	10.20	8.36	7.26	7.81	10.56	9.55	5.14	4.76	3.14	7.43	2.24
7	Total shifts/ month x 10^3	2.766	3.043	3.400	2.790	4.763	4.007	1.092	3.139	3.322	3.192	1.070
8	Cm/man shift	8.12	5.36	4.91	5.59	5.09	5.24	2.35	3.13	2.08	4.79	3.89

Assume "raw coal" density X 1.58 tons/cu metre.

Tonnage raw coal in panel = 5.479×10^5 tons. At tons net/ton gross = 0.76 gives tons net in panel = 4.164×10^5 tons.

Total development required for panel = 2.29 metres/1000 tons net = 2.29 x 416.4 = 953.6 metres.

If the average face length is 84 metres, this leaves 870 metres for gate road and other access roads. This is roughly split into headgate and tailgate each of about 350 m with an additional 170 metres of development in inclines up to the seam.

Approximate cost/metre

No. metres/month = 41.7
No. shifts/month = 1070

Labour costs at 170/shift = 1070 x 170 = F 181,900
Labour cost/metre = F 4,362 = \$ 872

Assume labour is 60% of costs
Total cost/metre = F 7,270/m = \$1474/m

This figure seems to be very high; too much reliance should not be placed on it.

APPENDIX 7. THE ALPINE CONTINUOUS MINER

The machine is manufactured by "La Société Autrichienne Alpine-Montangesellschaft".

1. The Chassis. The chassis is formed by two braced box girders, each girder supports a caterpillar track (fitted with cleats to assist the grip) and a 6-kW motor using a worm screw reduction gear, which drives the caterpillar track.

2. The cutting Arm and the Turret. The turret is bolted to the chassis. It carries three identical rams which move the head horizontally (1 ram, item 2, Figure A7.1) and vertically (2 rams, item 3). These double action rams operate a toothed rack gear which acts on a circular segment gear. The cutting head comprises a 30-kW motor (item 4), a reduction gear train (item 5) and the cutting head made from 2 half drums each with 16 picks. The cutting head describes an arc of 600 mm diameter with a linear speed of 4.7 m/s.

3. Loading mechanism. The loading mechanism consists of:

- A table, 2 m wide, with two small double action rams (item 7) which adjust its inclination.
- A system of "lobster claws" driven by reduction gears bolted to the table (item 6); shearing pins between the reduction gear and the claws ensure safety.
- A Galle double chain conveyor, driven by two 6-kW motors (item 13).

4. Electrical equipment. A 500 V supply is required for the motors and 42 volts for the auxiliary circuits. It is flameproofed to normal German regulatory standards; permission for use in France has been given. The control panel (item 8) is in the right of the machine and the junction boxes (item 9) are on the left hand side.

5. Hydraulic equipment. The rams are supplied with oil at 75 bars. The 6-kW motor (item 11) drives a pump of capacity 20 litres/minute drawing hydraulic fluid from an 80 litre reservoir (item 10). The controls are operated by 2 distributors - one control column distributor (item 12) for rotation of the turret and one distributor for the loading table ram.

An extendable module bridge conveyor is used to transport coal from the Alpine machine to the fixed haulageway conveyor.

Figures A7.2 and Figures 4(a) and (b) show photographs of the machine.

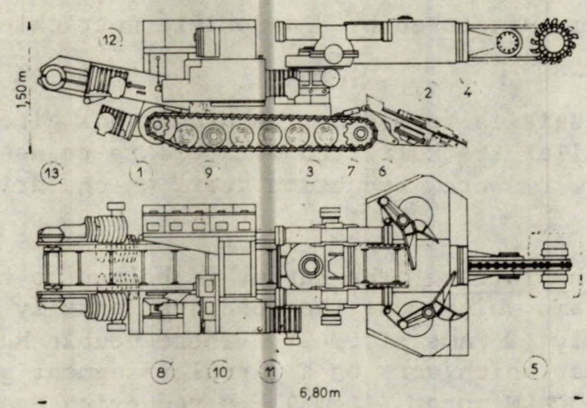


FIGURE A 7·1

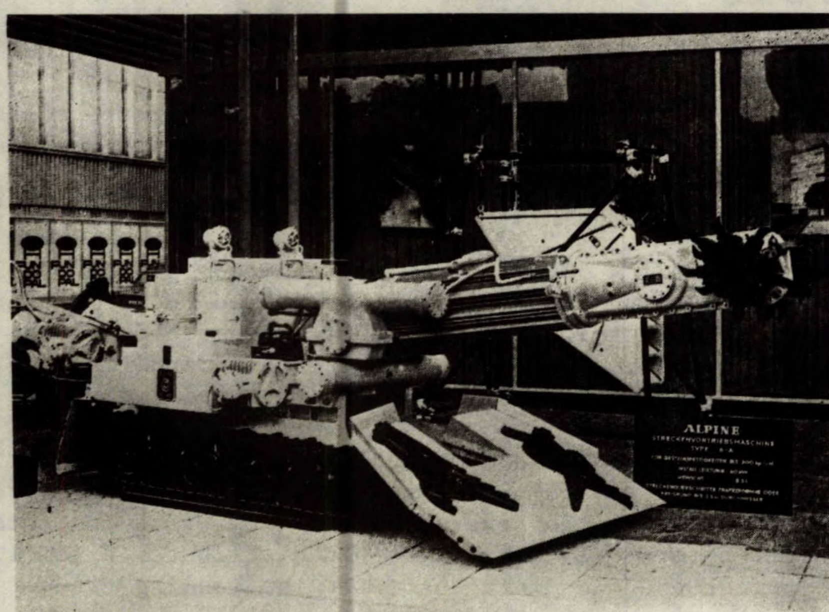


FIGURE A 7·2

FIGURES A 7·1 AND A 7·2 : THE ALPINE CONTINUOUS MINER

APPENDIX 8. DISMANTLING AND INSTALLATION OF THE FACE

Dismantling of one face and transfer of the equipment to a new face requires careful planning if continuity of production is to be maintained as far as possible. This was done in Blanzy using P.E.R.T. techniques (Program evaluation and review techniques). Table A8.1 and the ensuing charts show the programs planned for the dismantling of face S12b in the Rozelay mine and the transfer and installation of this equipment on face S3b. This program was planned to start on 17/1/72, however conditions on face S12b were so bad that the shearer could not be used. Consequently the transfer of the machine was started early (10/1/72) and mining of face S12b was completed by hand advance methods. The charts show the actual realization of this program in comparison with the plans.

Originally it was planned that the change-over would require 1132 man shifts; in practice it required 1571 man shifts. The charts show the distribution of this labour with time and by job. The major problem encountered during the change-over was in the dismantling and removal of the powered supports on face S12b. This was due to a lack of headroom which retarded the whole operation.

Placing of the powered supports on the face has been tried by two methods:-

(a) The supports are assembled completely on a specially built steel working platform in the headgate. This steel platform is a large "sledge" which may be pulled down the face by winches. The first props are installed at the tailgate end, working back towards the headgate. The roof over the support being installed is held in position by another powered support, the canopies of which are parallel to the face; this support is retreated back towards the headgate as each support is installed.

(b) Alternatively, props are installed at the headgate end working towards the tailgate. In this case the canopies of the supports already installed are used to support the monorail by means of which the next support is transferred into position along the face. In this case, however, the canopies of the support being installed must be transferred separately along the face and placed in position after the rest of the support is in place.

Currently method (a) is preferred since it allows complete assembly of the support in the headgate; placing of the canopies in position on the face was found to be very awkward in the second method.

Photographs, Figures A8.1 and A8.2 show a sequence of the operations during the dismantling and assembly of powered supports from one face to another.

TABLE A8.1

Schedule for Dismantling Face S12b and Installation of Face S3b

ITEM No.	ITEM	Total Man shifts PLANNED	ACTUAL
1	Dismantling DTS drum shearer	12	12
2	Installation of winch and pulley in face S3b	4	4
3	Installation of hydraulic feed line, face S3b, from headgate to tailgate	4	4
4	Monorail operators	114	161
5	Transport shearer from S12b to tailgate S3b	16	22
6	Transport electrical boxes and transformer for shearer	4	2
7	Transport of face conveyor loading plates to headgate S3b face	8	12
8	Transport conveyor drive unit from headgate S12b to headgate S3b	4	4
9	Transport conveyor drive unit from tailgate S12b to tailgate S3b	4	2
10	Transport transformer and electrical boxes of face conveyor to tailgate S3b	4	-
11	Transport pumps and their electrical equipment to tailgate S3b	4	4
12	Transport Bretby chain, cables and feeders	4	4
13	Transport powered supports to headgate S3b (72) and to surface (10)	112	80
14	Dismantle cables, chains and water pipes	15	8
15	Dismantle conveyor loading plates	18	19
16	Removal of loading plates, cables, feeders, etc. from tailgate S12b	13	20
17	?	2	2
18	Transport loading plates to S3b face	18	27
19	Mount loading plates on S3b	52	41
20	Recovery of face conveyor chain	8	3
21	Dismantle drive unit of face conveyor, headgate S12b + 6 elements of face conveyor	2	4
22	Dismantle conveyor drive unit, tailend S12b	4	10
23	Installation of winch ahead of face 12b	2	2
24	Dismantle powered support No. 80, remove from tailend S12b	3	12
25	Rotation of supports Nos. 81 and 82	4	5
26	Assembly of conveyor drive units in tailgate and S3b	6	9
27	Assembly of conveyor drive unit in headgate S3b and upper chain	10	13

TABLE A8.1 (continued)

ITEM NO.	ITEM	Total Man shifts PLANNED	ACTUAL
28	Dismantle 3 supports, remove from tailgate and S12b	11	20
29	Dismantle remaining powered supports and remove from tail S12b	286	347
30	Assemble 6 supports, transport on face, apply load; elements of face conveyor and loading plates, drawing conveyor, wire mesh, dismantling of sledge	24	54
31	Put 66 chocks under pressure on face S3b	264	396
32	Cut niche for drum shearer	6	15
33	Assemble shearer and haulage chain	20	28
34	Assemble drive unit of rear conveyor and upper chain	9	12
35	Dismantle pumps in tailend S12b	3	2
36	Assemble pumps and electrical drives in tailend S3b	6	5
37	Assemble Bretby chain, cables, water feed on face S3b	8	7
38	?	8	8
39	Remove winches from face S3b	4	4
40	Dismantle and remove working platform	2	4
41	Lengthen face conveyor	12	8
42	Tests	2	-
43	Installation of cables, water feed and Bretby chain	16	13
44	Transport elements of rear conveyor to S3b and assemble		33
45	Dismantle CBS 350 conveyor of face 12b		7
46	Remove monorail, working platform, sledge		25
47	Timbering of face 12b		4
48	Miscellaneous items on face S3b		46
49	Assemble face conveyor on S3b		47
50	Dismantling of powered supports		
51	Assembly of powered supports		
52	Planned labour - man shifts		
53	Actual labour - man shifts		
		1132	1571

Legend on chart 3 Planned (with No. man shifts)

6 Actual (with No. man shifts)

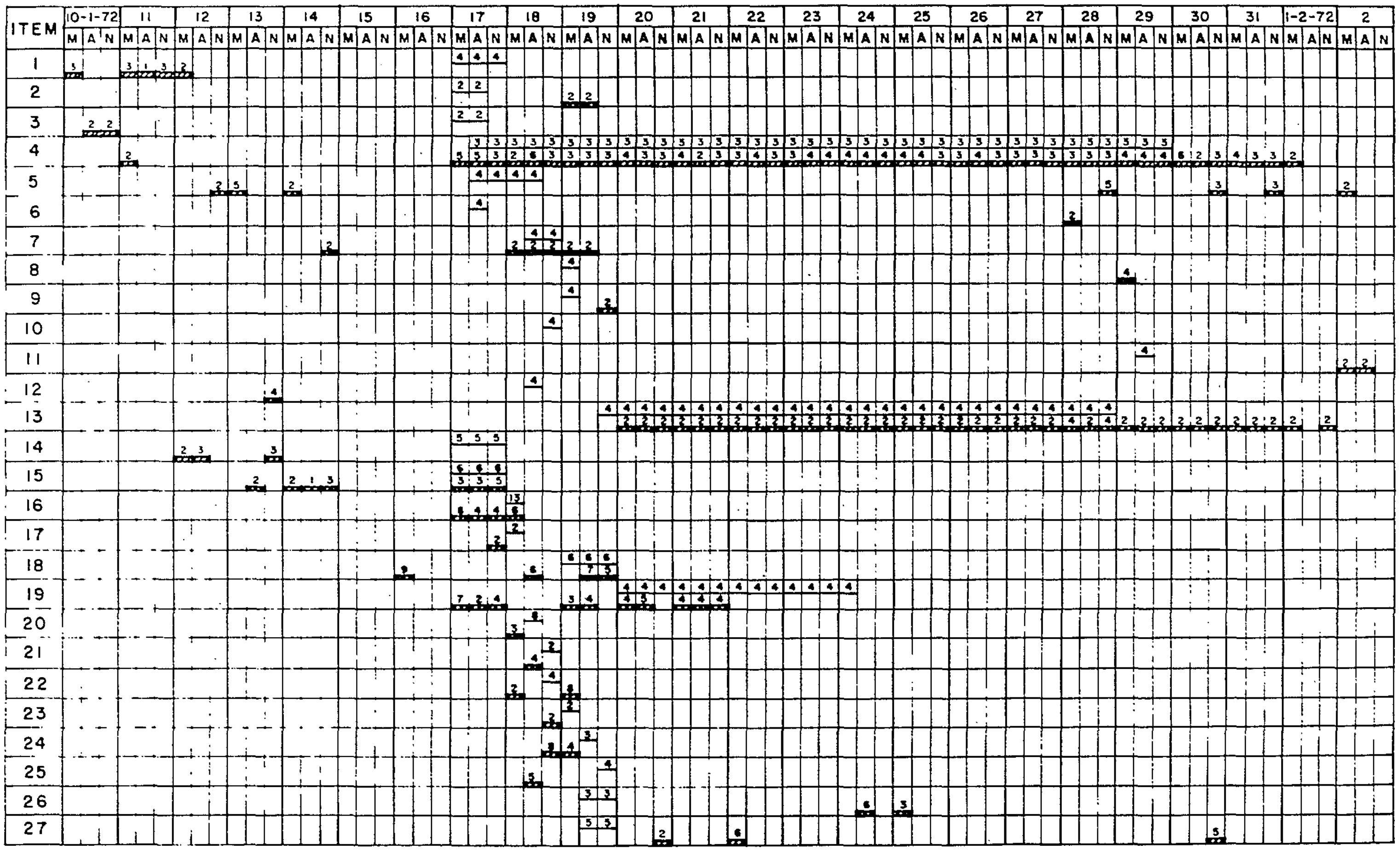
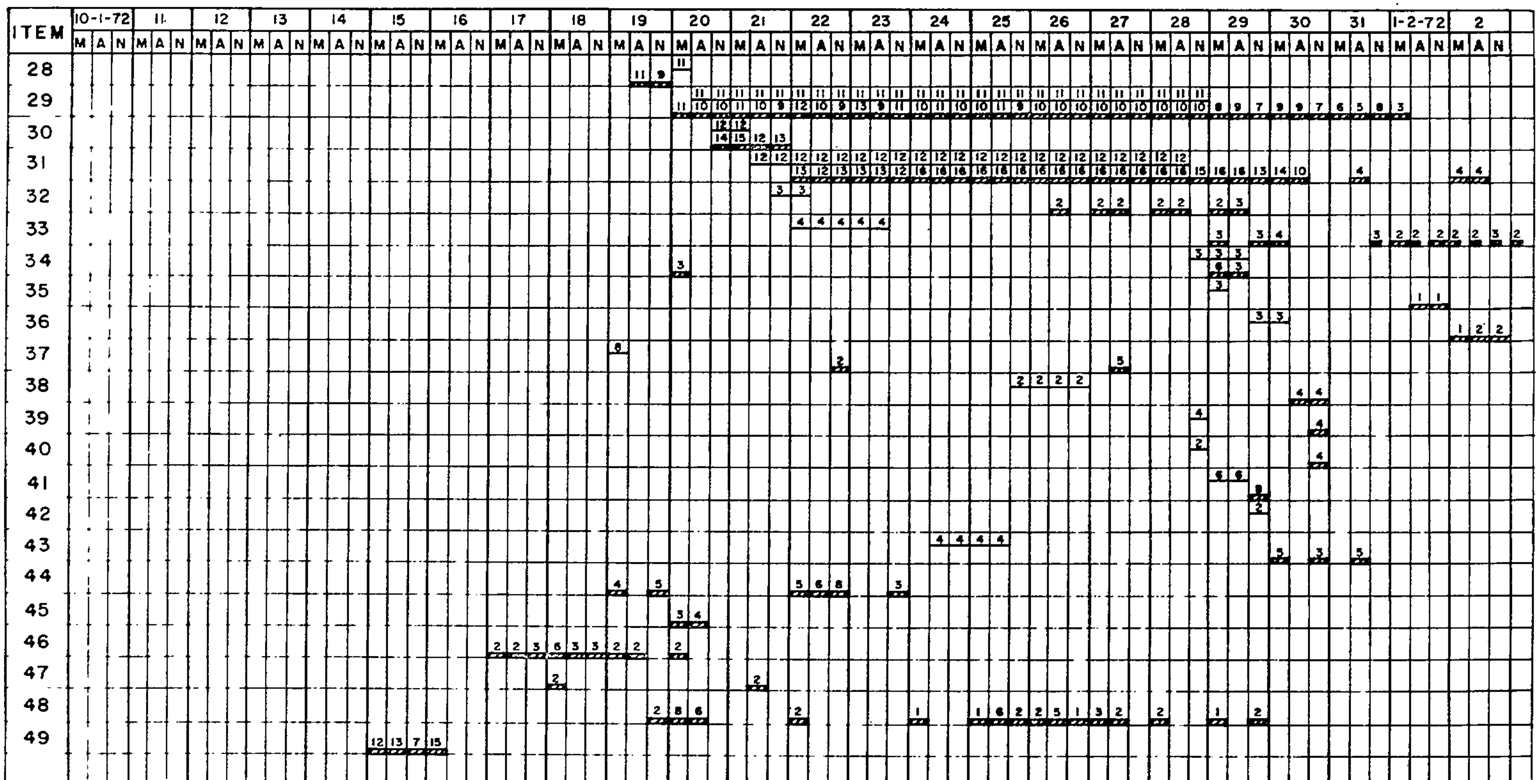


TABLE A8.1 Schedule for dismantling face S12b and installation of face S3b.



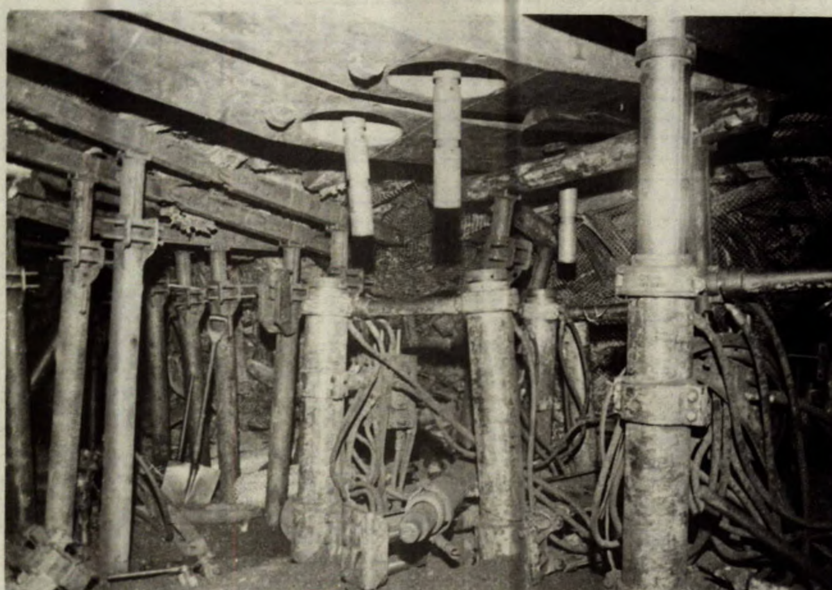
ITEM	18			19			20			21			22			23			24			25			26			27			28			29			30			31			1			2		
	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N	M	A	N
50	-	-	-	1	1	1	1	1	1	2	2	1	2	2	1	2	2	2	3	2	3	2	3	2	3	2	1	4	4	1	3	3	2	3	2	1	3	1	2	2	3	3	2	-	-	-	-	-
51	-	-	-	-	-	-	-	-	1	1	1	2	2	2	2	1	2	2	2	3	2	3	3	3	3	3	2	3	4	4	2	2	3	3	3	1	2	3	-	-	-	-	-	-	-	-	-	-

ITEM	10-1-72	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	1-2-72	2	
52	0	0	0	0	0	0	0	71	62	72	78	105	117	110	102	100	96	90	87	39	3	0	0	0	0
53	7	9	9	14	10	32	24	60	67	85	98	99	114	93	104	107	104	107	109	122'	102	50	17	26	2

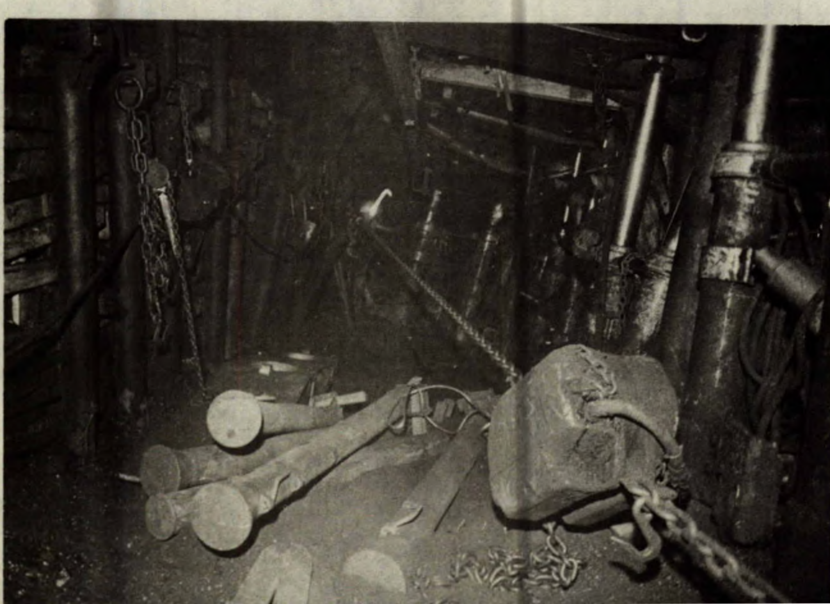
TABLE A8.1 (continued)

FIGURE A8·1 : DISMANTLING OF POWERED SUPPORTS ON FACE

(a) BEFORE DISMANTLING THE SUPPORTS TWO FRICTION PROPS ARE PLACED AT THE FRONT AND AT THE REAR TO HOLD THE CANOPIES IN PLACE.



(b) THE SUPPORTS ARE THEN LOWERED FROM THE CANOPIES.

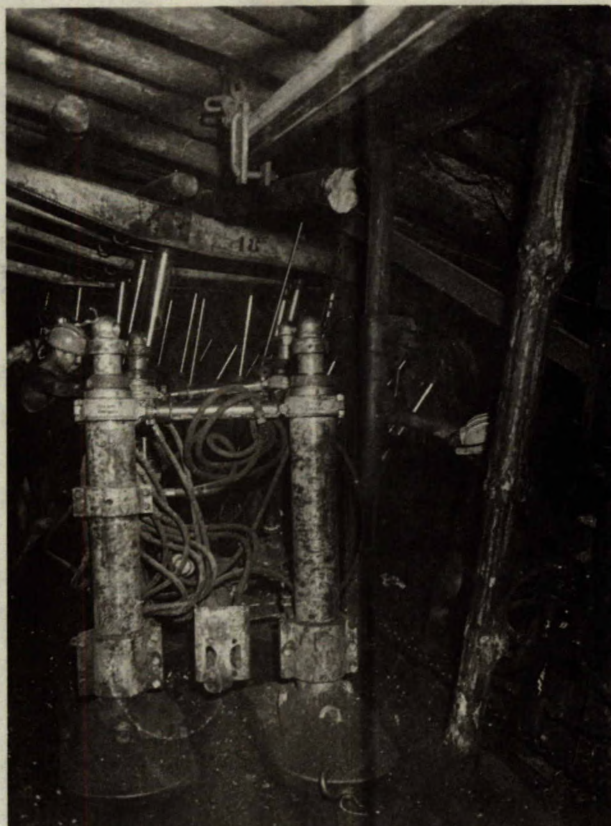


(c) THE SUPPORT IS THEN MANHANDLED OUT OF THE FACE WITH THE AID OF A WINCH.

(d) THE FRICTION PROPS ARE PULLED, DROPPING THE CANOPIES TO THE FLOOR.

(e) AFTER REMOVAL OF THE POWERED SUPPORTS, THE FACE IS SUPPORTED BY THREE ROWS OF FRICTION PROPS.

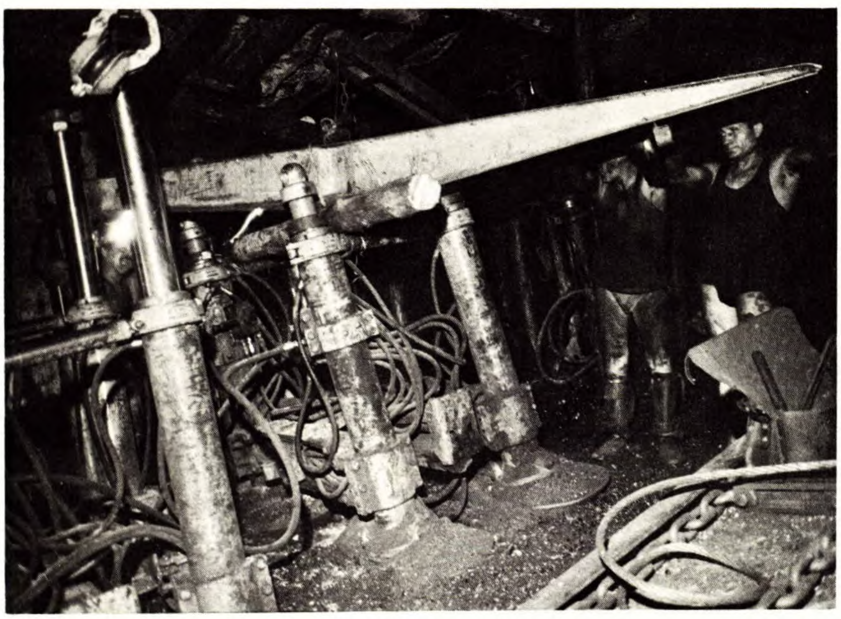
FIGURE A 8·2
ASSEMBLY OF THE POWERED SUPPORTS ON THE FACE



(a) THE BODY OF THE SUPPORT IS TRANSPORTED ON TO THE FACE BY MONORAIL

(b) THE SUPPORT IS MANHANDLE INTO POSITION.

(c) THE FIRST CANOPY IS PLACED



(d) THE FIRST CANOPY IS PLACED

(e) THE SECOND CANOPY IS PLACED

(f) TIMBER SUPPORTS ON THE FACE PRIOR TO THE INSTALLATION OF THE POWERED SUPPORTS; NOTE THE MONORAIL.

(g) ASSEMBLY OF THE BANANA PROPS.

APPENDIX 9. FACE, PANEL AND DEVELOPMENT STATISTICS FOR ROZELAY PANEL S3b

Table A9.1 gives the face statistics for Rozelay face S3b for a period of 6 months together with the average values. Figure A9.1 shows that there is a linear correlation between the gross tonnage produced from the face and the calculated seam volume (including the sandstone bed in the seam); this correlation is given by:

$$y = -1.93 + 1.06x$$

where y is the gross tonnage x 10^3 per month and x is the calculated seam volume x 10^3 cu metres/month.

Using this curve the percentage recovery is calculated to be 67.1%. This recovery rate is considerably less than that for the Darcy mine (95.1%); the difference is attributed to the presence of the sandstone bed in the middle of the seam. This sandstone bed generally caves in fairly large blocks which presumably cannot all be drawn through the mesh at the back of the face and these undrawn blocks will also trap coal above and behind them; thus the recovery is reduced.

Table A9.2 shows attempted correlations between the productivity and the face advance, the face length and the seam thickness. There is a very good correlation, see Figure A9.2, between the productivity and the face advance (unlike the Darcy mine case). Whereas in the Darcy mine an increase in face advance rate is difficult to achieve and would require more men on the face to achieve it, on the Rozelay face the advance is dependent on the number of passes the machine makes along the face and thus relatively few more men are required to increase the number of passes.

The data from this face is all taken from basically the same face length (~ 110 metres) thus it is not possible to establish whether or not there is a correlation between productivity and face length. Intuitively a longer face length should reduce the relative machine turn round to machine cutting time and thus, with the same labour, some increase in productivity might be expected. However it is not possible to establish this from the data available and thus it is not possible to determine whether this intuitively expected influence is significant or is a minor factor.

Neither was it possible, from the data available, to establish a correlation between productivity and seam thickness. In the available data there is very little variation in seam thickness, this probably accounts for why such a correlation is not shown here. Certainly a very good correlation between productivity and seam thickness was established for the Darcy mine data and there would appear to be no basic differences between the two mines that would counteract this correlation. It is therefore assumed that this correlation still probably exists for the Rozelay mine, but the lack of variation in data makes it difficult to establish from the available data.

Table A9.3 gives panel statistics for the No. 3 panel continuing face S3b.

TABLE A9.1

Face Statistics for Face S3b (Rozelay)

	ITEM	MAR	AP	MAY	JN	JLY	AUG	SEPT	OCT	MEAN
1	Length of face cut/day by shearer - metres	177	234	228	253	261	167	191	164	209
2	No. of passes per day of machine	1.61	2.15	2.09	2.37	2.40	1.50	1.74	1.50	1.92
3	Mean face height - metres	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8
4	Mean seam thickness - metres	12.9	12.7	11.9	12.15	12.15	12.15	12.15	11.9	12.25
5	Mean width of face cut/pass - metres	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55	0.55
6	Monthly face advance - metres	20.4	23.6	26.4	28.7	6.6	17.3	21.1	18.1	20.3
7	No. days worked in month	23	20	23	22	5	21	22	22	19.8
8	Mean daily face advance - metres	0.86	1.18	1.15	1.30	1.32	0.83	0.96	0.83	1.05
9	Tons net/tons gross	72.4	74.5	74.8	76.4	75.9	80.5	75.8	75.8	75.7
10	Monthly face production tons net x 10^3	22.44	25.11	27.44	26.19	5.74	18.38	20.79	15.39	20.18
11	Monthly production tons gross x 10^3	30.99	33.70	36.68	34.28	7.56	22.83	27.42	20.30	26.72
12	Mean daily production - tons net	975	1256	1195	1190	1148	875	943	700	1035
13	Mean daily production - tons gross	1347	1686	1596	1558	1512	1081	1244	923	1368
14	Calculated seam volume (including rock band) (4 x 1 x 5 x 7) x 10^3 cu metres	28.8	32.8	34.4	37.1	8.72	23.4	28.1	23.5	27.10
15	Shifts on face per 1000 tons net	100.5	64.5	67.5	70.2	66.6	89.2	87.8	108.5	81.8
16	Mean man shifts/day on face	78.0	81.0	80.7	83.5	76.5	78.1	82.8	76.0	79.6
17	Productivity - tons net/man shift	12.5	15.5	14.8	14.25	15.0	11.2	11.38	9.21	13.0
18	Productivity - tons gross/ man shift	13.7	20.8	19.8	18.7	19.8	13.84	15.02	12.10	16.17

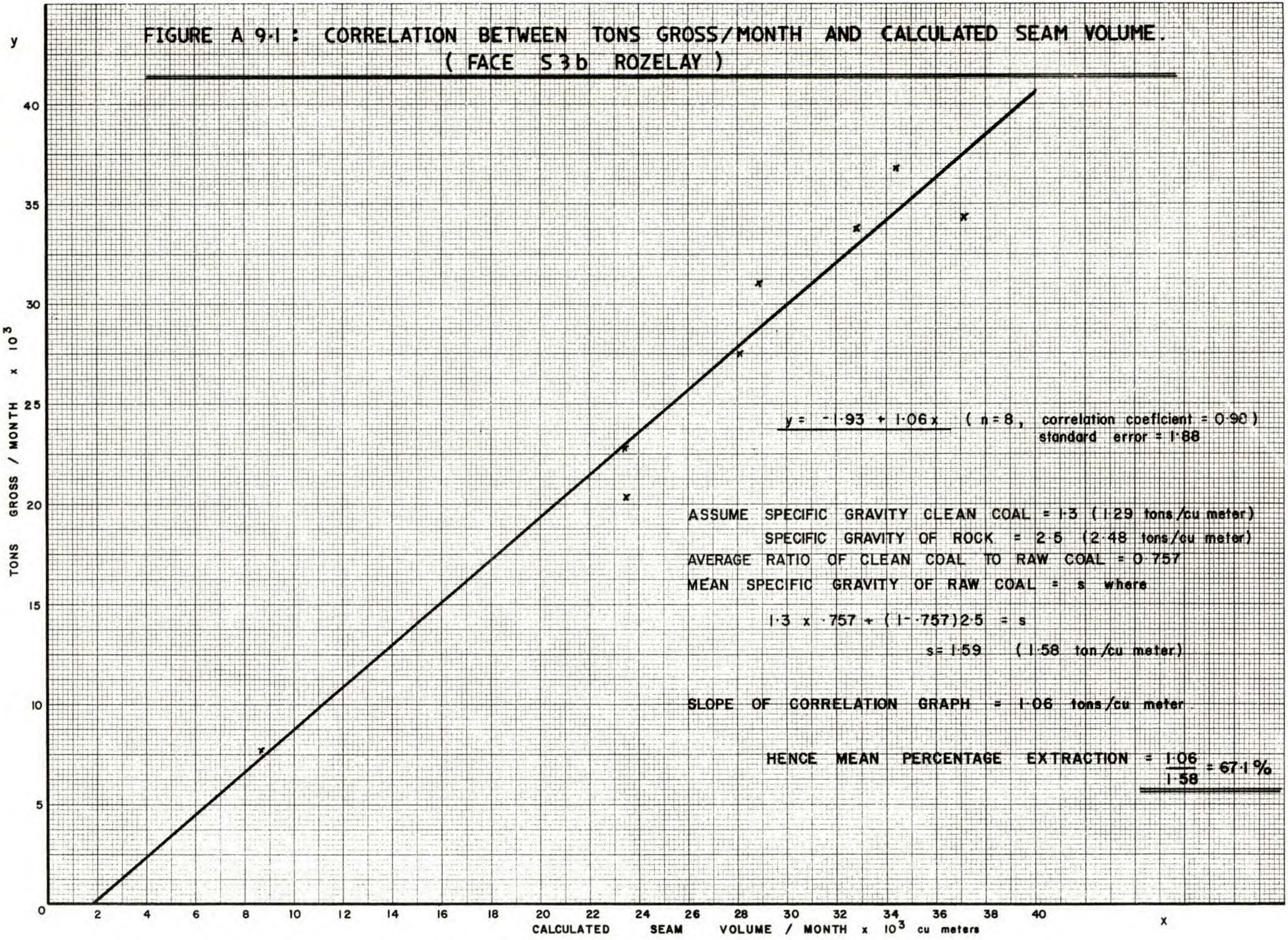


FIGURE A 9·1 : CORRELATION BETWEEN TONS GROSS/MONTH AND CALCULATED SEAM VOLUME.
(FACE S3b ROZELAY)

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TABLE A9.2

Productivity per Unit Face Advance, per Unit Face Length, per Unit Thickness

Productivity tons gross/ man shift	Thickness t metres	Length l metres	Advance a metres	$Y_1 = \frac{P}{at}$	$Y_2 = \frac{P}{lt}$
13.7	12.9	110	.86	.145	.0096
20.8	12.7	110	1.18	.160	.0148
19.8	11.9	110	1.15	.156	.0151
18.7	12.15	110	1.30	.131	.014
19.8	12.15	110	1.32	.136	.0148
13.8	12.15	110	0.83	.151	.0103
15.0	12.15	110	0.96	.142	.0112
12.1	11.9	110	0.83	.132	.0092

Attempted correlation

1. $Y_1 = 0.169 + 0.0104\ t \quad (\tau = 0.344,\ n = 8)$
   No significant correlation

2. $Y_2 = .00043 + .0113\ a \quad (\tau = .927,\ n = 8)$
   Significant to better than 1% level

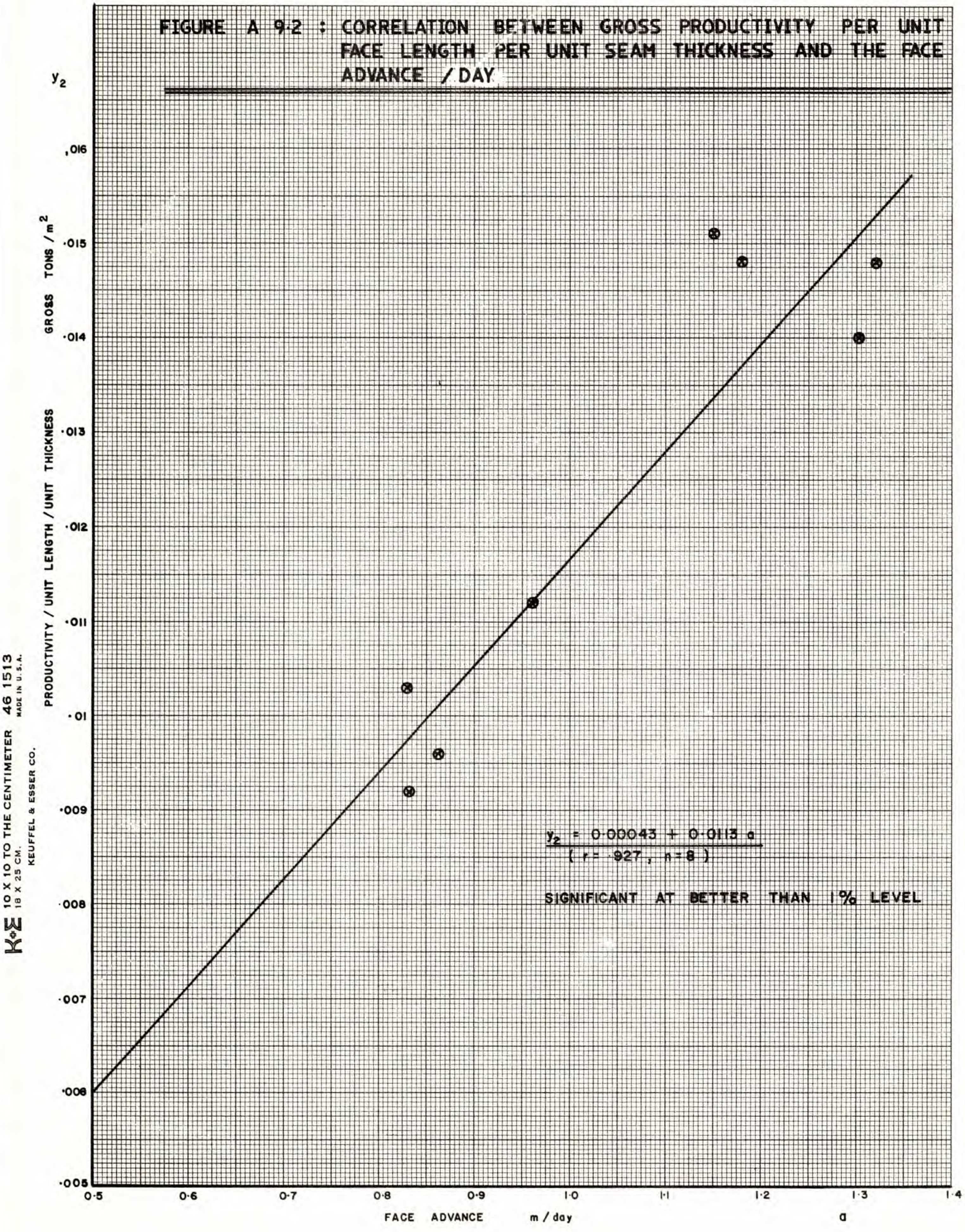


FIGURE A 9·2 : CORRELATION BETWEEN GROSS PRODUCTIVITY PER UNIT FACE LENGTH PER UNIT SEAM THICKNESS AND THE FACE ADVANCE / DAY

TABLE A9.3

Panel Statistics - Rozelay S3b

	ITEM	MAR	APR	MAY	JN	JLY	AUG	SEPT	OCT	MEAN PANEL
1	No. days worked	23	20	23	22	5	21	22	22	19.8
2	Production - tons net	25,386	27,377	28,473	26,155	5,742	20,617	24,201	16,684	21,829
3	Mean daily production - tons net	1,103	1,368	1,237	1,188	1,148	982	1,100	758	1,110
4	Tons net/tons gross %	72.4	74.5	74.8	76.3	75.9	80.5	75.8	74.9	75.6
5	Production - tons gross	35,063	36,747	38,065	34,279	7,565	25,611	31,927	22,275	28,941
6	Mean daily production - tons gross	1,524	1,837	1,655	1,558	1,513	1,219	1,451	1,012	1,471
7	Shifts per 1000 tons net									
	(a) on the face						79.5	75.2	99.7	84.9
	(b) development						29.4	30.1	34.9	31.5
	(c) panel services						60.1	61.1	87.7	69.8
	Total shifts/1000 tons	179.3	141.1	151.9	151.1	195.0	169.1	166.4	222.4	163.0
8	Shifts per day									
	(a) on the face						78.1	82.8	75.6	78.8
	(b) development						28.9	33.1	26.5	29.5
	(c) panel services						59.1	67.2	66.5	64.3
	Total shifts/day	178.2	193.1	188.0	179.6	233.9	166.1	183.1	168.6	180.9
9	Productivity tons net/man shift	6.19	7.09	6.58	6.62	5.13	5.99	5.89	4.49	6.13
10	Productivity tons gross/man shift	8.55	9.51	8.80	8.67	6.76	7.34	7.92	6.00	8.13

Ratio of face-panel labour = 78.8/180.9 = .436

Table A9.4 gives a cost breakdown for this panel.

Table A9.5 gives development statistics for this mine.

Development

Assume an average panel is 380 m long by 125 m wide by 12.25 m thick = 5.818×10^5 cu metres.

Assume raw coal density = 1.58 tons/cu metre.

Tonnage of raw coal in panel = 9.192×10^5 tons; tons net/tons gross = 0.756
Tons net in the panel = 6.949×10^5 tons
Tons net extracted at 67.1% extract = 4.662×10^5

Total development required for panel = 5.71 metres/1000 tons net
= 5.71 x 466.2 = 2662 metres

If average face length is 100 m, this leaves 2522 metres for gate roads and access roads. This splits roughly in 380 m each for head & tailgates, leaving an additional 1800 metres for development. This extra 1800 m seems excessive and too much reliance should not be placed in this figure; it would appear likely that the development rate over this period is in excess of that required to develop a replacement panel in the lifetime of this panel. Hence the metres/1000 tons required will be less than that being averaged over this time period.

Approximate cost/metre

No. metres/month = 1172
No. shifts/month = 639

Labour costs at 170 F/shift = 639 x 170 =	F	108,630	
Labour cost/metre =	F	926.8	\$185.4
Assume labour is 60% of costs			
Total cost/metre =	F	1,544/m =	\$309/m

TABLE A9.4

Cost Analysis - Rozelay Panel 3 (containing face S3b)

	ITEM	MAR	APR	MAY	JUNE	JLY	AUG	SEPT	MEAN		ATTRIBUTABLE TO AVERAGE FACE
A	GENERAL PANEL STATISTICS										
1	No. days worked/month	23	20	23	22	5	21	22	19.8		19.8
2	Average shifts/day	178.2	193.1	188.0	179.6	223.9	166.1	183.1	180.9		78.8
3	Production tons/net	25,386	27,377	28,473	26,155	5,742	20,617	24,201	21,829		20,180
4	Production tons/net/day	1,103	1,368	1,237	1,188	1,148	982	1,100	1,110		1,035
5	% net coal/raw coal	72.4	74.5	74.8	76.3	75.9	80.9	75.8	75.8		75.6
6	Productivity tons net/ man shift	6.19	7.09	6.58	6.62	5.13	5.99	5.89	6.13		13.0
B	LABOUR COSTS								F/ton		
7	Salaries - underground F/ton	11.28	10.88	10.55	11.12	15.48	13.12	12.88	12.18	Prorated	5.31
8	Additional emoluments F/t	1.01	1.02	0.93	1.08	4.17	1.00	1.14	1.48	by ratio of	0.65
9	Surface workers charged to panel F/t	0.10	-	-	-	-	-	-	0.01	face/panel	-
10	Bonus on results F/t	2.32	2.22	2.14	2.28	3.67	2.64	2.62	2.56	Labour	1.11
11	Fringe benefits F/t	11.05	10.60	10.76	11.45	18.44	13.26	13.14	12.67	= 0.436	5.52
12	Injuries, absenteeism F/t	0.63	0.46	0.73	0.58	1.95	0.52	0.64	0.79		0.34
	Total Labour Costs F/t	26.43	25.21	25.13	25.61	43.73	30.57	30.44	29.64		12.93
C	SUPPLIES										
13	Timber F/t	1.22	0.96	0.74	1.09	1.72	1.35	1.70	1.25	70% to face	0.87
14	Supports (arches, friction props)	0.63	0.80	1.15	0.12	-	0.16	0.32	0.45	0 to face	-
15	Walking props	1.17	1.77	1.99	1.35	1.11	2.75	1.32	1.64	100% face	1.64
16	Explosives	0.06	0.16	0.06	0.05	-	0.08	-	0.06	0 to face	-
17	Dismantling & loading	1.60	0.71	0.59	0.24	0.35	1.90	1.01	0.77	100% face	0.77
18	Conveyors, etc.	3.20	0.73	0.33	0.11	0.16	0.12	0.38	0.72	50% to face	0.36
19	Monorail	-	-	-	0.02	-	0.01	0.11	0.02	0 to face	-
20	Electrical	0.19	0.07	0.23	0.10	2.34	0.15	0.17	0.46	50% to face	0.22
21	Others	1.71	1.17	0.89	0.64	2.09	0.57	1.90	1.28	50% to face	0.64
	Total Supplies F/t	9.83	6.41	6.04	3.76	7.79	7.16	5.24	6.65		4.51

TABLE A9.4 (continued)

	ITEM	MAR	APR	MAY	JUNE	JLY	AUG	SEPT	MEAN		ATTRIBUTABLE TO AVERAGE FACE
D	RENTALS (from B.E.F)										
22	Walking props F/t	6.58	5.27	5.91	6.14	6.35	5.73	5.28	5.89	100% face	5.89
23	Dismantling & loading	1.63	1.76	2.69	1.36	1.41	2.47	2.28	1.93	100% face	1.93
24	Conveyors	1.25	1.13	1.27	1.24	1.27	1.33	0.91	1.20	50% face	0.60
25	Monorail	0.09	0.06	0.06	0.07	0.06	0.06	0.05	0.06	0 to face	-
26	Electrical	0.81	0.69	0.74	0.71	0.74	0.79	0.69	0.74	50% to face	0.37
27	Others	0.25	0.20	0.19	0.18	0.20	0.25	0.23	0.21	50% to face	0.10
	Total Rentals	10.63	9.13	10.85	9.73	10.06	10.66	9.47	10.03		8.89
E	MAINTENANCE										
28	Supports (arches, friction props)	-	-	-	0.12	0.18	-	-	0.04	0 to face	-
29	Walking props	-	0.04	0.06	0.05	-	0.06	0.03	0.03	100% face	0.03
30	Dismantling & loading	0.01	0.05	0.06	-	-	-	0.01	0.02	50% face	0.01
31	Conveyors	0.01	-	-	0.14	0.22	-	-	0.05	50% face	0.03
32	Monorail	-	-	-	-	-	-	-	-	-	-
33	Electrical	-	0.02	0.05	0.02	0.84	-	-	0.13	50% face	0.06
34	Other	0.18	0.08	0.04	0.03	0.10	0.10	0.04	0.08	50% face	0.04
	Total Maintenance	0.22	0.22	0.17	0.42	1.35	0.17	0.08	0.35		0.17
	TOTAL COSTS F/ton	47.12	40.99	42.21	40.46	62.95	48.57	45.24	46.67		26.50
	$/ton	9.42	8.19	8.44	8.09	12.59	9.71	9.05	9.33		5.30

TABLE A9.5

Rozelay - Development Statistics Using Alpine Miner

ITEM	AUG	SEPT	OCT	MEAN
1. Metres/1000 tons panel preparation	5.65	7.04	3.87	5.71
2. Total panel production/tons net	20,617	24,201	16,684	20,500
3. Shifts/1000 tons in development	29.4	30.1	34.9	31.5
4. No. working days	21	22	17 (machine under repair)	20
5. Total metres advance/month	116.5	170.5	64.5	117.2
6. Metres advance/working day	5.54	7.75	3.79	5.86
7. Total shifts/month on development	606	728	583	639
8. cm/man shift	19.22	23.42	11.06	18.33

APPENDIX 10. SPECIFICATIONS OF THE SAGEM D.T.S. 300 DOUBLE DRUM SHEARER

TABLE A10.1

Specifications	
Size and Weight	
1. Length	7140 mm
2. Length of machine body, without drums	6100 mm
3. Overall width	1595 mm
4. Minimum height below upper drum position from the base of the chain conveyor	1227 mm
5. Clearance under machine and chain conveyor - for passage of sheared coal	345 mm
6. Total weight	15,556 Kg
Power	
2 motors each of 150 kw	300 kw
Cutting	
Total cutting weight	2950 mm
comprising: height, below upper drum position from base of chain conveyor.	2568 mm
height, thickness of chain conveyor sides.	182 mm
height below the lower position of the drum and base of chain conveyor.	200 mm
Width of cut	650 mm
Movement of the Shearer	
Drive wheel	6 teeth
Speed of advance - slow speed	5 m/min
Speed of advance - mean speed	7.5 m/min
Speed of advance - high speed	15 m/min
Winding drum force - slow speed	26 tons
Winding drum force - mean speed	17 tons
Winding drum force - high speed	8 tons
Diameter of traction chain	22 mm
Step of traction chain	86 mm
Test load of traction chain	49 tons
Cutting Drums	
Drum diameter (attached to body by canual hub with hydraulic system for dismantling)	1600 mm
No. of pick positions per drum	70
comprising: on each of the 2 helical discs.	15
on the disc face.	40
Speed of drum rotation	67.5 revs/min
Ranging speed of drum, to the rise	675 mm/min
Ranging speed of drum, on descent	540 mm/min

FIGURE A10·1 : SAGEM DTS 300 DOUBLE DRUM SHEARER

APPENDIX 11. ANALYSIS OF THE MINING CYCLE ON FACE S3b; CALCULATION OF CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COSTS

A11.1 Breakdown of Working Cycle

Figure 4.8 in the text shows a breakdown of the working cycle for advance of face S3b for two shifts studied by the mine persônnel. Figure A11.1 shows a similar breakdown based on study shifts carried out by the author. The Table A11.1 summarizes the times for various jobs on the face based on 2 shifts studied on the 16/5/72 and another 2 shifts studied on the 3 and 6 of 11/72; based on these values, mean values are given which will be used as representative values for calculation purposes.

A11.2 Estimated Maximum Rate of Face Advance in Ideal Conditions

It is seen from Table A11.1 that the utilization efficiency of the shearer is only between 20 - 27%; because the machine must wait for the prop advance crews to catch up. The following calculations therefore are aimed at determining the maximum rate of face advance that might be achieved after removing this bottleneck by putting more prop advance crews on the face.

1. Assume a 100 m face length, i.e. 64 powered supports
2. Assume that the machine cuts for 100% of the available time (less turn around time)
3. Assume that the number of prop advance crews are increased to cope with this; calculate how many crews required.

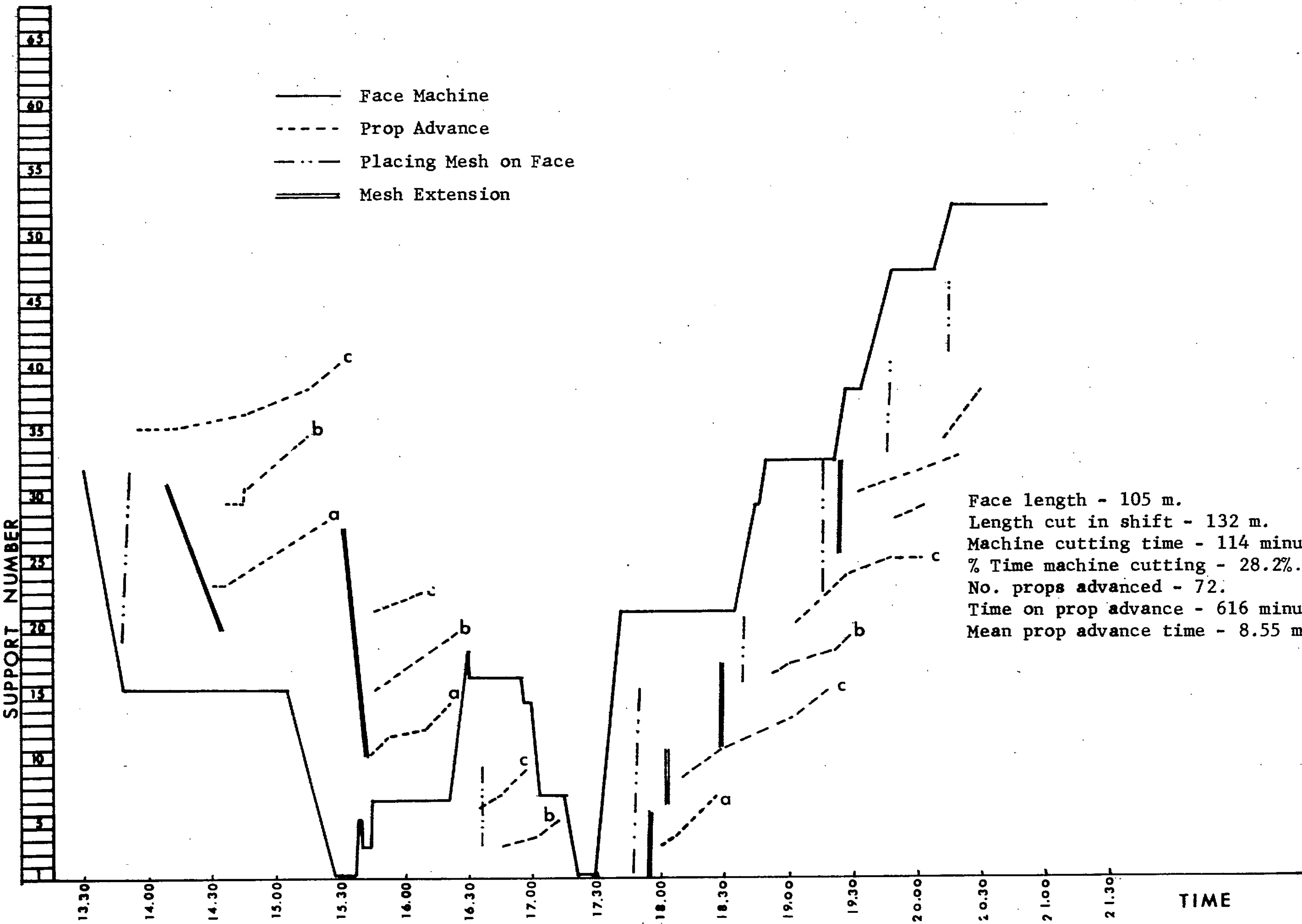
From Table A11.1 the machine cuts 523 metres in 402 minutes actual cutting time, i.e. it will take 76.8 ≈ 80 minutes to cut along a 100 m face. Allow 115 minutes for turn around time at each end of the face.

4. Assume that total time available, per production shift, on face is 420 minutes. (7 hours). Then we have:

		Time	Cumulative time
Machine Pass No. 1:	Cuts 100 m	80 min	80 min
	Turn around	115 min	195 min
Pass No. 2:	Cuts 100 m	80 min	275 min
	Turn around	115 min	390 min
Starts Pass No. 3, unable to complete, completes 0.375 pass in 30 min		30 min	420 min

i.e. Maximum possible rate for the machine on a 100 m face is 2.375 passes/ production shift.

FIGURE A 11·1 : ROZELAY - FACE S36 - AFTERNOON SHIFT - MONDAY 6TH NOVEMBER 1972



Face length - 105 m.
Length cut in shift - 132 m.
Machine cutting time - 114 minutes.
% Time machine cutting - 28.2%.
No. props advanced - 72.
Time on prop advance - 616 minutes.
Mean prop advance time - 8.55 minutes

TABLE A11.1

Face Advance Time Utilization

ITEM	16/5/72	$^{3}_{6}$/11/72	Representative value
1. Total working time on face - 2 shifts	865 minutes	795 minutes	420 mins/shift
2. Total face length	108 metres	105 metres	100 metres
3. Length of face cut during 2 shifts	286 metres	237 metres	
4. No. of passes of shearer	2.64	2.29	
5. Time of shearer actually cutting	228 minutes	174 minutes	
6. Efficiency of shearer utilization % $^{(5)}/_{(1)}$ x 100	26.4%	21.8%	
7. Machine turn around time at 1st end	112 min	20 min	
at 2nd end	116 min	123 min	
Total	228 min	143 min	
8. Total number of props advanced	171	139	
9. Total time spent advancing props	1019 min	1142 min	
10. Average advance time per prop	5.95 min	8.20 min	- 7 min/prop
11. No. of 2 man prop advance crews (over the 2 shifts)	6	6	
12. No. of props advanced per crew per shift	28.5	23.2	- 25 props/crew/shift
13. Time spent on mesh extension (same crews as for prop advance)	222 min	no record	
14. Time spent snaking front conveyor	199 min	no record	
15. Time spent moving conveyor motors forward (estimated)	50 minutes each end	no record	

Manpower distribution on face - per shift		16/5/72	$^{3}_{6}$/11/72 (1)	$^{3}_{6}$/11/72 (2)	Representative value
(a) Face advance	Advancing props			6	
	Machine operators			2	
	Snaking front conveyor			2	
	Snaking rear conveyor			2	
	Conveyor operators on face			1	
	Headgate niche			1	
	Tailgate niche			2	

TABLE A11.1 (continued)

ITEM		16/5/72	$^{3}_{6}$/11/72		Representative value
			(1)	(2)	
(b) Caving & drawing	Hydraulic technicians			3	
	Shift boss			1	
	Drawing			5	
	Conveyor operator			1	
	Shot firer			1	
	Supervisor			1	
	TOTAL			28	

5. Thus the total number of props to be advanced for 2.375 passes is 151. Assume that 1 prop advance crew (2 men) can advance 25 props per shift. Then the number of prop advance crews required to achieve this rate is $\frac{151}{25}$ = 6 crews per shift (compared with the current 3 crews per shift).
6. It is now necessary to check on the efficiency of other working crews.
7. Face conveyor crew (2 men): 2 moves of the face conveyor take 100 minutes. Previously 1 crew snaked 268 metres in 199 minutes.

   Hence to snake 237.5 metres takes 156 minutes
   Total face conveyor crew time 256 minutes

   i.e. this crew need not be increased since 420 minutes are available, allowing plenty of time for delays, etc.
8. Production from a 12.25 m seam at 2.375 passes per shift is approximately 1700 tons gross/shift. The rear conveyor can handle up to 500 tons/hour, i.e. the conveyor must operate continuously for 203 minutes to remove the coal, i.e. a rear conveyor efficiency of 48% is required; this seems reasonable even allowing for time spent on unblocking draw points and breaking up lumps. However a limit of 50% rear conveyor efficiency with current equipment has been set for the ensuing calculations.
9. However to pull this amount of coal per shift will probably require more personnel on caving and drawing. Let us assume that it is necessary to double the caving and drawing crew from 5 to 10 men. In addition, the number of men required to advance the headgate and tailgate niches, and the number of hydraulic technicians will also be increased.
10. This results in the following face manpower distribution now necessary to achieve this increased rate of face advance:-

TABLE A11.2

New Face Manpower Distribution Required to Achieve Maximum Rate of Face Advance, per shift

On face advance:	Advancing props	12
	Machine operators	2
	Snaking front conveyor	2
	Snaking rear conveyor	2
	Face conveyor operator	1
	Headgate niche	2
	Tailgate niche	2
	Hydraulic technicians	6
	Shift boss	1
On caving & drawing:	Drawing	10
	Rear conveyor operator	1
	Shot firer	1
	Supervisor	1
	TOTAL	45

A11.3 Maximum Production and Productivity (working 2 shifts with 1 maintenance shift)

No. of men required on face for 2 production shifts	=	90
No. of men required in maintenance shift (unchanged)	=	20
Total face manpower per day	=	110

These men will, at maximum, advance the face by 2 x 2.375 = 4.75 machine passes/day.

Figure A9.1 shows the correlation established between gross production and seam volume; this is given by $y = 1.06x - 1.93$ where y is the gross production x 10^3 tons) and x is the seam volume (m^3) and the percentage extraction was 67.1%.

If n is the number of passes made by the machine per day, ω is the width of cut per pass, t is the seam thickness and l is the face length then.

$$y = n \times t \times l \times \omega \times 1.06 \text{ tons gross/day}$$

or $$y_1 = n \times t \times l \times \omega \times 1.06 \times .757 \text{ tons net/day}$$

Figure A11.2 shows the net daily production variation with number of passes per day for a thickness of cut of 0.55 m, for a face length of 100 m for different thickness seams. ($y = n \times t \times 100 \times .55 \times 1.06 = 58.3$ nt gross tons/day) or ($y_1 = 44.1$ nt net tons/day).

Table A11.3 gives the seam thickness for various daily production levels and number of machine passes per day in a 100 m face.

Hence for n = 4.75 passes per day the maximum daily production would be 3390 tons gross (2570 tons net) and based on the estimated work force given above gives a face productivity of 30.8 tons gross/man shift (23.4 tons net/ms).

A11.4 Comparison of the "Real Case" with the Maximum Theoretical Case

The analysis of the working cycles given in section A11.1 is called the real case and is compared below with the maximum theoretical case calculated above.

However it will be noted that the "real case" given above, although based on actual shift studies, yields production and productivity figures better than those currently averaged over 6 months of the Rozelay face. These average actual figures are given in column 3 of Table A11.4.

Hence the real case above should be scaled down to conform to the actual averages achieved on face S3b and thus likewise the maximum theoretical figures have been scaled down by the same proportions to give an "achievable" figure which more probably reflects the maximum achievable production figures.

These "achievable" figures are therefore:

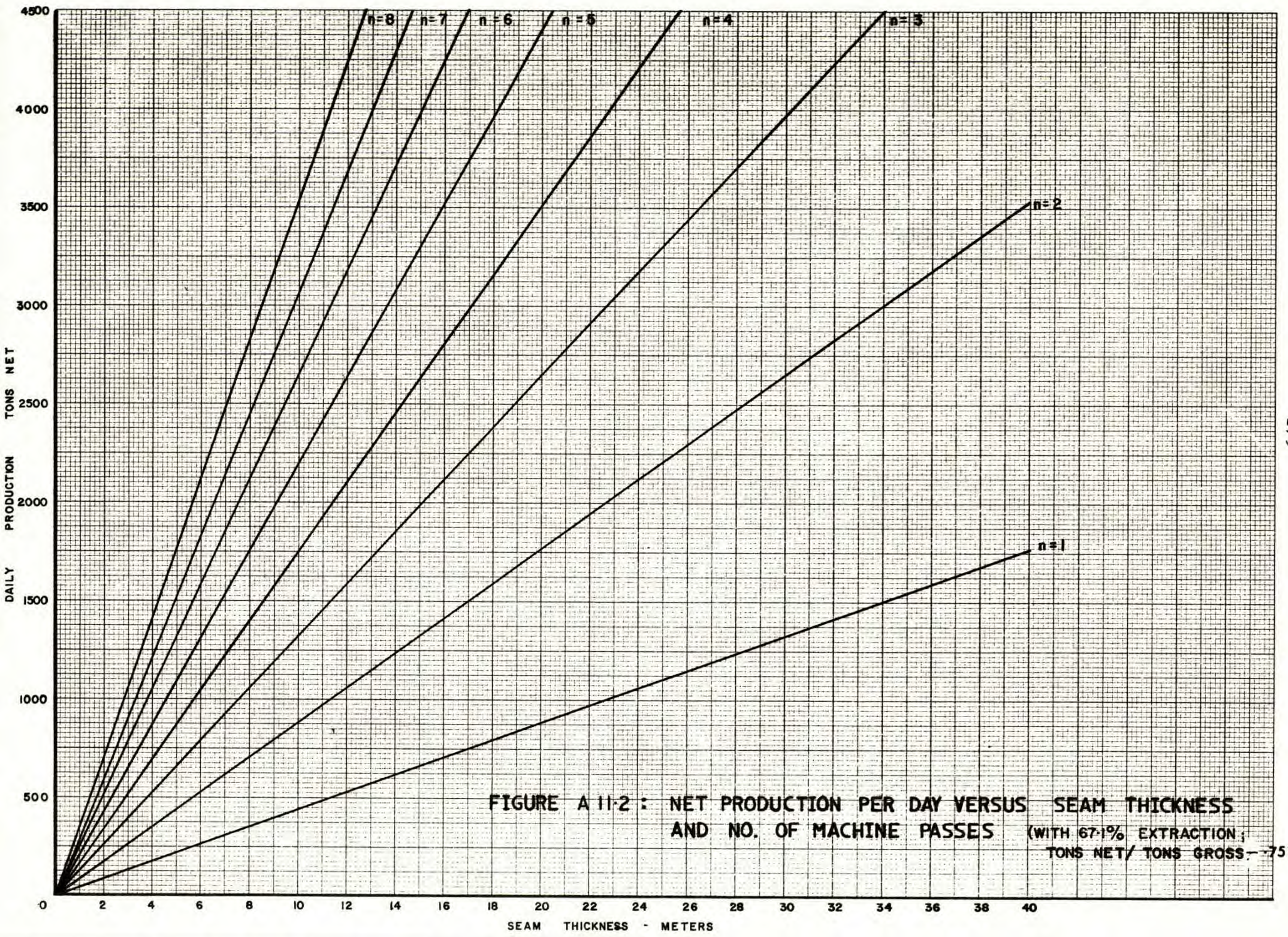


FIGURE A II-2 : NET PRODUCTION PER DAY VERSUS SEAM THICKNESS AND NO. OF MACHINE PASSES (WITH 67·1% EXTRACTION; TONS NET/ TONS GROSS - ·757)

TABLE A11.3

Variation of Production with Seam Thickness and Number of Passes/Day

(100 m face; 0.55 m width of cut. y tons net/day = 44.1 n t)

y_1 tons net/day	Number of passes/day n: 1	2	3	4	5	6	7	8
	Seam Thickness/metres							
500	11.33	5.66	3.77	2.83	-	-	-	-
1000	22.67	11.33	7.55	5.66	4.53	3.77	3.23	-
1500	34.01	17.0	11.33	8.50	6.80	5.66	4.85	4.25
2000	45.3	22.67	15.11	11.33	9.07	7.55	6.47	5.67
3000	-	34.01	22.67	17.0	13.60	11.33	9.71	8.50
4000	-	45.35	30.2	22.7	18.1	15.11	12.95	11.33
5000	-	-	37.8	28.3	22.7	18.89	16.20	14.17

TABLE A11.4

Comparison of Real Case with the Maximum Theoretical Case

ITEM	Real case	Maximum Theoretical case	Current Average over 6 months
No. of machine passes/day - 100 m face	2.4	4.75	1.92
Daily production tons net	1297	2570	1035
tons gross	1714	3390	1368
No. of men on face per day	76	110	796
Productivity tons net/man shift	17.0	23.4	13.0
tons gross/man shift	22.5	30.8	16.7
% increase in face labour	44.7%		
% increase in production	97.8%		
% increase in productivity	37.6%		

No. passes per day	3.80
Daily production tons net	2050
tons gross	2710
No. of men on face/day	110
Productivity tons net	19.90
tons gross	24.6

Scaling down in this manner in fact reduces the machine availability time from 100% to approximately 75% which would seem reasonable taking into account the delays due to breakdown of the machine, face conveyor or gateroad conveyor.

A11.5 Current and Achievable Production and Productivity: Variation with Seam Thickness

In the section above, the current and achievable production and productivity figures apply to a 12.25 m thick seam. Now Figure A11.2 indicates that the production is directly proportional to seam thickness and, if the face manpower is kept constant for each case, this is so regardless of the seam thickness. Hence it is possible to calculate the variation in production and productivity with seam thickness for both the "current" and "achievable" cases. This has been done and is plotted in Figure A11.3. As with all the preceding calculations, the percentage recovery is 67.1% and the ratio tons net/tons gross is .757.

A limit has been imposed on this figure on the assumption that the drawing conveyor will not operate at more than 50% efficiency.

A11.6 Current and Achievable Costs per Gross Ton: Variation with Seam Thickness - for the face

Table A11.5 estimates the current and achievable cost/day for operating the face with 1.92 and 3.80 machine passes per day. These costs are then used as a basis for calculating the variation with seam thickness.

Now the daily costs of operating these faces at 1.92 and 3.80 passes per day will not vary significantly with the seam thickness. Consequently by calculating the daily production (tons gross) from the formula $y = 58.3\ nt$ tons gross/day and putting $n = 1.92$ and 3.8 respectively and keeping the respective daily costs constant, it is possible to calculate the current and achievable costs/gross ton for a range of seam thicknesses; this has been done in Table A11.6 and the results have also been plotted on Figure A11.3. Note: In making use of the graphs in Figure A11.3 comparisons can only be made validly by moving up the vertical axis, i.e. for the same seam thickness. For example:

e.g. 1 For a seam thickness of 15 m, reading from the graph:-

The current daily production is 1670 gross tons/day with a productivity of 20.3 gross tons per man shift on the face at a cost of 16.35 Francs/gross ton. The achievable production is 3300 gross tons/day with a productivity of 30 gross tons/man shift on the face at a cost of 11.25 Francs/gross ton.

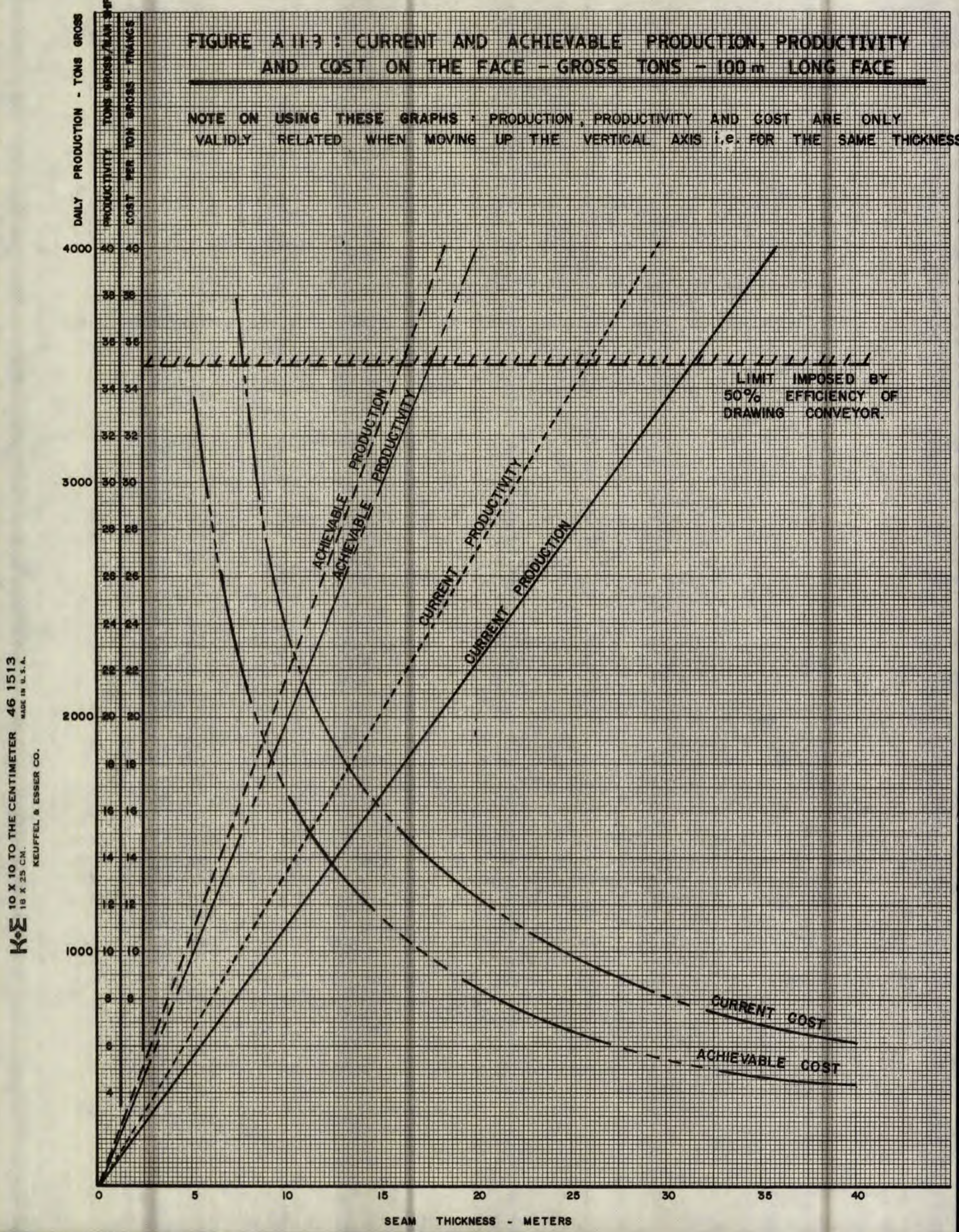


FIGURE A II-3 : CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST ON THE FACE - GROSS TONS - 100 m LONG FACE
NOTE ON USING THESE GRAPHS : PRODUCTION, PRODUCTIVITY AND COST ARE ONLY VALIDLY RELATED WHEN MOVING UP THE VERTICAL AXIS i.e. FOR THE SAME THICKNESS
DAILY PRODUCTION - TONS GROSS
PRODUCTIVITY TONS GROSS/MAN SHIFT
COST PER TON GROSS - FRANCS
LIMIT IMPOSED BY 50% EFFICIENCY OF DRAWING CONVEYOR.
ACHIEVABLE PRODUCTION
ACHIEVABLE PRODUCTIVITY
CURRENT PRODUCTIVITY
CURRENT PRODUCTION
CURRENT COST
ACHIEVABLE COST
SEAM THICKNESS - METERS
4000
3000
2000
1000
0
5
10
15
20
25
30
35
40
K+E 10 X 10 TO THE CENTIMETER 18 X 25 CM. 46 1513 MADE IN U.S.A.
KEUFFEL & ESSER CO.

TABLE A11.5

Current and Achievable Costs for 100 m Face in 12.25 m Thick Seam

ITEM	Current	Comment	Achievable
1. No. of passes per day 2. Seam thickness - metres 3. Daily production - tons gross 4. Face productivity - tons gross/man shift	1.92 12.25 1,368 16.7		3.80 12.25 2,710 24.6
Labour costs 5. Labour (79 men on face/day) =	12.3 F/ ton net 9.79 F/ ton gross	Labour 110 men on face/day	9.09 F/ ton net 6.88 F/ ton gross
6. Daily labour costs	13,390 F/day		18,644 F/day
7. Supplies (from Table 4.3) Cost Francs/ton net Cost Francs/ton gross	 4.51 3.41		
8. Total daily cost of supplies	4,670 F/day	Increase proportional no. of passes	9,240 F/day
9. Rental costs (from Table 4.3) Cost Francs/ton net Cost Francs/ton gross	 8.89 6.73		
10. Total daily cost of rentals F/day	9,206 F/day	Remains the same	9,206 F/day
11. Maintenance (from Table 4.3) Cost Francs/ton net Cost Francs/ton gross	 0.17 0.13		

TABLE A11.5 (continued)

ITEM	Current	Comment	Achievable
12. Total daily cost of maintenance	176 F/day	Increase proportional no. of passes	348 F/day
13. TOTAL DAILY COST FOR OPERATING FACE (6 + 8 + 10 + 12)	27,442 F/day		37,348 F/day
14. Cost/ton net - 12.25 m seam	26.50 F/ ton		18.25 F/ ton
Cost/ton gross - 12.25 m seam	20.06 F/ ton		13.81 F/ ton

TABLE A11.6

Current and Achievable Cost/Gross Ton on Face: Variation with Seam Thickness

Seam thickness metres	CURRENT		ACHIEVABLE	
	Tons gross/day	Cost F/ton	Tons gross/day	F/ton
5	559	49.03	1107	33.71
10	1119	24.51	2215	16.95
15	1679	16.34	3323	11.23
20	2238	12.25	4430	8.42
25	2798	9.80	5538	6.74
30	3358	8.17	6646	5.61
35	3917	7.00	7753	4.81
40	4477	6.12	8861	4.21

e.g. 2 For a seam thickness of 20 metres. The current figures would be 2250 gross tons/day, 27.1 tons/man shift at 12.30 Francs/ton. For the achievable case, at 20 metres thickness, the limit implied by a drawing conveyor capacity of only 50% is invoked. Theoretically the production would be 4360 tons/day but this is more than the rear conveyor can handle; thus in this case the maximum daily production figure of 3500 tons per day is used and the corresponding productivity and costs are (reading down the vertical axis) 31.8 tons/man shift at cost of 10.6 Francs/ton. Thus in this case, unless a higher capacity rear conveyor is used or a better conveyor efficiency can be achieved, the full potential advantages of increasing the number of prop advance crews could not be realised. Assuming that this limitation cannot be overcome, then the problem should be re-examined to see if the number of prop advance crews could be reduced, thus increasing the productivity and decreasing the cost.

A11.7 Current and Achievable Costs per Gross Ton: Variation with Seam Thickness - for the panel

A similar analysis to that given in section 11.6 above for the face will now be carried out for the panel. It will be assumed in this case that production comes entirely from the face, i.e. the development production is insignificant. It will also be assumed that the amount of development required will increase directly in proportion to the face advance rate, i.e. to the number of passes of the machine. Finally it will be assumed that the labour on the panel services must be increased by 25% to above the current level to cope with the achievable level. Table A11.7 gives the resulting current and achievable panel costs.

As before, this can now be related to seam thickness and likewise the panel productivity can also be calculated. This has been done in Table A11.8 and the results are plotted in Figure A11.4.

TABLE A11.7

Current and Achievable Costs for 100 m Face, 12.25 m Seam - Panel Costs

ITEM	Current		Achievable
1. No. of machine passes/day 2. Seam thickness 3. Daily production gross tons 4. Panel productivity gross tons/man shift	1.92 12.25 1,368 7.6		3.80 12.25 2,710 -
5. Labour distribution No. man shifts/day on face No. man shifts/day in development No. man shifts/day on services to panel	 78.8 29.5 64.3	 Proportional to no. of passes Increase by 25%	 110 58.4 80.4
6. Total panel labour	180.9		248.8
7. Labour costs Cost Francs/ton net Cost Francs/ton gross	 29.64 22.44		
8. Total labour cost/day	30,694 F/day	~169.97 Francs/ man shift	42,215 F/day
9. Supply costs (from Table 4.3) Cost Francs/ton net Cost Francs/ton gross	 6.65 5.03		
10. Total daily supply costs	6,886 F/day	Increase proportional no. of passes	13,630 F/day
11. Rental costs (from Table 4.3) Cost Francs/ton net Cost Francs/ton gross	 10.03 7.59		

TABLE A11.7 (continued)

ITEM	Current		Achievable
12. Total rental costs	10,387 F/day	Will remain the same	10,387 F/day
13. <u>Maintenance costs</u> (from Table 4.3) Costs Francs/ton net Costs Francs/ton gross	 0.35 0.265		
14. Total maintenance costs	362 F/day	Increase proportional no. of passes	717 F/day
15. Total daily operating cost for panel	48,329 F/day		66,949
16. Panel cost/ton net - 12.25 m seam	46.67		32.63
17. Panel cost/ton gross - 12.25 m seam	35.32		24.70

TABLE A11.8

Current and Achievable Cost/Gross Ton and Productivity for the Panel: Variation with Seam Thickness

Seam thickness metres	Current			Achievable		
	Tons gross/day	Cost F/ton	Productivity	Tons gross/day	Cost F/ton	Productivity
5	559	86.45	3.09	1107	60.47	4.44
10	1119	43.18	6.18	2215	30.22	8.90
15	1679	28.78	9.28	3323	20.14	13.35
20	2238	21.59	12.37	4430	15.11	17.80
25	2798	17.27	15.46	5538	12.08	22.25
30	3358	14.39	18.56	6646	10.07	26.71
35	3917	12.33	21.65	7753	8.63	31.16
40	4477	10.79	24.74	8861	7.55	35.61

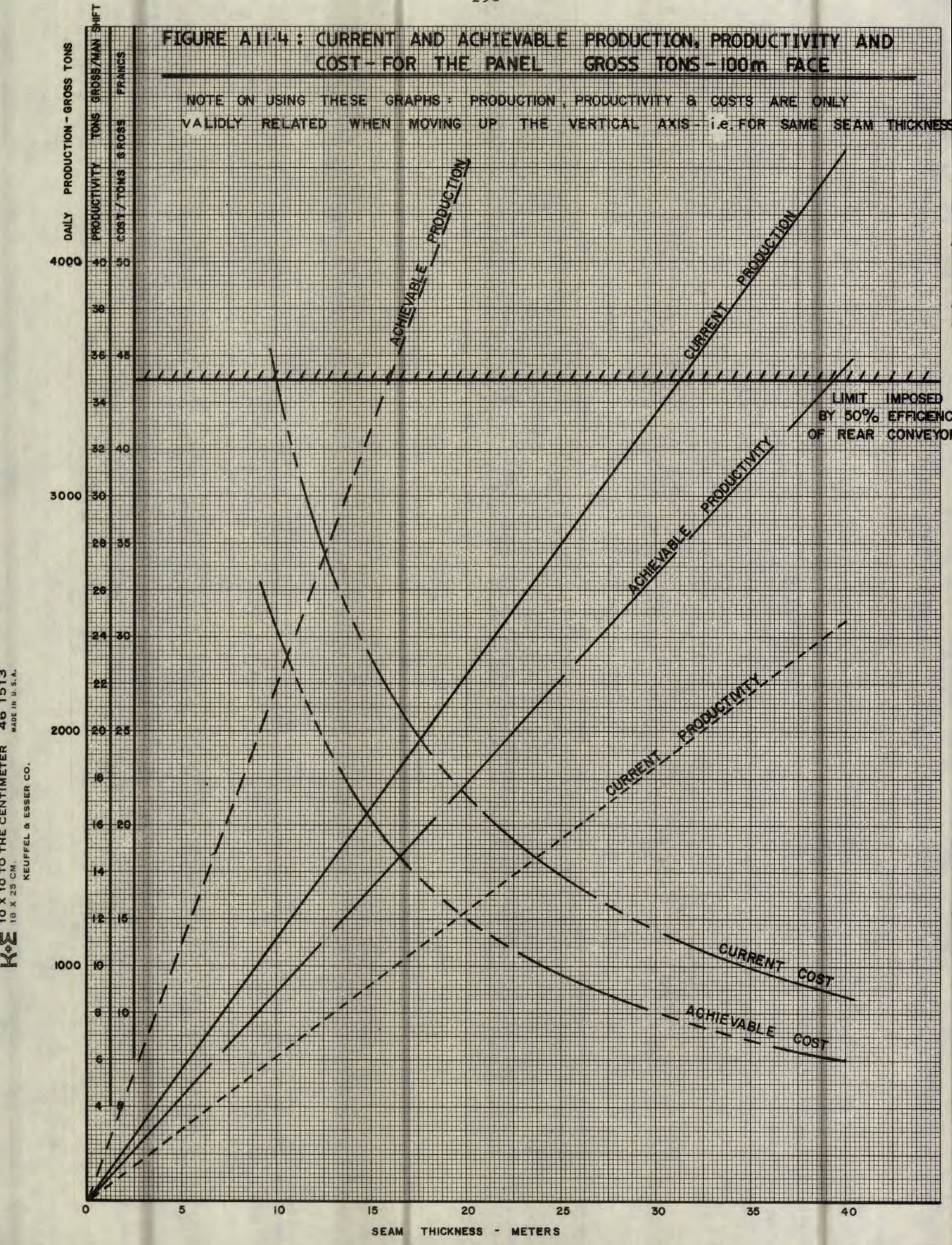


FIGURE A11-4 : CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST - FOR THE PANEL GROSS TONS - 100m FACE
NOTE ON USING THESE GRAPHS : PRODUCTION, PRODUCTIVITY & COSTS ARE ONLY VALIDLY RELATED WHEN MOVING UP THE VERTICAL AXIS - i.e. FOR SAME SEAM THICKNESS
DAILY PRODUCTION - GROSS TONS
PRODUCTIVITY TONS GROSS/MAN SHIFT
COST / TONS GROSS FRANCS
ACHIEVABLE PRODUCTION
CURRENT PRODUCTION
ACHIEVABLE PRODUCTIVITY
CURRENT PRODUCTIVITY
CURRENT COST
ACHIEVABLE COST
LIMIT IMPOSED BY 50% EFFICIENC
OF REAR CONVEYOR
SEAM THICKNESS - METERS
0
5
10
15
20
25
30
35
40
1000
2000
3000
4000
K+E 10 X 10 TO THE CENTIMETER 18 X 25 CM. 46 1513 MADE IN U.S.A.
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APPENDIX 12. CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COSTS FOR CANADIAN CONDITIONS

A12.1 Assumptions

The analysis of the Rozelay mine given previously in Appendix 11 will differ from Canadian conditions in several aspects:-

1. The relation between face advance and production for Rozelay yielded an extraction ratio of 67.1%; whereas for Darcy the figure was 95%. The difference was attributed to the presence of the thick sandstone bed in the Rozelay seam; such a condition is unlikely to exist in Canada, consequently a figure of 90% will be assumed in these calculations.

2. The labour rate in Canada is higher than that in France. A cost of \$50 (F250)/shift will be assumed for labour in Canada.

3. It is thought that the number of men on the face given in the calculations in Appendix 11 (110 man/day) represent a maximum and that this might be reduced as follows

on the face:-	Advancing props	12
	Machine operators	2
	Snaking front & rear conveyors	2
	Face conveyor operator	1
	Headgate niche	2
	Tailgate niche	2
	Hydraulic technicians	4
	Shift boss	1
on caving and drawing:-	Drawing	8
	Rear conveyor operator	1
	Shot firer	1
		36 x 2 shifts

+ 20 on maintenance shift = 92 shifts/day.

It is also assumed that the remaining labour in the panel could be reduced by 20%, i.e. from 138.8 shifts/day to 111 shifts/day for a total panel labour force of 203 shifts/day.

4. It is assumed that other costs remain the same.

A12.2 Production per Pass for 100 m Face, 90% Extraction; Current and Achievable Production and Productivity

For 90% extraction the relationship between gross production and thickness will be $y = 1.42\ n\ t\ l\ w$ tons gross/day, which for a face length of 100 m and a width of machine cut of 0.55 m gives $y = 78.1\ n\ t$ where n is the number of passes per day of the machine and t is the seam thickness.

Hence the current and achievable production and productivity on the face and on the panel for different seam thicknesses, can be calculated as before, assuming 1.92 passes/day and 3.80 passes per day of the machine with face and panel labour of 79 and 180.9 shifts per day respectively for the current labour and 92 and 203 shifts per day for the achievable labour. The variations with seam thickness have been calculated for both these cases and the results are plotted in Figure A12.1 for the face and Figure A12.2 for the panel. Table A12.1 gives a tabulation of these results.

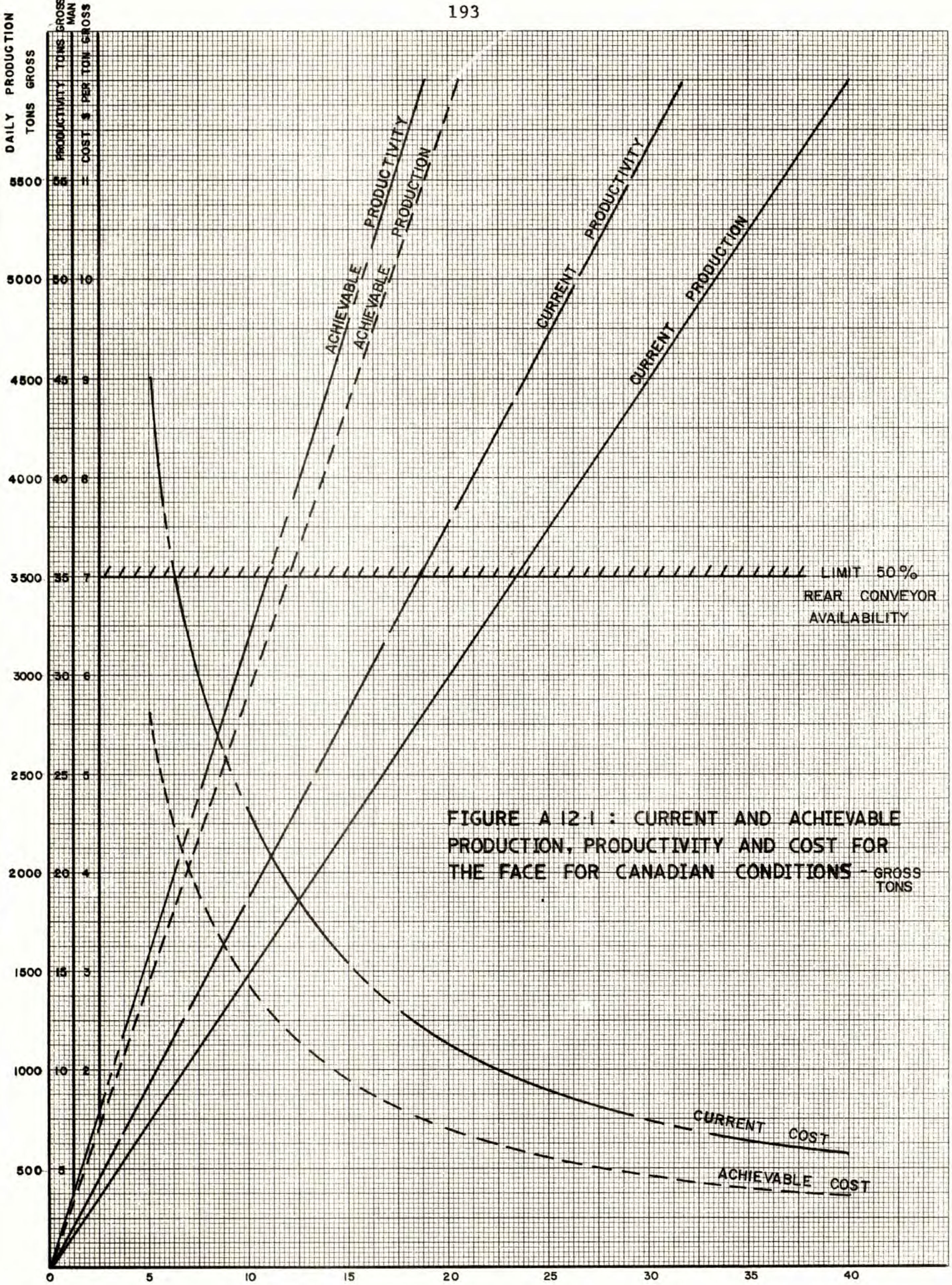

FIGURE A 12·1 : CURRENT AND ACHIEVABLE PRODUCTION, PRODUCTIVITY AND COST FOR THE FACE FOR CANADIAN CONDITIONS - GROSS TONS

FIGURE A12-2: CURRENT AND ACHIEVABLE PRODUCTION PRODUCTIVITY AND COSTS FOR THE PANEL FOR CANADIAN CONDITIONS – GROSS TONS

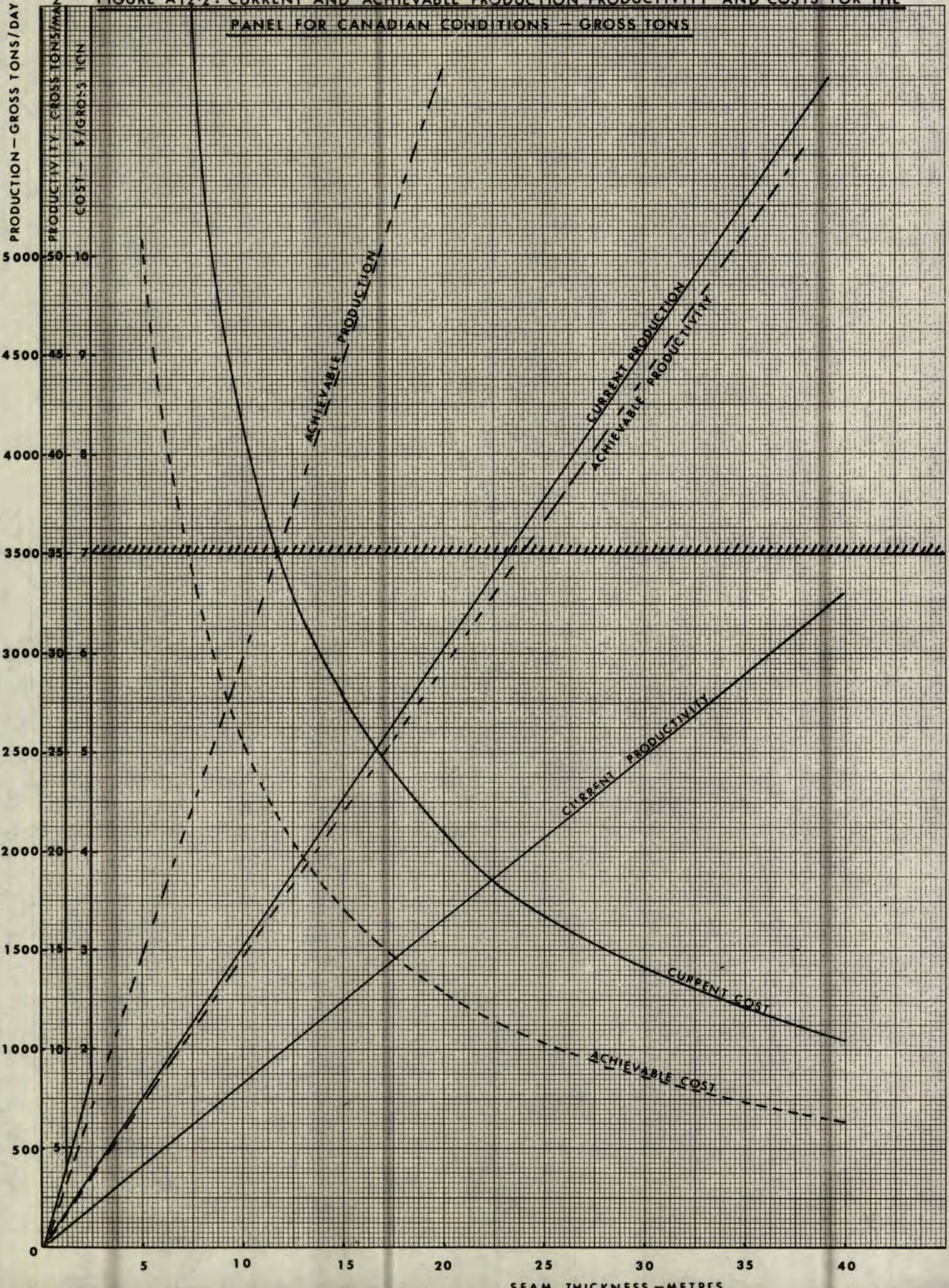

TABLE A 12.1

Current and Achievable Production, Productivity and Operating Costs, Variation with Seam Thickness - Canadian Conditions

Seam thickness Metres	CURRENT			ACHIEVABLE		
	Production tons gross/day	Face productivity tons gross/m.s	Panel productivity tons gross/m.s	Production tons gross/day	Face productivity tons gross/m.s	Panel productivity tons gross/m.s
5	750	9.50	4.15	1484	16.13	7.31
10	1499	18.97	8.28	2968	32.26	14.62
15	2249	28.46	12.43	4451	48.38	21.93
20	2990	37.84	16.53	5936	64.52	29.24
25	3748	47.44	20.72	7420	80.65	36.55
30	4498	56.94	24.86	8903	96.77	43.85
35	5248	66.43	29.01	10,387	112.90	51.16
40	5998	75.92	33.16	11,871	129.03	58.47

Operating costs

ITEM	Current		Achievable	
	Face	Panel	Face	Panel
1. No. passes/day	1.92		3.80	
2. Face labour m.s /day	79		92	
3. Panel labour m.s/day		180.9		203
Labour costs at $50/man shift	F 19,750	F 45,225	F 23,000	F 50,750
per day	$ 3,950	$ 9,045	$ 4,600	$ 10,150
4. Supplies	F 4,670	F 6,886	F 9,240	F 13,630
	$ 934	$ 1,377	$ 1,848	$ 2,726
5. Rentals	F 9,206	F 10,387	F 9,206	F 10,387
	$ 1,841	$ 2,077	$ 1,841	$ 2,077
6. Maintenance	F 176	F 362	F 348	F 717
	$ 35	$ 72	$ 69	$ 143

TABLE A12.1 (continued)

Cost/gross ton variation with seam thickness

Seam thickness metres	CURRENT			ACHIEVABLE		
	Production	$/ton face	$/ton panel	Production	$/ton face	$/ton panel
5	750	9.01	16.76	1484	5.63	10.17
10	1499	4.51	8.39	2968	2.82	5.09
15	2249	3.00	5.59	4451	1.88	3.39
20	2990	2.26	4.20	5936	1.41	2.54
25	3748	1.80	3.35	7420	1.13	2.03
30	4498	1.50	2.80	8903	0.94	1.69
35	5248	1.29	2.39	10387	0.80	1.45
40	5998	1.13	2.10	11871	0.70	1.27