

Mining of Narrow Steeply Dipping Veins

By B E Hall¹

Introduction

Many mineral deposits occur as steeply dipping narrow veins. In the past, these have been worked either by high cost labour intensive methods which are able to follow the vein with minimal dilution for high grade production, or by lower cost mechanized methods with large equipment, with increased dilution levels and lower production grades. A number of mines overseas are introducing mechanization into steep narrow vein mining, obtaining the increased productivity which this brings, but maintaining the advantage of high grade production associated with narrow stope widths.

This paper reports on methods used in a number of mines in North America, Europe and South Africa, visited by the author in May and June, 1986. The study tour was sponsored by the Queensland Chamber of Mines through the Julius Kruttschnitt Travel Grant. The state of the art is described for metalliferous mines, ranging from labour intensive to mechanized operations.

Also described are a number of techniques, used in steeply dipping coal seams, which could perhaps be modified for use in metalliferous mines. In the mechanized methods seen in coal mines, various coal cutters and shearers were in use. For adaptation to hard rock mining, these machines would have to be replaced by some other rock breaking method until advances are made in hard rock cutting technology. Suitable methods, depending on the type of rock encountered, could be roadheader type machines, impact rippers, or drilling and blasting. The latter may require further modifications to the method for prevention of blast damage to equipment installed in the face.

The original intention of the author was to provide a detailed analysis of the labour, consumable stores, capital equipment and maintenance inputs for each method. As the tour progressed, it became obvious that most, if not all, of these inputs were dependent on conditions specific to each site, such as ground conditions, maintenance policies, and other "traditional" operating practices peculiar to the company, mining field or country. The author has therefore endeavoured to describe each mining method in sufficient detail to permit potential users to understand the principles of the method, and adapt it and specify the necessary inputs to meet the conditions of their particular sites.

Mining methods are described under three headings: "Conventional" Labour Intensive Methods; Mechanized Methods; and Steep Narrow Coal Longwall Techniques. Some items of equipment not currently in use in Australia are then described. Finally, some concluding remarks on ground support practices and a summary of the state of the art are presented.

The units in this report are generally those used by the company concerned, although some conversions have been made by the author for consistency within sections or to correspond more closely with Australian usage. The unit for productivity used throughout is tonnes per manshift (t/ms).

"Conventional" Labour Intensive Methods

Galena Unit, Asarco Inc., Idaho, USA

The Galena mine produces 4,000,000 oz. of silver from 200,000 short tons of ore per annum by cut and fill methods. The ore occurs as veins along fractures in quartzite. The veins are generally vertical, and range in width from a few inches to 20 feet, with an average of six feet. A sketch of a typical stoping area is given in Figure 1.

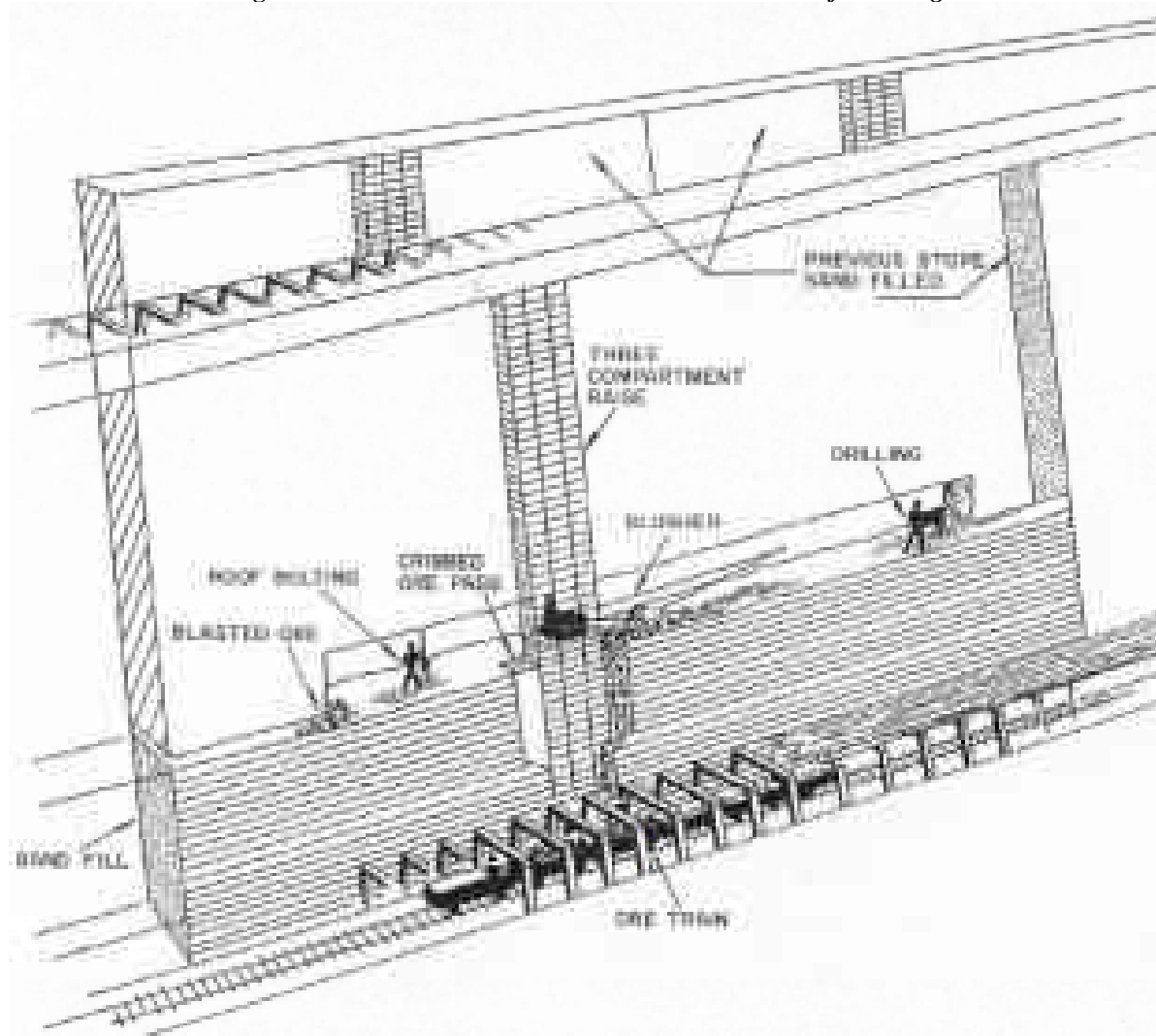
A stoping area is blocked out by driving a sill along the vein on levels 300 feet apart vertically. Drives are nominally 8'6" x 9'6". A three compartment timbered raise is driven between levels in the middle of a 100' stope block. Mined 8' x 19', the raise is timbered to give a central 62" x 66" manway with a 50" x 66" compartment on either side. Care is required in mining to ensure that there is room for timber caps to be cut square on the ends for optimal blocking and wedging. When a raise is breaking through to a timbered drive at the base of an earlier stoping area the drive timbers must be anchored to prevent collapse of timbering and supported stope fill into the new raise.

Raise mining is done by a crew of three men. Two carry out the mining at the face, and the third transports material on the level and operates a winch hauling a small supply skip in the raise. A three shift cycle is achieved for a 6' cut, using two shifts for drilling, blasting and rockbolting, and one shift for timbering. Generally one shift per day is worked on raising.

Stoping operations installing timber sets and sealing the timbering with planking and hessian. Onto this is poured the first layer of fill. Subsequent cut and fill operations advance away from either side of the central raise. Cuts are generally taken using horizontal handheld drilling. Advance is generally 8' per blast, and the lift height varies between 6' and 12'. Mining widths are generally between 4' and 10', with an average of 6', though it is possible to come as narrow as 3', the width of the scraper hoe. Stope headings are generally 50' - 100' long, though up to 300' may be mined on occasion. In long or twisted headings, an access raise and orepass may be built up in the fill remote from the central raise. Where an ore block does not extend through to the level above, or where a stope is urgently required for scheduling reasons, the whole stope may be mined from such blind accesses, but a predeveloped raise is preferred to provide through ventilation. Stope headings are ventilated by forcing fans in the central raise blowing air through 12" diameter ducting.

1. Principal Mining Consultant, AMC Consultants Pty Ltd,
8/135 Wickham Terrace, Brisbane QLD 4000
E-mail: bhall@amcconsultants.com.au

Figure 1 - ASARCO INC. - Galena unit narrow cut and fill mining



Mucking is by scraper with the winch mounted in the central raise timbering, or moved forward to a remote cribbed pass. For the first six sets up from the sill, the two outer compartments of the timbered raise are used as orepasses. Above this, the pass moves outside the raise and is cribbed up in the fill. This permits the scraper winch to be positioned outside the central ladderway/skipway compartment and provides a dead break in the pass a little above the chute. A 12" grizzly is installed at the top of each pass. After a heading is mined out, a 2-3 man fill crew takes 3-5 shifts for fill preparation and filling with mill tailings. Fill preparation involves cribbing the orepasses and remote access raises and passes, and sealing these with hessian for drainage and fill retention. Fill is generally to within 1'-2' of the back of the stope, leaving a gap for breaking the next lift.

Because of the depth of mining - in excess of 5,000' - rock stresses are a major factor to be considered in mine planning. An extensive seismic monitoring network is installed for detection of seismic activity and rockburst prediction. Where recordings indicate a build up of activity at a stope, an attempt is made to induce the coming rockburst at blasting time. This is achieved by leaving the stope unfilled after a lift, and drilling out the whole back of the stope with 8' upholes, which are blasted with millisecond delay detonators over the full stope length. The sudden increase in stress in the pillar above the stope will frequently induce a "controlled" rockburst at blasting

time, when no men are in the vicinity. This method is used particularly where a stope is approaching the sill horizon of the stope above, with the crown pillar becoming smaller and more highly stressed.

The 200,000 tons per annum is produced in 250 days by a total workforce of:

Days pay:	
Underground	161
Mill	12
Maintenance	37
Surface General	17
Salaried Staff	28
TOTAL	255 men

Durban Roodepoort Deep Ltd., Transvaal, South Africa

This is one of the few South African gold mines producing from steeply dipping reefs. Stope widths range from 1.0m to 3.0m, and dips from 50° to vertical. A number of methods are used, all of which are labour intensive. The author visited one operation, and obtained information on two other methods.

Wall Shrinkage Stoping (See Figure 2)

A stoping area is prepared by driving on reef and raising to the level above. Production starts by mining out 6m above the reef drive and installing timber packs and chutes (Figure 3). This area is maintained 20m ahead of the main stope face, both to protect workers in the drive from stope face activities, and to avoid blocking the entrance for the stope ventilation. The main stope face is mined overhand on a true dip equal to the angle of repose of the broken rock. The face panels are drilled with horizontal holes by airleg mounted drifters operated by a two man crew working on the broken rock slope which is maintained on the face.

The method is different from standard shrinkage methods in that the broken reef remaining in the stope for support is in panels supported on the bottom by timber packs and laggings, and on the sides by walls built of sorted, hand packed large pieces of reef (Figure 4). Lines on dip, known as "diplines", are established between the reef walls at 14.5m intervals along

strike. These are used as ore passes from the face to the reef drive. As the face advances, laggings and packs are installed to extend the base of the panel of reef, and excess reef is drawn off down the dielines, which are covered over at the top, and at dead breaks at strike accesses, for maintaining safe access to and along the face.

As the face advances, the "walls" of reef are reclaimed and replaced with waste walls (Figure 4). An "intertrack" is established between main levels to permit reef wall reclamation in the lower portion of the stope to commence sooner than would otherwise be possible. This prevents the pinching of broken reef by closure of the stope walls, and makes reclamation less labour intensive. Reef drawn from dielines above the intertrack is trammed by hand back to completed dielines through waste in the reclaimed area, where it gravitates to the chutes on the main level reef drives.

Figure 2 – Durban Roodepoort Deep Ltd - Wall shrinkage stope

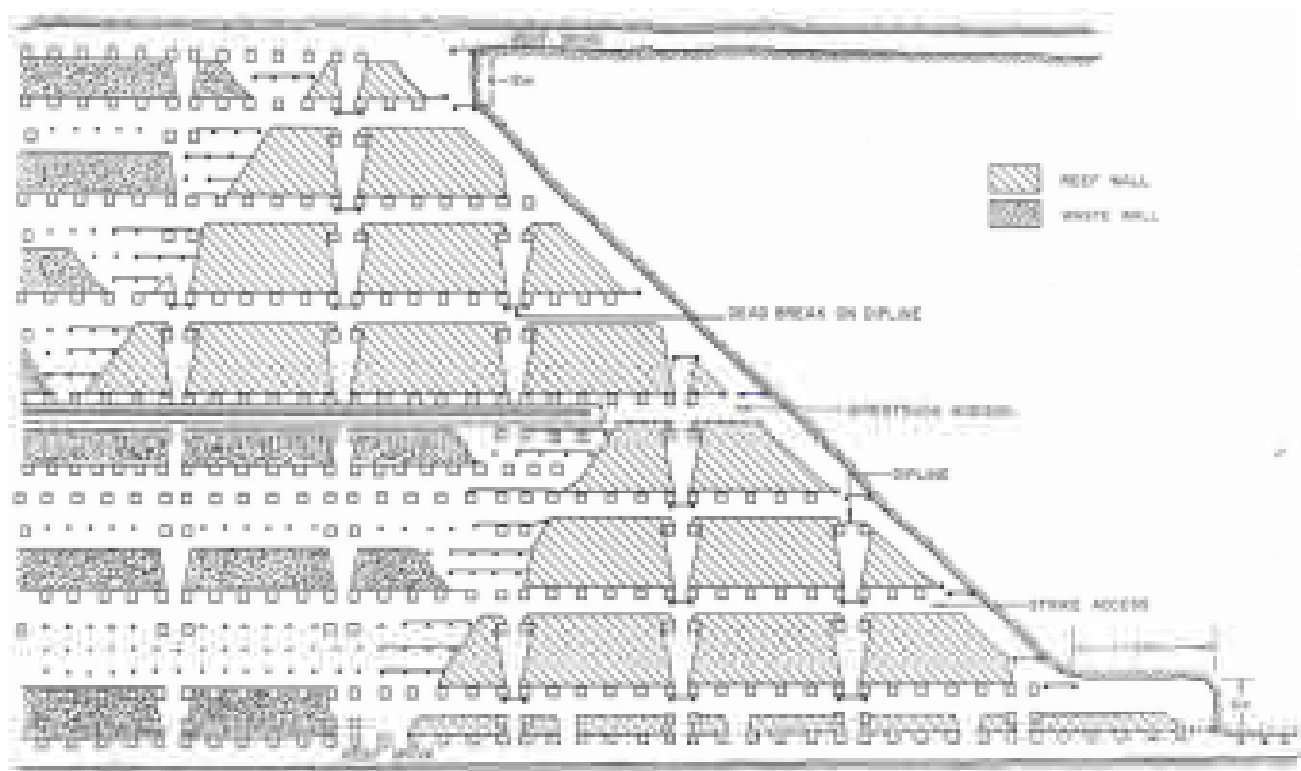


Figure 3 – *Durban Roodepoort Deep Ltd – Reef drive and timber chute*

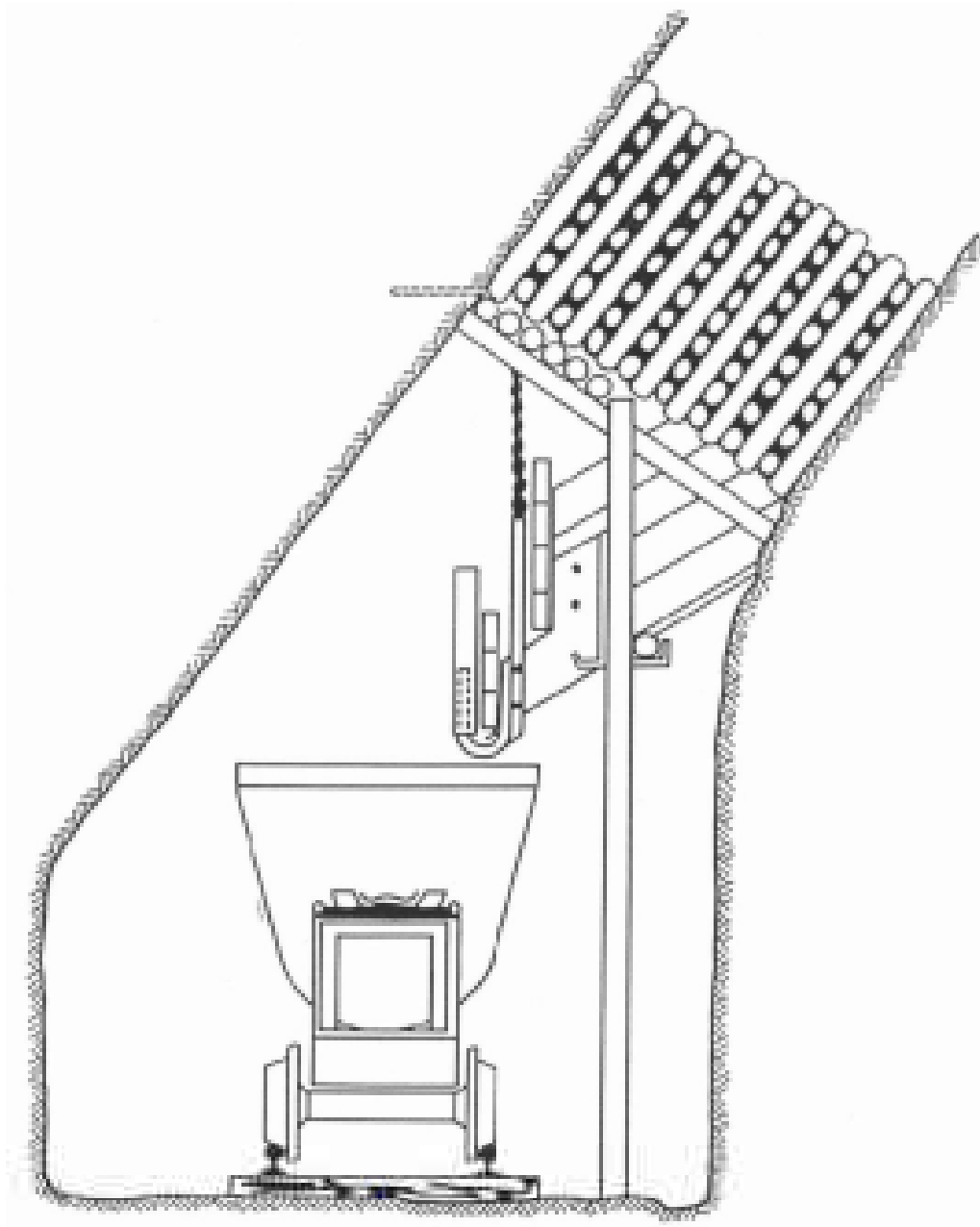
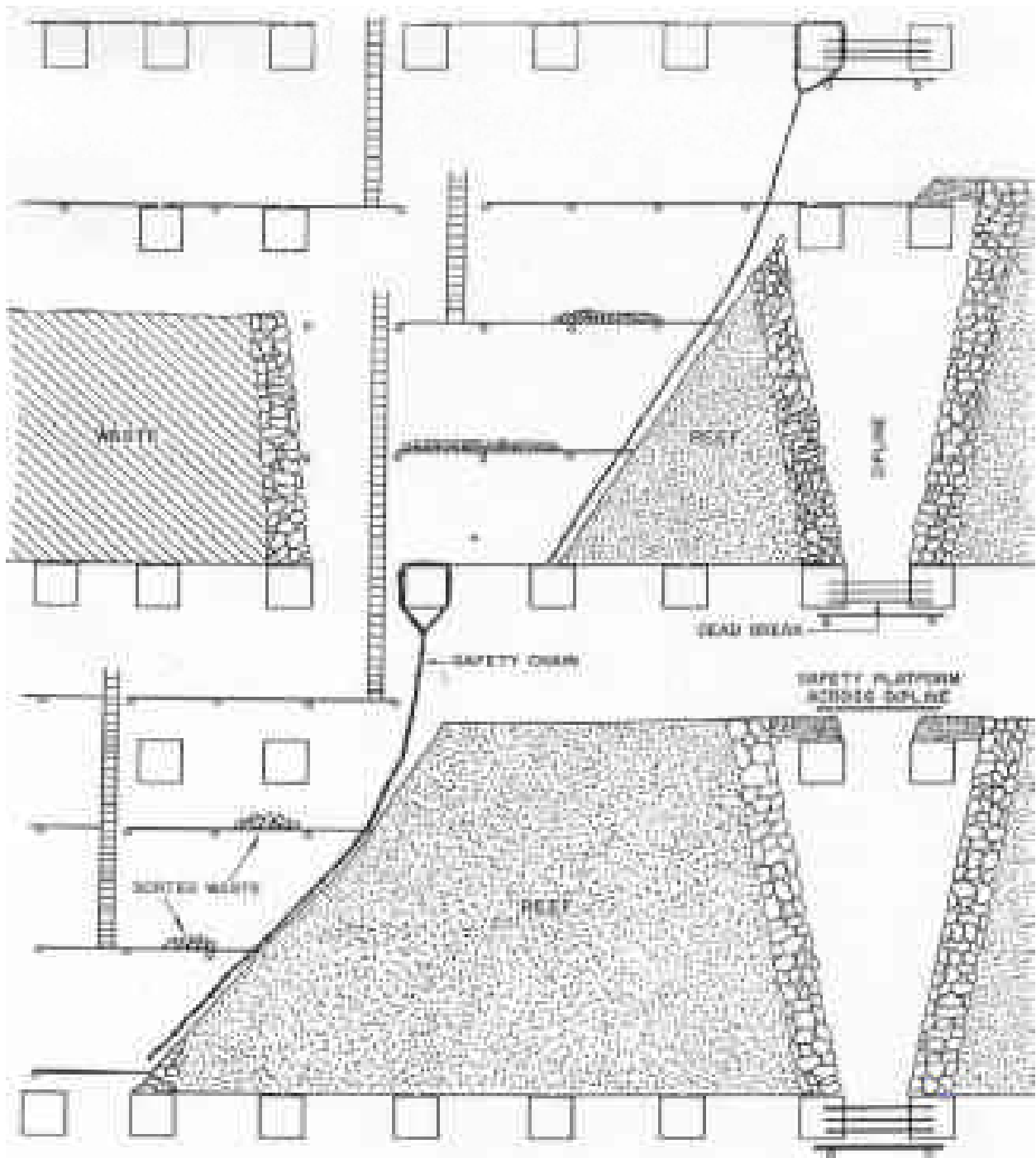


Figure 4 – Durban Roodepoort Deep Ltd – Reef wall reclamation

**Triangular (Rolling) Panel Stoping (See Figure 5)**

The reef drive, raise, face shape and intertrack development in this method are similar to those for walled shrinkage described above. The main differences relate to the amounts of timber support and broken rock tied up in the stope. For triangular panel stoping, one triangle of broken reef is maintained behind each advancing face panel. Reef is reclaimed from the back of each triangle to maintain its apex 2 m below the leading corner of the triangle above. Reef blasted on each face panel gravitates or is cleaned down the face side of its triangle, then down the back of the triangle below, from which it gravitates to chutes on the drive below. Reef drawn from chutes on the intertrack is

trammed forward towards the face and tipped behind the triangle immediately below the intertrack.

This method requires considerably more timber support in the back area than the wall shrinkage method. Until recent years, when pipesticks and other innovative systems have taken over many ground support applications from traditional timber packs in South African gold mines, the wall shrinkage method was preferred. With the advent of these cheaper and less labour intensive support methods, the triangular rolling panel method has become more popular.

The back of the stope is carried horizontal, and is drilled out using airleg mounted hand held drifters. Holes are 2.4m long at approximately 45° to the horizontal, giving an advance of about 1.8m. Reef is drawn out of the stope from the level below to maintain an adequate working height from the back to the broken rock floor. Extraction is by rail mounted compressed air rocker shovels loading into rail trucks through a series of drawpoints developed from a strike drive about 6m from the reef (Figure 7). The locations of the drawpoints are marked in the working area of the stope by paint lines extended up the walls as the face advances. Radio communication is maintained between the stope and haulage crews for withdrawing stope workers from above drawpoints to be mucked, and for controlling the amount of broken reef extracted.

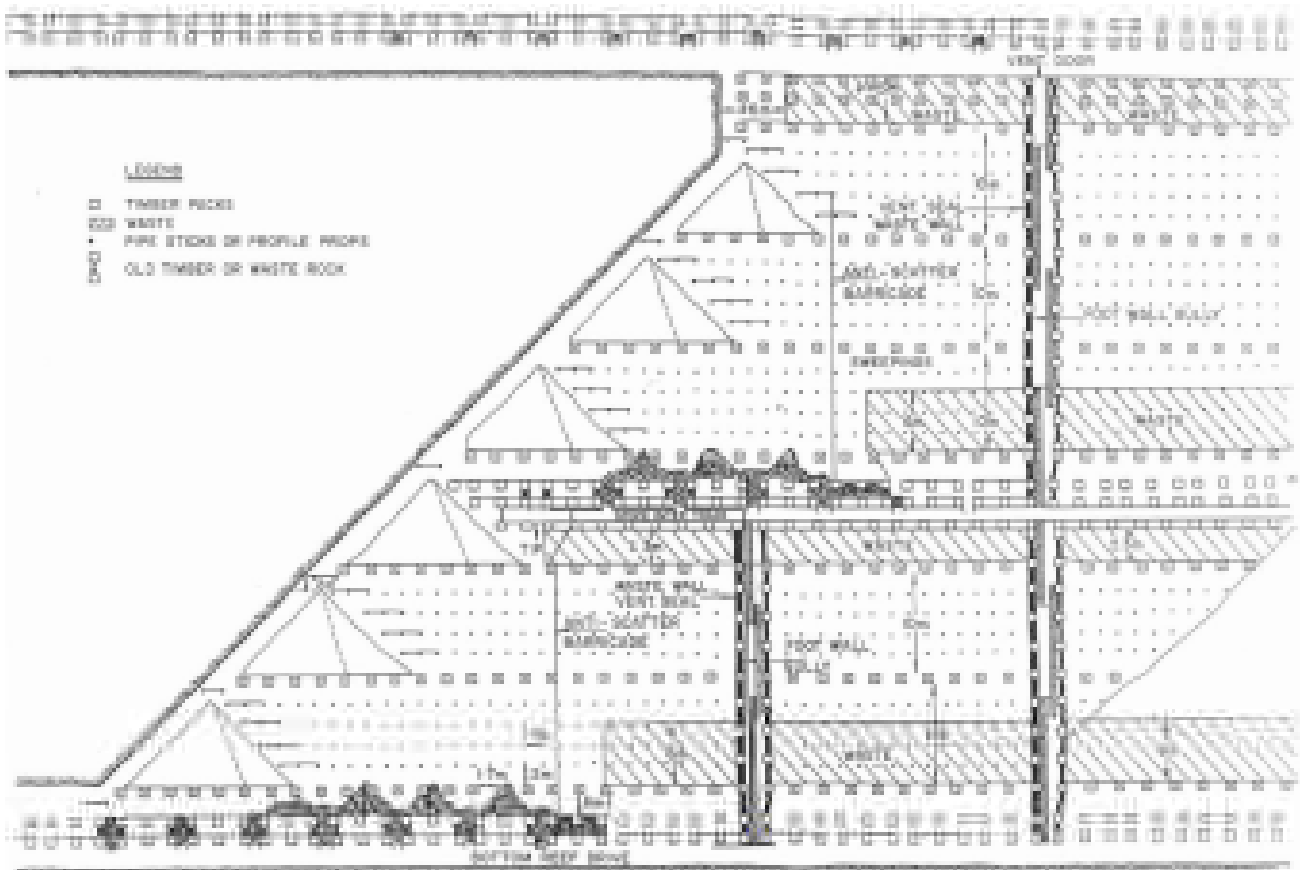


Figure 6 – Durban Roodepoort Deep Ltd – Shrinkage stoping
Typical longitudinal section
Scale 1:500 (approx)

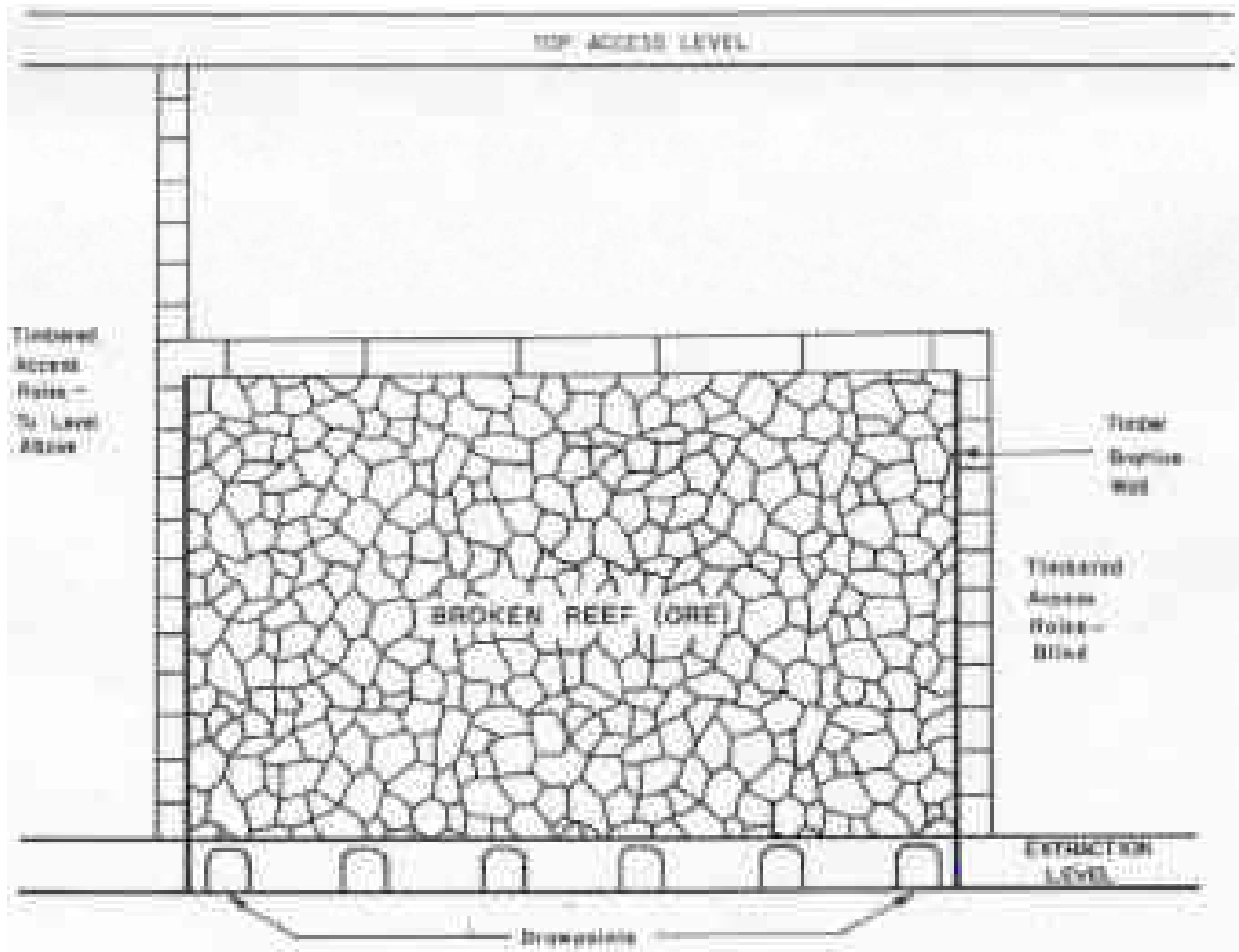
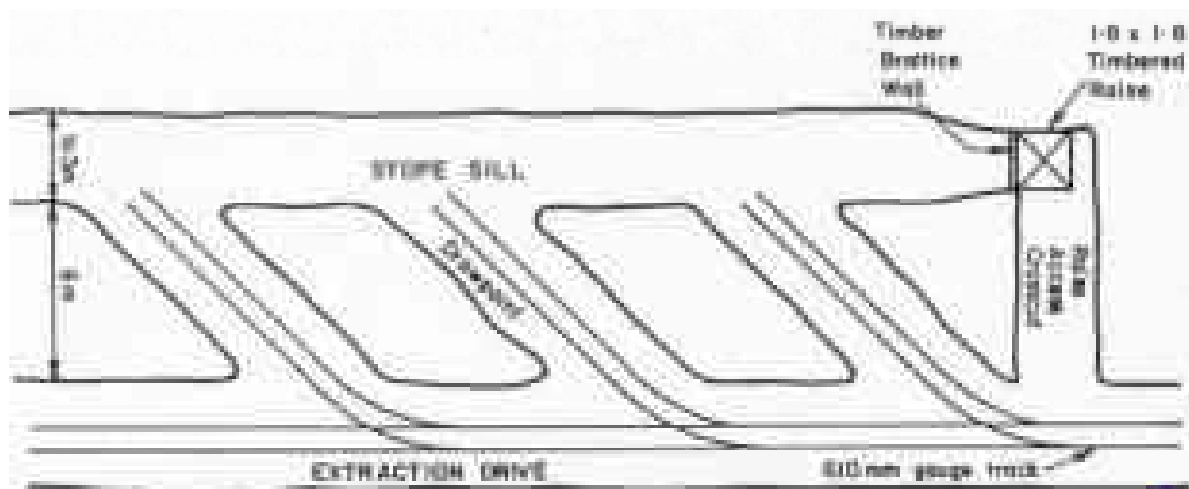


Figure 7 – Durban Roodepoort Deep Ltd – Shrinkage stoping
Typical extraction level plan
Scale 1:250 (approx)



Vouters Colliery, Houilleres du Bassin de Lorraine, Charbonnages de France, France

The Lorraine Basin mines of Charbonnages de France work several coal seams by a variety of methods in dips from horizontal to vertical. At the Vouters mine a number of closely spaced seams dipping from 60° to vertical are mined by cut and fill type methods. The leading seam is mined slowly by labour intensive methods to destress adjacent seams, which can then be mined quickly by mechanized methods. The manual method is described here, while mechanized operations are described in Section 3.1 below. Details which are common to both methods are described here. Figure 8 shows a generalized three dimensional view of a steep seam mining area.

In all cases, access is through a raise built up in the fill from a main access crosscut on the level below. The raise is constructed of steel rings, known as tubbing, divided into three compartments (see Figure 9). The lower, third is a pass for coal to gravitate to a conveyor belt in the crosscut below, while the

upper section is divided into a skipway for materials transport, and a ladderway in which the air, water, fill and electrical services are also installed. Ventilation enters through the tubbing, travels along each face, and exhausts at the extremities, generally through a return air raise to the level above, but sometimes back down through steel rings in the fill to the level below.

Broken coal is transported from the mining face to the central tubbing by a scraper conveyor on the floor of the stope. The head end drive assembly, powered by a 90kW motor, discharges above the pass compartment of the tubbing, and drives a single 26mm diameter chain down the centre of the conveyor. The conveyor is built up in 3m long sections, mounted on pontoons which float the conveyor to its new position during the filling operation. For a 3m length, the complete assembly weighs 1563kg, while the pontoons have a volume of 1680 litres.

Figure 8 – Vouters Colliery – Generalised view of a mining area

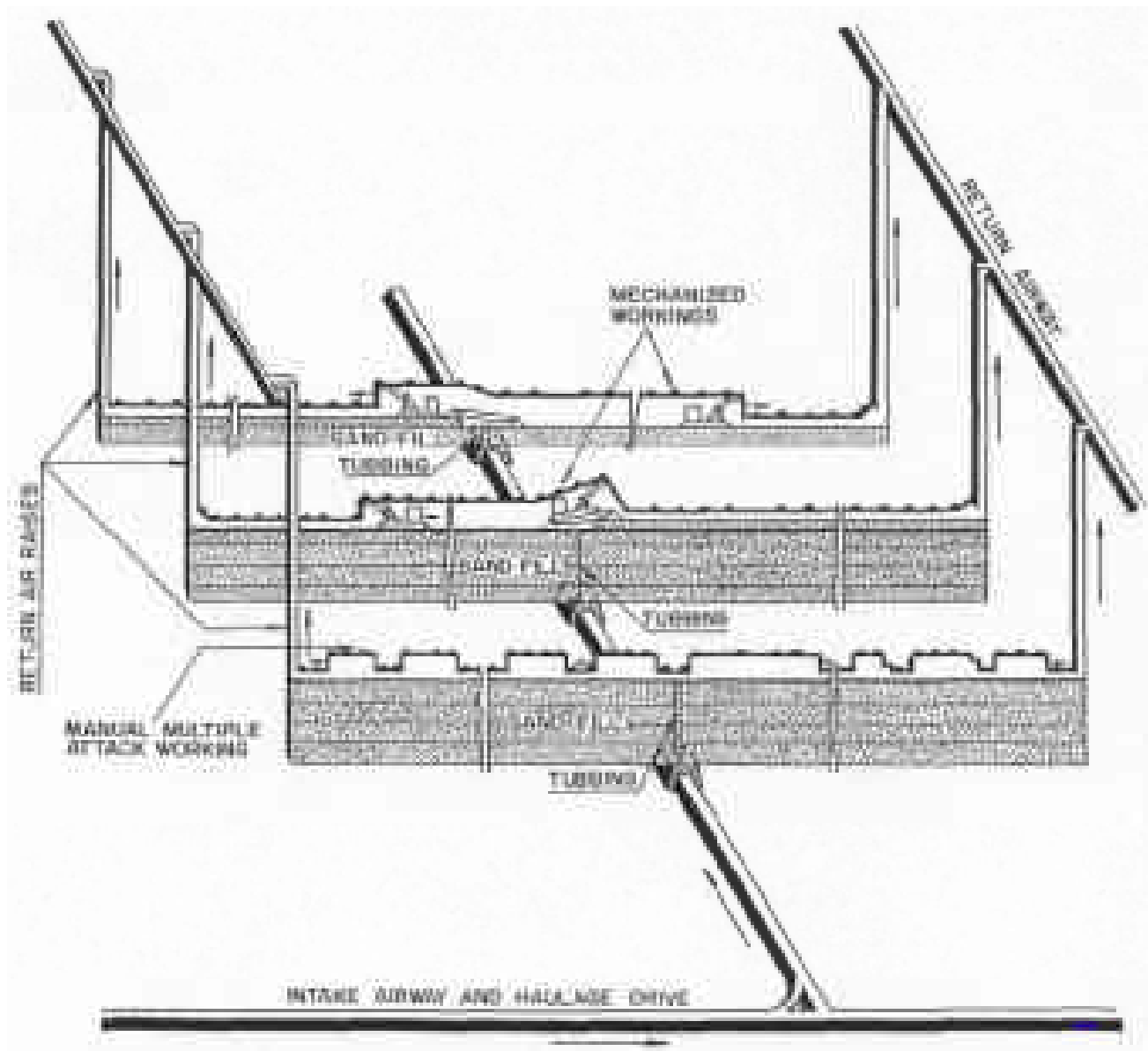
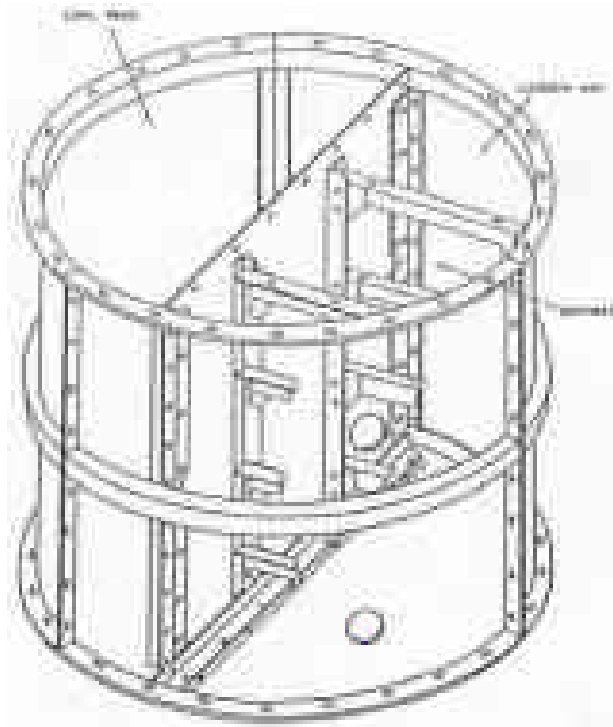


Figure 9 – Vouters Colliery – Central access tubbing

Fill is produced by mining poorly consolidated sandstone in a surface quarry. The backfill preparation plant is located in old near surface mine workings. Feed from silos between the plant and surface passes through hammer mills, and +80mm material and trash are scalped off. Undersize material enters mixing tanks where water is added to achieve the desired density of 60% solids by weight. These tanks are tapped directly by boreholes leading to the fill distribution level 545m below surface. Pulp density is able to be simply controlled by maintaining the levels in the mixing tanks, water addition being increased or reduced as appropriate to maintain a balance between the viscosity, the head loss in the reticulation system, and the head created in the tanks. No pumps are used. From the fill distribution level the fill is reticulated to the coal producing areas between 686m and 1,250m below surface.

In the stopes, fill retaining dams are built of timber and sealed with hessian, approximately 4m from the central access tubbing. This allows the tubbing to be carried below the level of the fill so that coal discharging from the conveyors can gravitate to the top of the tubbing pass compartment. The area between the dams and the tubbing is allowed to fill up with fill leakage, fine coal spillage, and general rubbish from the mining operations (See Figure 16).

The seams are blocked out into mining areas 300m - 800m long on strike, and 210m or 110m high, between main levels.

Manual mining takes place in smaller strike length blocks, by what is known as the "multiple attack" method (Figure 10). The stope is subdivided along strike into a number of "attacks", generally 30m long. The coal is drilled with horizontal holes and blasted down onto the conveyor. Timbering is installed as shown in Figure 11 for both hangingwall support and providing access to the top of the face. The back is supported with a reclaimable steel cap supported by the hangingwall timbers, which are butted onto timbers from the lift below to transmit vertical forces down onto previously installed support embedded in the fill. Faces are 3.5 - 5.0m high, 1.8 - 4.0m wide, and advance is 2.0 - 3.0m per blast.

Typical labour organization is:

- 2 men drilling and timbering per attack
- 2 men blasting in the area
- 1 conveyor operator
- 1 organiser

giving 10 men per shift for three attacks.

Four overlapping eight-hour shifts per day are worked, consisting of three shifts with six hours at the face, and the fourth shift for such operations as timber and materials transport and tubbing extensions. A typical cycle is 20 days mining, followed by 5-7 days fill preparation and filling. In the manual method, fill preparation involves, apart from dam building and similar activities, the complete dismantling of the conveyor, lifting, and reassembling it above the timbering. Overall productivity is 8-10 tonnes per manshift.

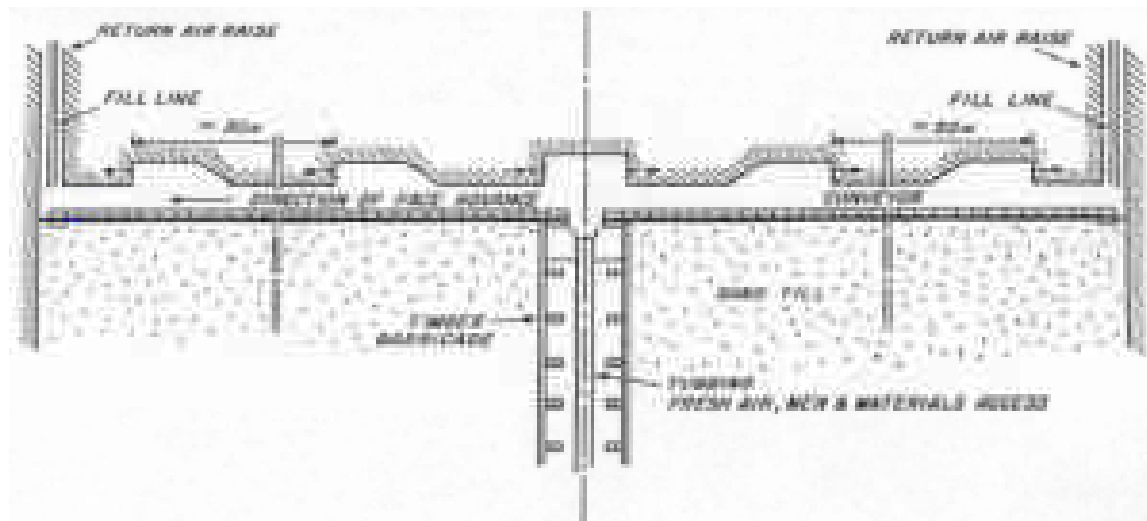
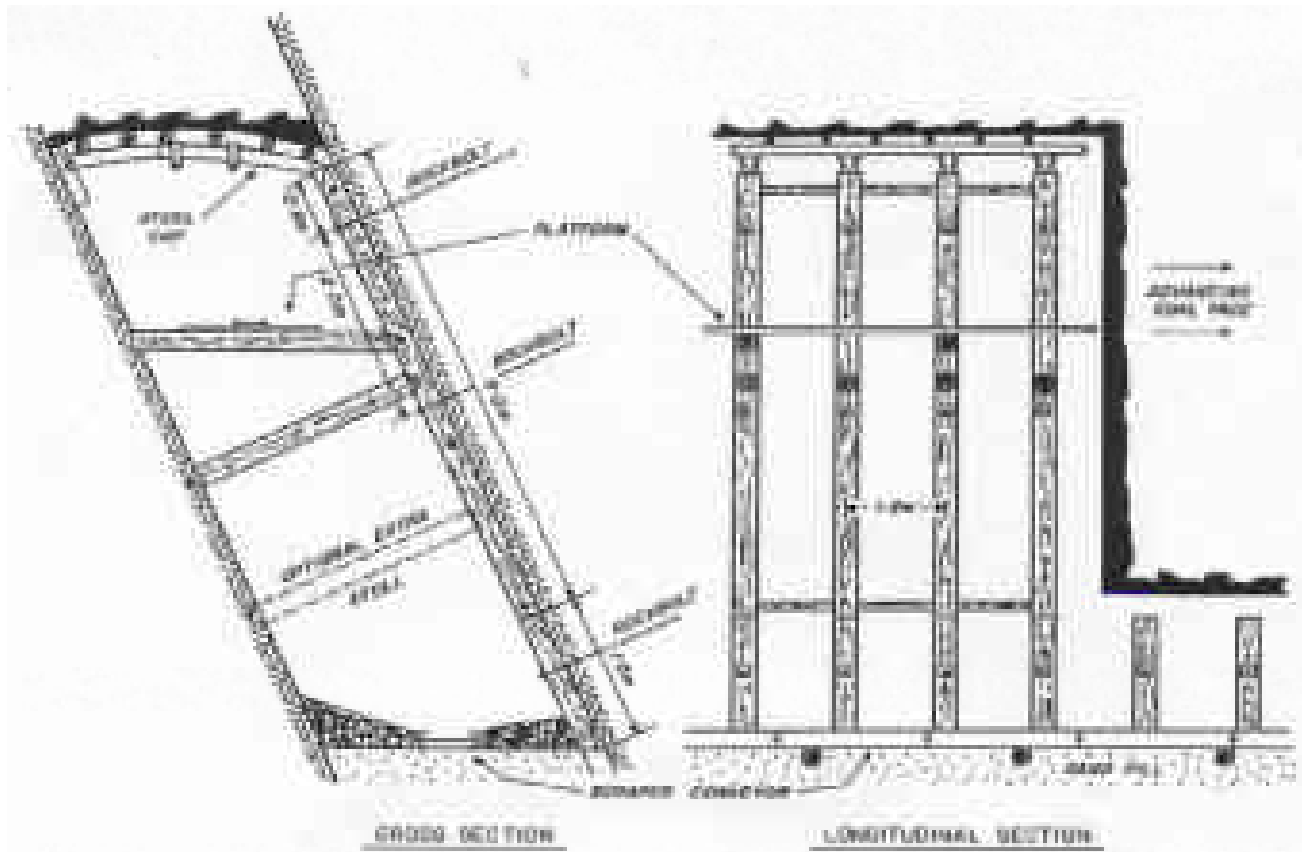
**Figure 10 – Vouters Colliery – Multiple attack method
Longitudinal section**

Figure 11 - Vouter Colliery – Multiple attack timbering



Fondon Colliery, Empresa Nacional Hulleras del Norte S.A. (HUNOSA) Asturias, Spain

The national Spanish coal mining company operates a number of pits in the Central Asturian Basin, mining coal seams in dips from 55° to vertical, at widths ranging from 0.45m to 2.0m. The narrowest areas are mined by timbered stopes, which are the most labour intensive of the operations. Although the actual cutting of coal is by shearers, this method is described in this section because of the high component of manual work in the mining cycle. Other mechanized methods are described in Section 4 below.

Timbered stopes are used to mine coal seams as narrow as 0.35m, though the minimum mining width is 0.45m to permit adequate access. Mining takes place between main levels 85m apart, and faces are mined underhand, generally on a 60° plunge, giving a face length of 100m (Figure 12).

The bottom 5m of the face is mined ahead of the main face, and timber supports and chutes are installed for loading broken coal into 3m³ rail trucks. Access is not normally available to the face from below.

Because of the steepness of the stope, and the danger of falling materials, the number of concurrent operations permissible in the stope is severely restricted. To overcome this problem, the working day is broken into four shifts, with each shift carrying out a separate operation, viz:

- coal cutting
- timber transport into the stope

- timber installation
- filling

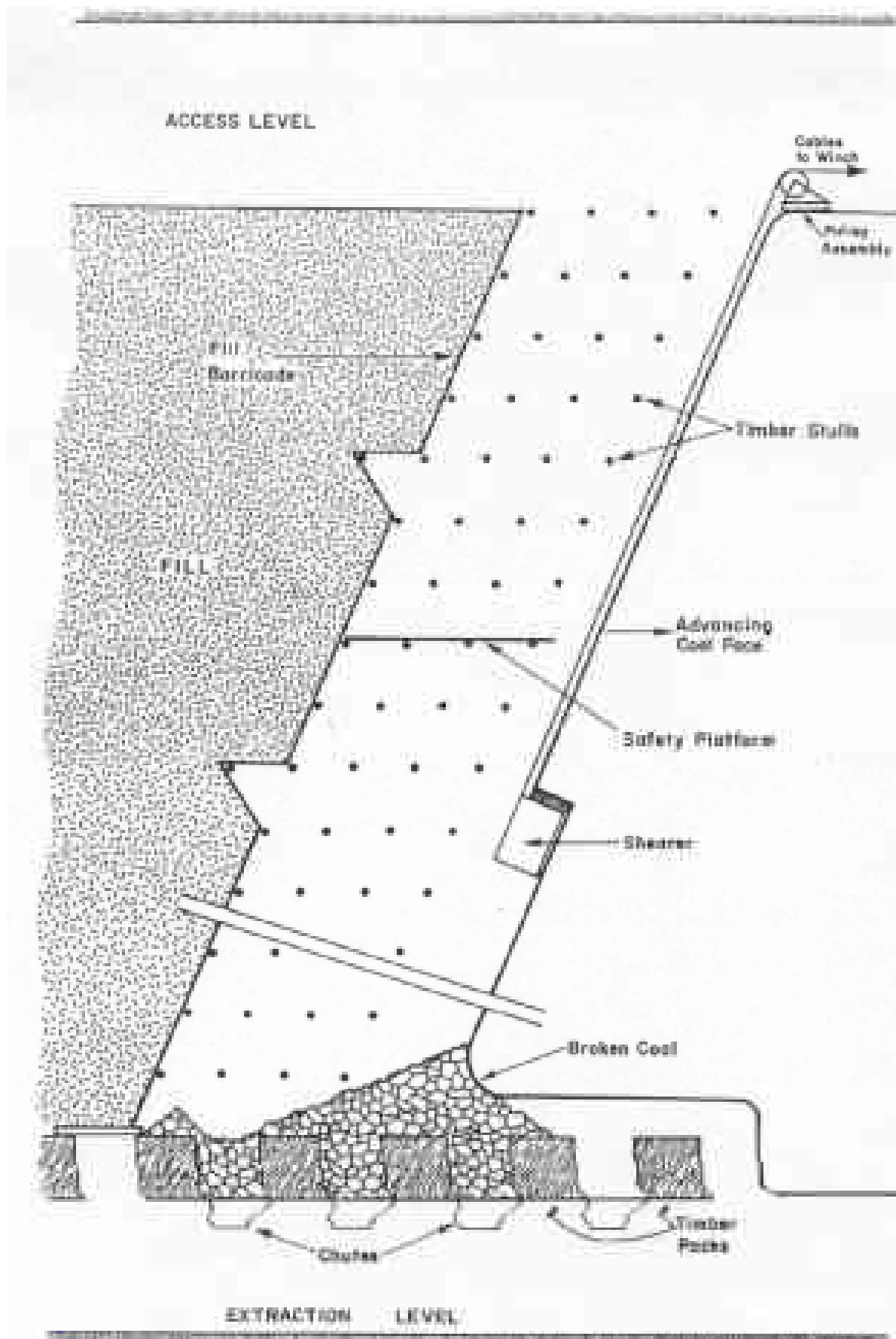
The distribution of labour between the various shifts depends on such factors as the stope width and ground conditions.

Coal cutting is by shearer, which is hauled up the face by cables from a winch on the top access level. Two types of shearer are used:

1. an HUNOSA developed machine, designated H1, cutting to 0.8m and
2. a Russian Poisk compressed air operated unit, cutting to 0.45m.

Timber is lowered into the stope by small compressed air winches, and is stored on existing timbering at a number of elevations within the stope. Stulls of 75-100mm diameter are placed between hangingwall and footwall in lines parallel to the face, with caps and sills of half round laggings, approximately 3m long, over three or four stulls (Figures 12 and 13). The spacing of both rows, and stulls within a row, is 1.0 -1.2m. Stulls are sawn to size on site, and the ends of the stulls and notches in the laggings are shaped with hand axes. Laggings may be placed horizontally across several rows for working and safety platforms, although most of the work in, and travelling through, the stope is done on the stulls. The narrowness of the stope and the density of the stulls minimises the risk of men falling in the stope.

Figure 12 – Fondon Colliery, Hunosa – Manual steep seam method – Long section



As the face advances, a fill fence is built on the fourth row of stalls back from the face at four row intervals (Figure 13). The fence consists of 10mm square wire mesh, supported against the stope walls by laggings. The stope is filled with a mixture of development waste and washery rejects, which is transported to the worked out area in sidetipping rail trucks, which permit direct tipping of the fill into the void (Figure 14). This method of filling only gives trouble if water inadvertently enters the fill

material, or if overbreak of the stope walls gives rise to irregularities, which are difficult to seal.

On average, 30 men per day work in a timbered face to produce 145t of coal, an average of 4.8 t/ms. For this, 75t of fill is required in each stope each day. Transport of fill is not included in these productivity figures.

Figure 13 – Fondon Colliery, Hunosa – Section through manual timber stope looking down the face

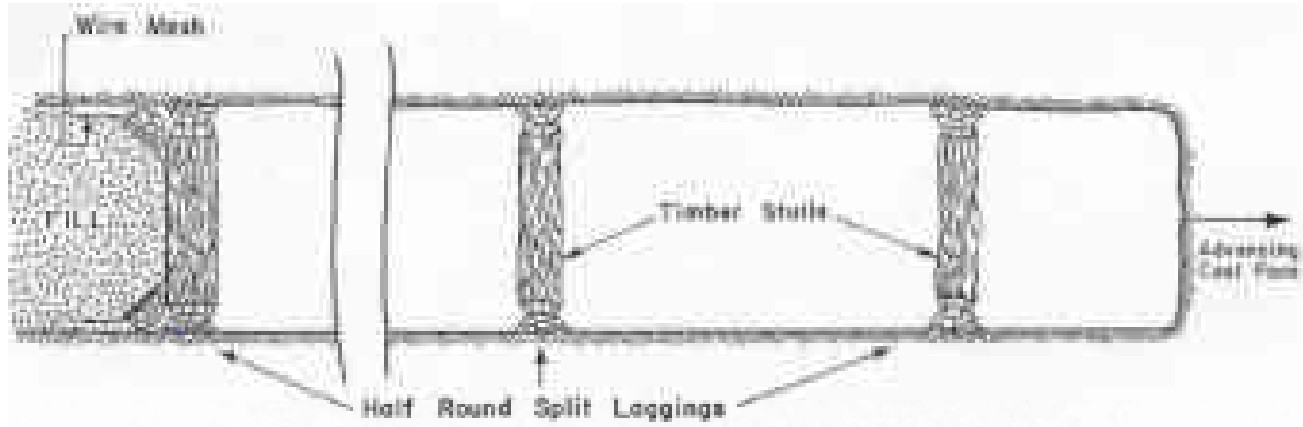
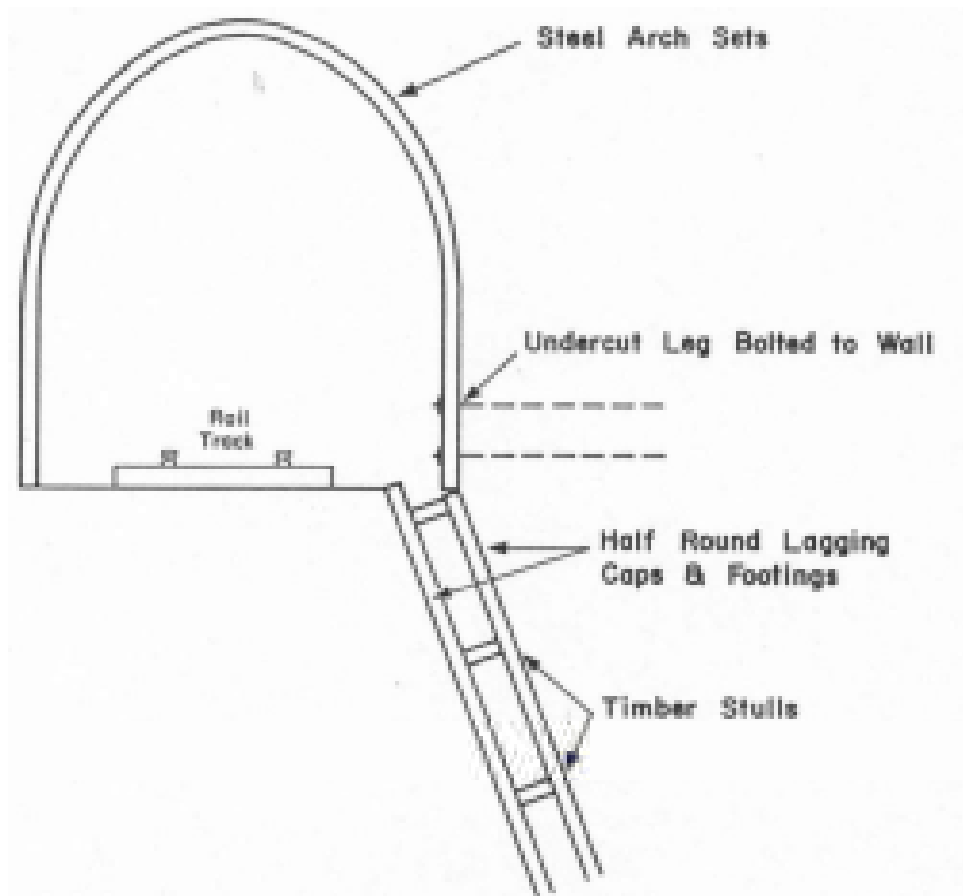


Figure 14 - Fondon Colliery, Hunosa – Cross section of top access to timber stope



Mechanized Methods

Vouters Colliery, Houilleres du Bassin de Lorraine Charbonnages de France, France

The general methods of working mechanised seams at Vouters are similar to those described for manual methods at this mine. The main difference is in the method of coal winning, which is by coal cutter in the mechanized areas. Two coal cutting machines are used in each stope, with the method of working being as shown in Figure 15.

The workings on either side of the tubing are phased such that the machine from one side can be retreated to the other side to permit fill preparation and filling on its side, without being delayed by, or interfering with, the other machine. A bridge is constructed across the tubing, and the fill level is staggered by half the lift height on opposite sides of the tubing. This is shown in Figure 16, which also shows the general arrangement of the tubing and access crosscut.

Three types of cutter are used, details of which are tabulated in the table 1.

Availabilities of 90% are quoted for the machines during the mining phase, when only breakdown maintenance and "patch up" repairs are done. Preventative maintenance and more permanent repairs are done while the machine is withdrawn for filling, and on weekends.

Ground support is similar to that for manual stopes. The coal cutters are equipped with arms for moving the steel caps from the brow at the bottom of the face to their new positions at the back of the face being mined. Stulls across the stope are not installed, so that the retreat of the machine is not impeded,

though occasionally, some may be placed to warn of stope wall convergence. These are removed before the machine is retreated at the end of the lift.

The absence of timbering across the stope means that the scraper conveyor does not have to be dismantled before filling. It is merely loosened in the existing fill, and floats up as an entity as the new fill is placed. Whereas in the manual method the conveyor is located so as to minimize the amount of broken coal which does not gravitate onto it, in the mechanized stopes it is located towards the footwall to permit access for a sidetipping Eimco 630 loader, which cleans up along the hanging wall side (Figure 17).

Table 1

Manufacturer	Ateliers du Nord de la France	Sagem
Type	ANF-D10	ANF-D11
Length	13.0m	13.0m
Mass	46-60t	46-60t
Installed Power		
-cutting	100kW	100kW
-movements	36kW	36kW
Drum diameter/width	700/1800mm	700/1800mm
Range of dips	55° -90°	55° -90°
Mining widths		
-90° dip	2.45-4.90m	2.60-5.10m
-65° dip	2.90-5.40m	3.10-5.10m
Face height	3.0-5.5m	3.0-5.5m

Figure 15 – Vouters Colliery – Phases of mechanized mining

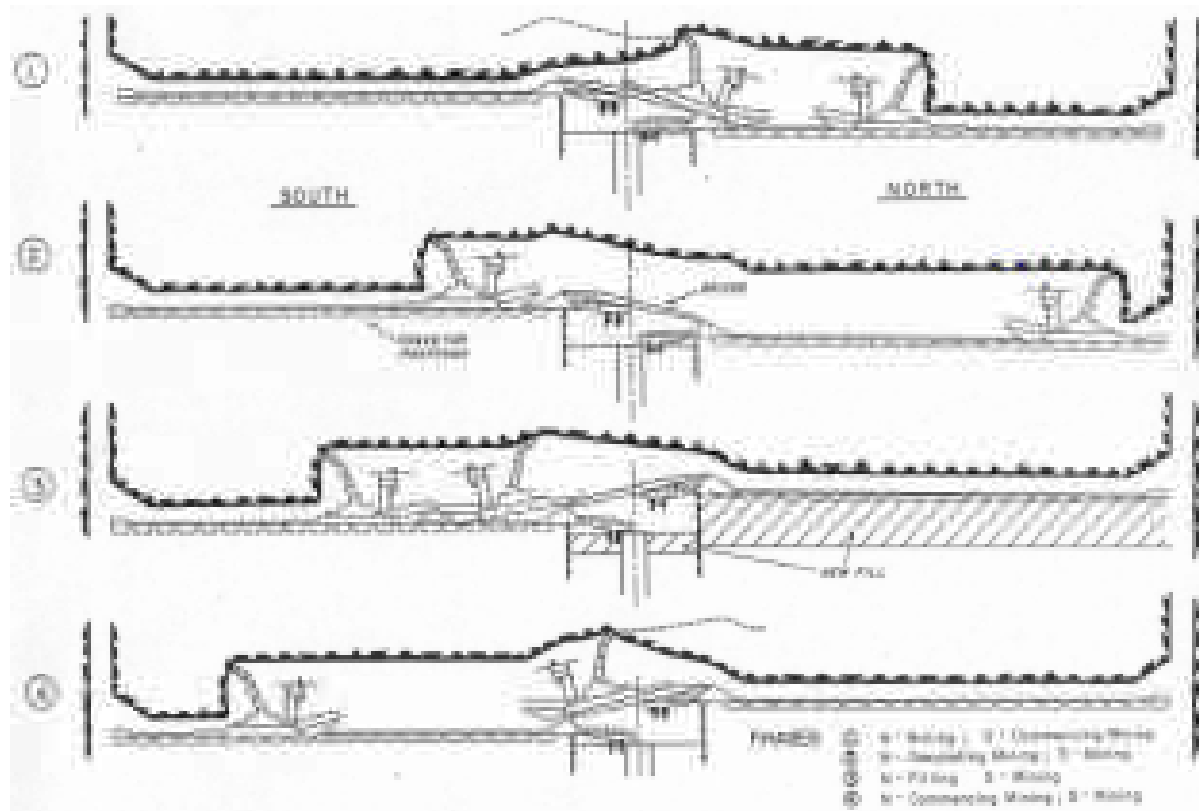


Figure 16 – Vouters Colliery – Stope access details

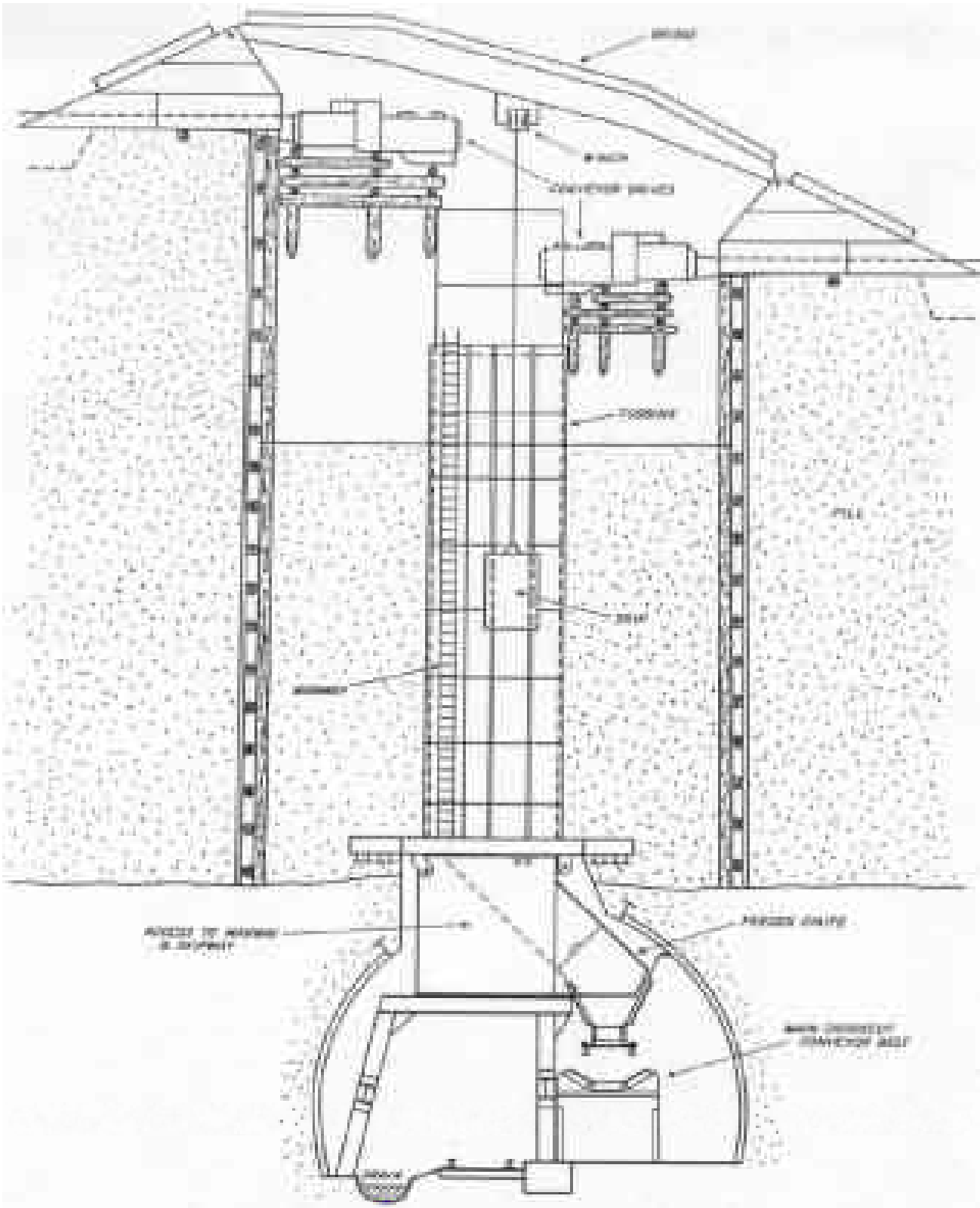
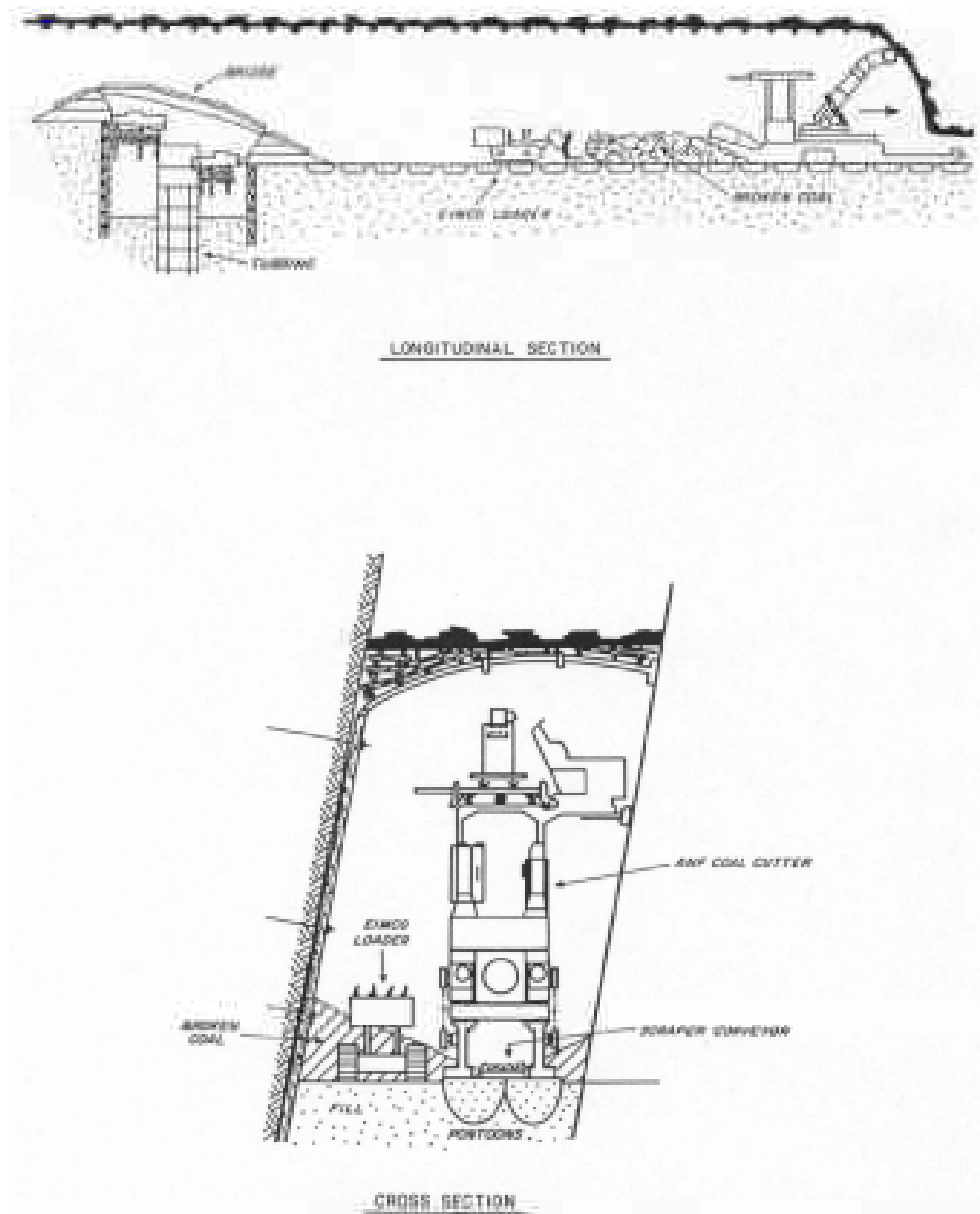


Figure 17 – Vouters Colliery – Arrangement of equipment in mechanized face



The typical labour complement for a mechanized face at Vouters is:

2 machine operators
1 man for ground support and clean up with Eimco 630
1 conveyor operator
1 organiser

An additional 3-4 men circulate around a number of faces for fill preparation and stowing. Four shifts are worked, as in the manual method. A typical mining cycle is 25-30 days mining, followed by five days of fill preparation and filling. Productivities (including the fill crew) range from 10-40 t/ms, with an average of 17-18 t/ms.

Le Fraisse Mine, La Crouzille Division, Cogema, Limoges, France

The La Crouzille division of the Compagnie generale des matieres nucleaires (Cogema), located 30km north of Limoges in western France, produces over half of the company's annual production of some 2500t of uranium. The Le Fraisse mine produces approximately 100t of uranium annually from narrow steeply dipping deposits associated with lamprophyre dykes and brecciated fault fractures in a granitic host rock. Veins are 100-200m long on strike, up to 300m deep, at dips from 70° to vertical, and from several centimetres to 20m wide. Ground conditions in the ore veins generally limit excavations to 3m wide. Mechanized methods have almost entirely replaced manual methods. Older sections of the mine are serviced by a shaft with rail haulage levels, while the newer parts have trackless decline access from surface. Several variations of cut and fill and undercut and fill methods are used. Details of the mining method in any stope depend on the nature of the vein and its location in the mine. Manual cut and fill and undercut and fill have been documented by Chadwick (1982) and are not described in this report. The total Le Fraisse operation produces 60,000 tpa of ore with a workforce of 85 men. Three quarters of these are mining staff, and the remainder are maintenance crews.

Cut and Fill.

Where ground conditions permit, ascending cut and fill is used. Stope blocks are 70-80m long on strike, and 40-45m high - the level interval in the shaft section of the mine. Stope widths range from 1.5 to 3.0m, and average 2.0m. Ore density is 2.3-2.5 t/m³. Crosscuts are driven through the vein in the centre of the block on both top and bottom horizons, and connected by a 1.8m diameter service raise 6-8m from the footwall of the orebody. Silting takes place on the bottom rail level, and the crosscut is re-established with sets before pouring the first lifts of fill. An ore chute and an entrance to a manway are also provided here, as both the stope, ore pass and emergency access are built up as steel rings in the fill.

The sequence of mining is shown in Figure 18. Two faces are advanced away from the service raise. These are mined with handheld drifters, drilling horizontal holes. Advance is 1.6m, and the lift height is 2.5m. The height from fill floor to back is 3.0m. Granitic sidewalk are generally fairly competent, and shell anchor tensioned rock bolts are installed as required. Split sets are generally used to support the weaker ore material in the back. Blasted ore is mucked with an electric LHD (L'equipement Minier Franceloader CT 1700 HE or CT 500 HE), which is captive in the stope, and parked in the service raise access when not in use.

The service raise contains a ladderway, separated from the rest of the raise by hoops connected by longitudinal steel straps, a skipway for materials transport, and all service cables and pipes including vent duct, which carries air from forcing fans on the level above to the mining faces.

When mining of the faces is complete, a half height fill pour is placed in one face. The LHD is parked on this fill while a slot for the next lift is cut in the back at the service raise access, and the access itself stripped for the next lift. Mullock from this stripping is left on the floor, services are reconnected, the loader is driven up onto the new level at the service raise, and, after extending the orepass and manway rings and fill drainage system, the filling of both sides of the stope is completed. Fill is repulped deslimed mill tailings from the central processing plant for the La Crouzille division at Bessines, 12 km from Le Fraisse.

Undercut and Fill

Over the last decade, the undercut and fill method has been replacing cut and fill, and now is used for 80% of the production at La Crouzille. This method is a descending cut and fill working under a reinforced concrete slab cast on the floor of the previous lift. This slab is an expensive item in the system. However, experience has shown that the improved productivities obtained by working under a solid concrete back rather than weak ore, and a consequent increase in the lift height to 4.0m, have made undercut and fill the cheaper method overall in weaker orebodies. As in cut and fill, handheld drilling and electric LHD mucking are employed. Because the LHD is operating on a floor of weak ore, it tends to churn the floor to mud. The largest possible LHD is therefore chosen for each stope to reduce the amount of tramming. Generally, Franceloader CT 1700 HE's are used, and CT 500 HE's find application only in the narrowest stopes.

The concrete slab is a reinforced concrete floor designed to civil engineering standards. The operating staff are provided with tables which specify, for a range of stope widths and surcharge loadings:

- concrete type and strength
- slab thickness
- number and size of shear rebars per metre of stope length for shear support of the weight of the slab
- number and size of tension bearing rebars per metre of stope length, for carrying tensile stresses in the lower section of the slab
- number of layers of reinforcing wire mesh.

Surcharge loading is in the form of development mullock stowed on the slabs. Shear bars are 2.0m long and grouted 1.5m deep into the sidewalk. Reinforcing mesh is supported on these, and the tensile bars cut to the stope width are placed on the mesh. Premixed concrete is pumped from surface directly to the slab being cast underground. Circular holes are left in the slab at intervals for ventilation.

In the trackless area of the mine, each cut is accessed by a ramp from the main access decline. Stope crews here are also responsible for trucking their production to surface in 7.8t trucks. In the shaft area, various combinations of "ramp-in-stope" and captive equipment methods are used.

Figure 18 – Le Fraisse Mine – Cut and fill

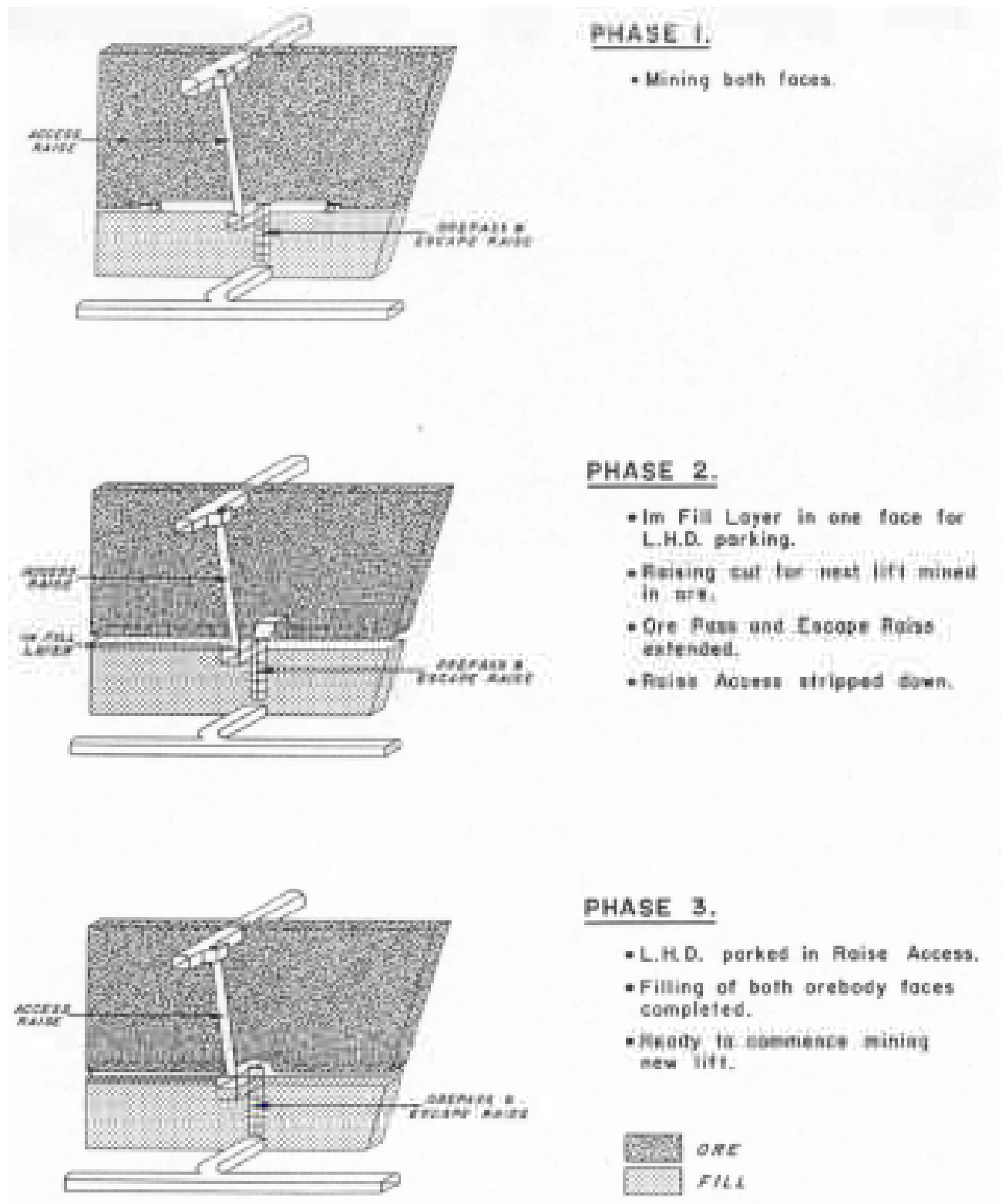
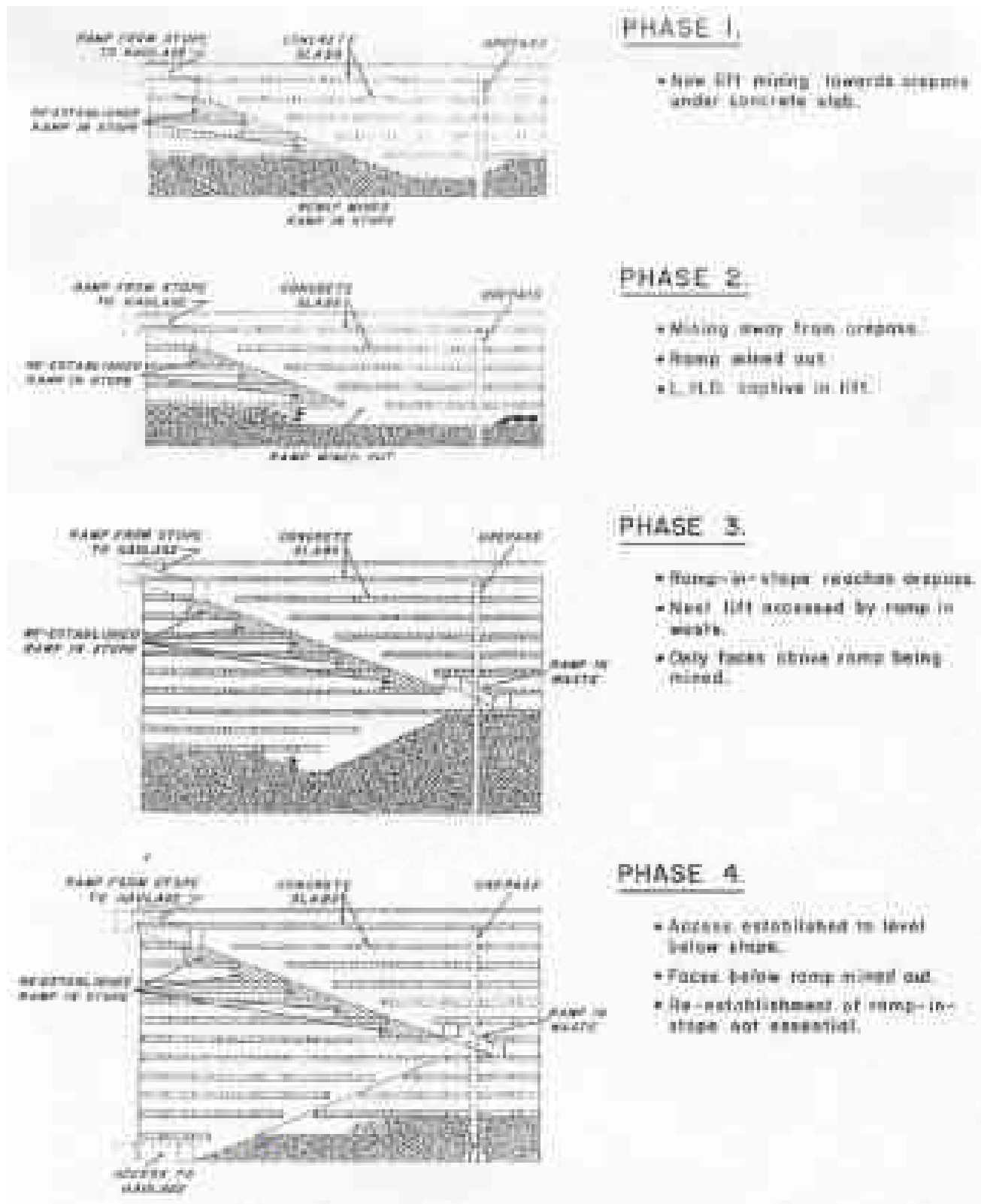


Figure 19 – Le Fraisse Mine – Ramp-in-stope undercut and fill



“Ramp-in-Stope” Undercut and Fill (Figure 19)

Access to the top of the stope is at one end of the block, and towards the other end, an orepass is established. The block of ore is approximately 100m long. A descending ramp at a gradient of 1:5 is established, and mining progresses on the new horizon under the concrete slab from the ramp towards the orepass. Broken ore is mucked from the face back up the ramp in the stope to the level above, where it is directly tipped into granby trucks. When the stope limit in this direction is reached, the LHD is turned around, the portion of the ramp within the lift is mined out, and the face is advanced to the opposite end of the stope. At this stage the loader is captive in the lift, and broken ore is mucked back to the orepass to the level below. When mining of the lift is complete, a new concrete slab is cast, with a gap left for the mining of the ramp down to the next lift. The gap is sized such that the vertical distance from the end of the slab to the descending ramp below is 2.5m. The ramp to the level above is re-established by placing development mullock and/or fill on the relevant part of the slab.

This process continues until the stope is half mined out, at which stage the ramp will have reached the opposite end of the stope. From here, a 20m turning loop is mined in mullock, declining to the next lift. Production now continues only in faces advancing away from the ramp and orepass. Access is available up to the top level, but broken ore is only trammed to the orepass. Once all lifts above this section of ramp have been mined out, and access to the bottom level has been obtained, the remaining lifts below the ramp are mined out, with ore being trammed back down to the level below. At this stage it is not essential to re-establish the ramp to the level above, as access is available from below.

Undercut and Fill with Captive Equipment (Figure 20)

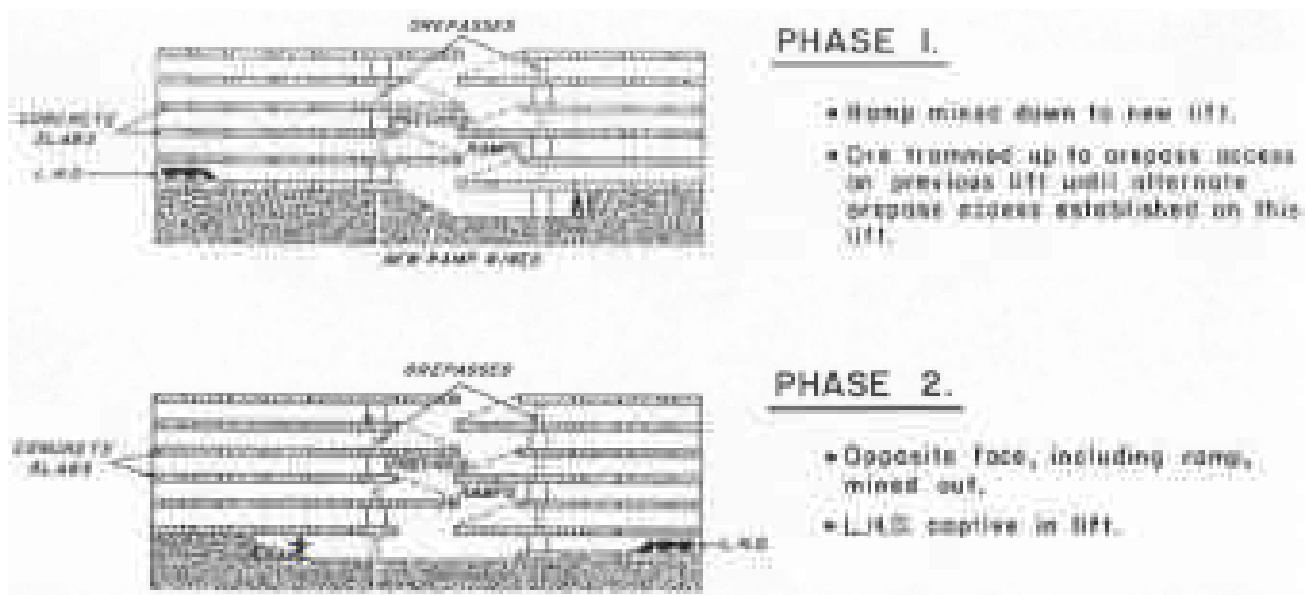
Two orepasses are raisebored 8m to the footwall of the stope. A new lift is accessed by ramping down in ore from one orepass access crosscut in the direction of the other orepass. When the new level is established the face continues mining in that

direction until the end of the stope has been reached and an access has been mined to the second orepass. The direction of mining is reversed, the ramp is mined out, and the face is advanced to the other end of the stope block. After the slab has been cast the procedure is repeated. The slabs overlap at the ramp position so that nothing can fall from the top of the stope to the working area. The minimum distance between orepasses is therefore a function of the amount of overlap required, the lift height (4m), the gradient of the ramp (1:5), and the clearance from the edge of the slab to the ramp below (2.5m). Two accesses for men and materials are maintained through the holes in the slabs, one on each side of the stope between the orepass and the stope limit.

The two undercut and fill methods described can be combined in a variety of ways depending on the ratio of stope height to stope length. Where the stope length is reduced such that the vertical height cannot be covered by two stope length ramps as in the ramp-in-stope method, the top section can be mined by ramp-in-stope, the middle section by the captive method, and, when the lower level can be reached by a stope length ramp, by reverting to the ramp-in-stope method. Alternatively, if two orepasses are placed towards the ends of the block, the middle section of the stope could be worked similarly to the ramp-in-stope lower section method, but with the LHD captive on a ramp between the two orepasses, or the whole block could be mined by the ramp-in-stope method.

Stope labour for all methods consists of two men on each of two shifts for a five day week and a third man on one shift for rail haulage of ore from the stope to the skip loading pocket in the shaft area of the mine. The overall cycle for one lift in a stope takes approximately one month. The stope labour productivity ranges from 7-25 t/ms and averages 15 t/ms. The overall productivity for the mechanized methods workforce is 12 t/ms. These figures are quoted for both cut and fill and undercut and fill, as the latter, as noted above, becomes more productive than the former as ground conditions deteriorate.

Figure 20 - Le Fraisse Mine - Undercut and Fill with Captive Equipment



Cluff Lake, Mine, Cluff Mining, Saskatchewan, Canada

Cluff Mining, owned 80% by Amok Ltd and 20% by the Saskatchewan Mining Development Corporation, is based in Saskatoon, and mines several uranium orebodies at Cluff Lake, which is 720km NNW of Saskatoon, 80km south of Lake Athabasca, and 30km east of the provincial border with Alberta. In Phase 1 of the operation, 4,000t of uranium were recovered from an open pit from ore averaging 6% U, and running as high as 40% U. Phase 2 is presently recovering 16,000t of uranium from several open pit and underground operations at an average head grade of 0.5% U (5 kg/tonne). The mill processes 800t of ore per day, to recover annually 135,000 kg of uranium.

The mine operates on a flyin-flyout basis. Mining crews, from Mine Superintendent level down, work seven days on - seven days off. Different levels within the hierarchy change over on different days to maintain continuity in the operation. The potential problem of having two Superintendents in charge of the mining operation has been overcome by having one primarily responsible for open cutting and the other primarily responsible for underground. While each is totally responsible for all the day-to-day operations while he is on site, major decisions relating to one of the components are made by the appropriate man.

The orebodies being mined are veins associated with near vertical fractures in sandstone. Ground conditions in the ore limit the width of openings to 2.5m. Where an ore body is wider than this, a room and pillar or drift and fill type method is employed. In narrow veins, an undercut and fill method is used. The present underground operation is producing 150t of ore per day at 0.7% U. Individual stopes run as high as 1% U.

An undercut and fill stope is accessed by a short ramp off a main access decline outside the orebody. The floor of the ramp is stripped down as successive lifts are mined in the orebody. Orepasses are established adjacent to these stope access ramps. Ore gravitates through these to drawpoints for loading into trucks for haulage to surface.

A 3.5m high lift is mined and a 2.2m long cut is drilled with hand held drifters. A blast advance of 2m is achieved each shift giving a total face advance of 4m per day against a target of 5 m/day. Mucking is by electric LHD. Jarvis Clark JS 100E's are used in stopes wider than 2m, and Francelader CT 500 HE's in narrower stopes. The latter are planned at a minimum width of 1.5m though 1.3m is obtained. After the lift is mined out, steel wire mesh is placed on the floor and pinned to the sidewalls. A timber catwalk for access during filling is built, and on this a 150mm concrete line is laid. Fill is poured to a depth of 2m, with ventilation holes left at intervals along strike. The fill is a 10 MPa concrete mixed in 1.5m³ batches in a surface batch plant. Batches are pumped to the stopes at 3-5 minute intervals. Sand and gravel are produced from local gravel pits, the product being screened to produce sand at - 3/8" and aggregate at -1" + 3/8". Two mixes are used, with cement:aggregate:sand ratios of 10:34:64 and 10:30:84. Plasticizers and accelerators are added to give slumps of 9.5"-10", and a setting time of 1.5 hours. Mining recommences under the fill after five days, the time taken to strip the stope access ramp down to the next elevation.

There are five mining crews, consisting of two men on each of two 10 hour shifts. Generally, four crews are on production and

one on development. A stope crew carries out all operations in its stope. A typical cycle is:

Mining (drill, blast, muck, ground support)	18-20 days
Fill preparation	4-5 days
Filling	1-2 days
Ramp access stripping	5 days
Total (average)	30 days

The figures in the following table have been calculated as the target production rates for a variety of stope widths and lengths for the two types of LHD. Stope labour productivities in tonnes per manshift (NB. 10 hour shift) can be obtained by dividing these figures by 4. All operations in the stope cycle as listed above are included. Actual productivities are approximately 80% of these target figures.

Target Production Rates – Tonnes/day

LHD Type	Stope Width (m)	Stope Length (m)				
		50	75	100	125	150
CT 500	1.0	33.3	37.5	39.8	40.9	41.4
	1.5	44.6	50.1	52.7	53.7	53.8
	2.0	54.8	60.9	63.4	64.0	63.5
	2.5	63.9	70.4	72.6	72.6	71.5
	3.0	72.1	78.6	80.4	79.7	78.0
JS 100	2.0	58.5	67.3	72.4	75.3	76.8
	2.5	69.0	79.0	84.4	87.4	88.7
	3.0	78.7	89.6	95.2	98.0	99.1

Dome Mines Ltd, Ontario, Canada

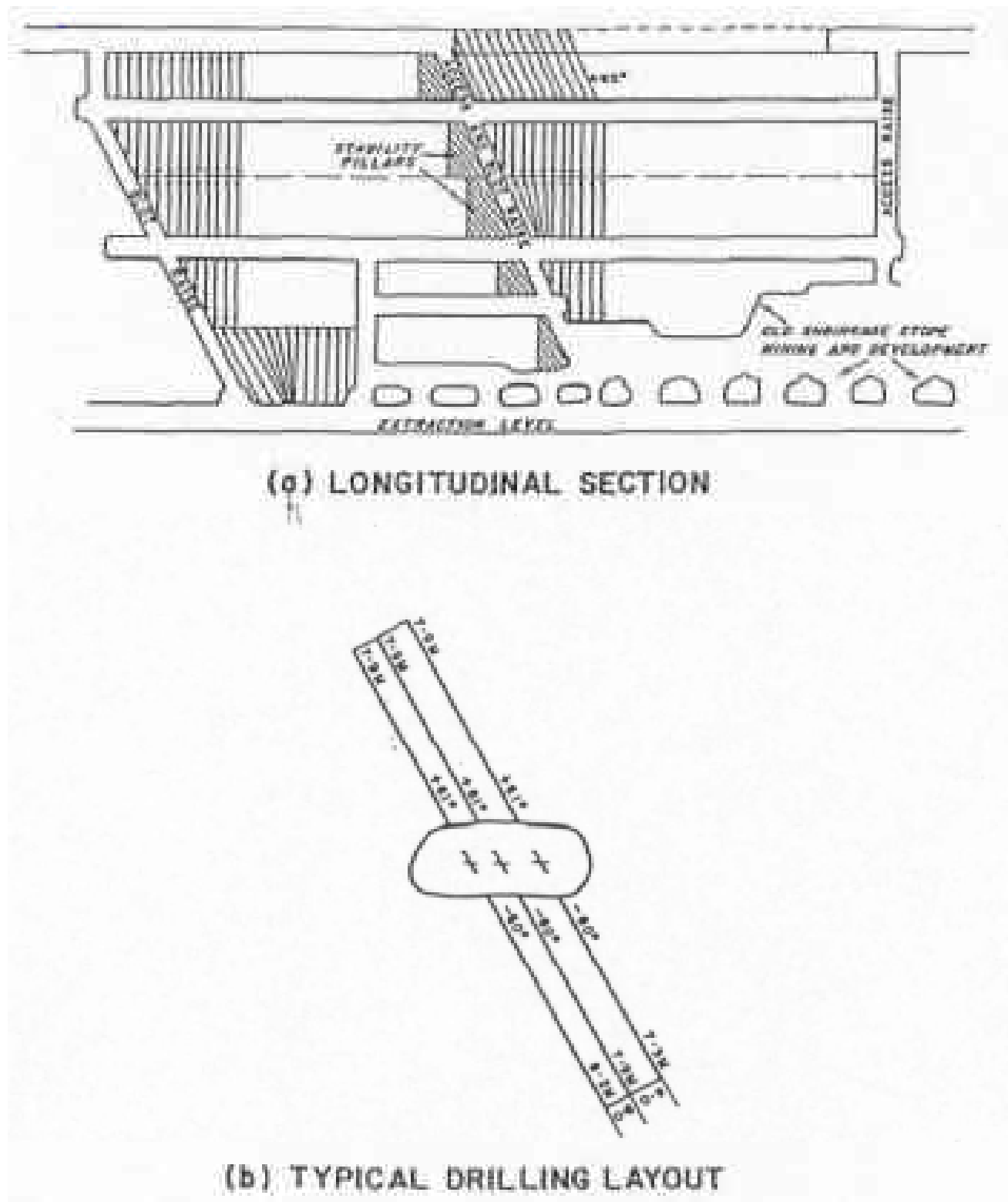
The Dome Mine at Timmins, Ontario, produces 3,800 tonnes per day at 4.5 g/t Au. The bulk of production is obtained from "wide" orebodies, but a certain amount of production comes from narrow orebodies. The latter are quartz and quartz-ankerite veins in a variety of country rocks. Widths range from a few centimetres to 6m, and dips are generally 65°.

Mining methods in narrow orebodies at Dome have been comprehensively described by Robertson (1981, 1984, 1986). In this paper, therefore, descriptions of methods used will be brief, giving only the basic outline, with productivities, unpublished details obtained by the author on his visit, and some changes made since Robertson's papers were written.

Sublevel Blasthole Stopping (Figure 21)

A sublevel open stopping layout has been used for stopes as narrow as 1.5m. The method is only applicable in areas with competent stope walls and favourable stress regimes; otherwise overbreak rapidly generates excessive dilution, rendering the grade from the stope uneconomic. Originally designed with 17m sublevel intervals and 1m maximum hole lengths, holes up to 15m long are now drilled. Hole diameters remain at 51mm, burdens are 1.2m, and toe spacings for the three hole pattern are 0.9m on the hangingwall side and 0.6m on the footwall. Drilling rates of 49 m/shift are achieved. Overall stope productivities are quoted at 15.5 t/ms.

Figure 21 – Dome Mines Ltd – Sublevel blasthole stope (after Robertson, 1981)



Modified "Avoca" Method (Figure 22)

Dome's modified Avoca method is now a cross between the original Avoca method and cut and fill. Upholes inclined at 65° retreating from a slot raise at the end of the stope are blasted in 20m sections. The lift height is 7.7m. The newly created back is made safe from the muckpile, and mucking is by remote controlled LHD to reduce the amount of ground support work necessary on the sidewalls. Initially, fill was poured tight against the stoping brow, with one bulkhead being built under the brow. The next block was commenced by tight firing into the fill. This was found to be unsuccessful, and the current method involves placing the fill in two or three pours, building stepped fill fences for each pour. Not only is a void maintained for breakage of the first firing of each block, but also the ventilation circuit through the stope is not broken.

Cut and Fill

Dome have replaced slushers with the Francel loader CT 500 HE Microscoop in the narrowest cut and fill stopes, down to 1.2m wide. Narrow stopes are drilled with 2.4m hand held upholes, and wider stopes with the Joy Stope Wagon up to 3.6m. Effective lift heights are 2.3m and 3.3m respectively. Fill is poured to 2.4m from the back and has a 0.3m cap of cemented fill to improve ore recovery and reduce fill dilution. Stope floors are very clean, giving an added benefit of greatly reduced tyre wear.

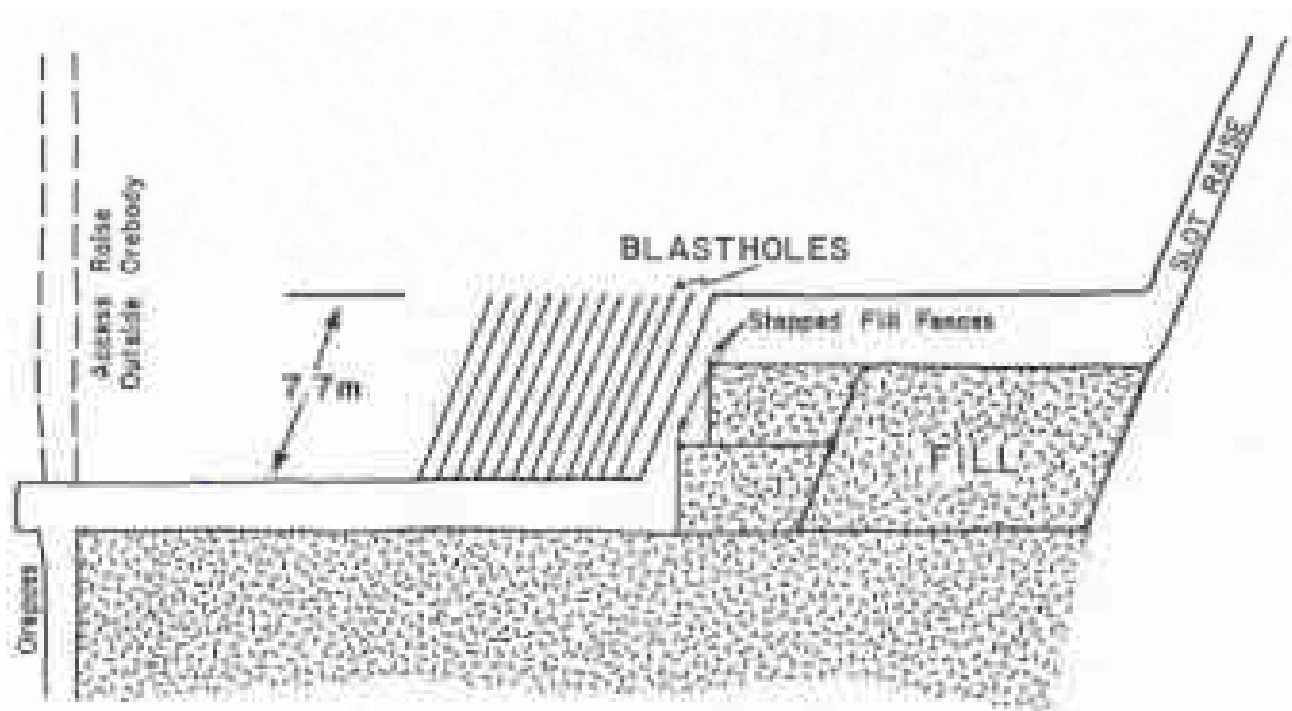
Productivity figures for some typical narrow cut and fill stopes and a modified Avoca stope are tabulated below for the time periods shown.

	Cut and Fill		Modified Avoca
	(1stope - 12months)	(2 stopes - 3 months)	(1stope - 12months)
Manshifts			
Drilling	56.5	37.9	65.0
Mucking	86.3	46.0	90.5
Rockbolting	71.9	35.9	13.8
Fill preparation*	35.1	10.6	49.9
Filling	79.9	23.4	77.8
Miscellaneous**	159.3	95.1	170.5
Total Manshifts	489.0	248.9	467.5
Rockbolts installed	1421	898	269
Tonnes mucked	4397	2756	5542
Tonnes/manshift	9.0	11.1	11.8

* Includes fill fences, orepass extensions and drainage installation

** Includes scaling, manway maintenance and transport of gear

Figure 22 - Dome Mines Limited - Modified "Avoca" method - Longitudinal section



Coal Longwall Techniques Applied to Steep Narrow Seams

A number of European coal mines are using longwall mining methods in steep seams with dips up to vertical, in widths down to 1.2m. As noted in the Introduction, these methods, using coal cutters, are not immediately transferable to hard rock mines, but can conceivably be adapted by replacing the cutter with another type of rock breaking system more applicable to the ground being mined.

Three types of methods are described below, grouped by method rather than by mine. The methods are differentiated by their direction of advance, viz. up dip, along strike, and down dip.

Steep Updip Longwalling

The updip method is in use in what are known as semi-steep seams at the Simon colliery of Charbonnages de France's Houillères du Bassin de Lorraine (HBL). In practice, the method is used in dips up to about 50°. The method uses conventional longwall systems, with some modifications as described below to account for the steep dips.

The chocks are manufactured specifically for the steeper dips. There is an angle between the cap and the hydraulic legs such that, with the chock installed in the seam, the legs are vertical, rather than perpendicular to dip. The shearer face conveyor assembly is also angled so that the conveyor is horizontal, while the face is cut perpendicular to the hangingwall and footwall of the seam. Where the seam conditions require it, the caps of the chocks are fitted with extensions which swing down perpendicular to the cap to press against, and thus give support to, the overhanging coal face. The caps also have flaps which swing down to the edge of the conveyor remote from the face, thereby separating the face, shearer and conveyor from the travelling way between the legs of the chocks. This keeps workers in the chocks safe from the danger of falls of ground in the face area.

As the face advances, hydraulic fill is poured behind the chocks. A pass is maintained in the fill at the conveyor discharge end of the face for the broken coal to gravitate to the main level below.

Steep Longwalls Advancing Along Strike

Variations of this method are used at HBL's Vouters and Simon collieries in Lorraine (Figure 23) and at Hunosa's Fondon Shaft in Asturias, Spain (Figure 24). In all, cases, the face in the steep seam lies underhand, at 20-25° off vertical. Hydraulic chocks are installed along the face except for the top few metres, which are supported by timber stulls or single hydraulic props near the face, and anhydrite pillars behind the face to maintain the integrity of the top access drive.

HBL places sand fill behind the advancing chocks. Wet fill from surface is dewatered in a dam near the face, and conveyed to the stope by conveyors slung on a monorail in the top access drive. Material is also transported to the face by a second monorail in that drive. Hunosa employs a combination of caving and filling. Development mullock and washery rejects are transported to the stope in sidetipping rail trucks, as for the manual method described earlier for Hunosa.

The chocks used by HBL are conventional longwall chocks with some modifications to account for the steep dip. They are linked together in groups of three (see Figure 23) to assist in preventing creep down dip, and are fitted with plates on the sides and rear to prevent ingress of fill into the working area. Chocks have been recently redesigned to extend the footings as well as the caps up to the coal face in order to counteract footwall failures experienced in the latest working areas.

Hunosa's chocks are designed specifically for the steep dips. As well as the usual hydraulic legs across the seam, an additional jack is placed in the plane of the seam between the "sides" of the chock. This jack may be used during chock advance to push the overlying chock up dip, thus countering creep down dip. Since Hunosa uses a coarser fill than HBL, a total sealing off of the goaf is not required. The chocks accordingly are fitted with angled back plates, expandable with changing seam width, which overlap to prevent fill/caved waste rilling into the working area (See Figure 24).

The method of manoeuvring the coal cutter along the face is also variable. HBL retains the conveyor pan, carried over from flat dip systems, on the steep dip chocks to guide the shearer. This is moved along the face by the "Dynatrac" system, whereby a drive on the shearer itself pulls it along a stationary chain supported on the conveyor pan. Cutting is done on the downstroke only. Hunosa uses a shearer mounted on a beam parallel to the face and attached to the chocks. The shearer is pulled up the face by a cable on a winch mounted in the top access drive. Cutting can therefore only upwards.

Hunosa's chocks are fitted with handholds for providing access through the face. HBL fits its chocks with ladders between the legs, mesh between the ladders and face, and trapdoors on the ladderway at regular intervals.

A wide range of productivities is quoted. Hunosa quotes 8.5 t/ms for one cut per day, with 20 workers in the face. HBL quotes previous achievements of 12-19 t/ms, and hopes to improve on this after making changes to the chocks. The crew, spread over four working shifts, consists of:

- 12 chock operators
- 8 shearer operators
- 5 chock maintenance men
- 2 shearer maintenance men
- 12 men outside the face, on tailgate conveyor etc.

HBL claims productivity improvements over the multiple attack method resulted in payback of capital costs for a longwall system in 15 months, corresponding to approximately 1 million tonnes of coal.

Figure 23 – Vouters Colliery – Steep dip strike longwall

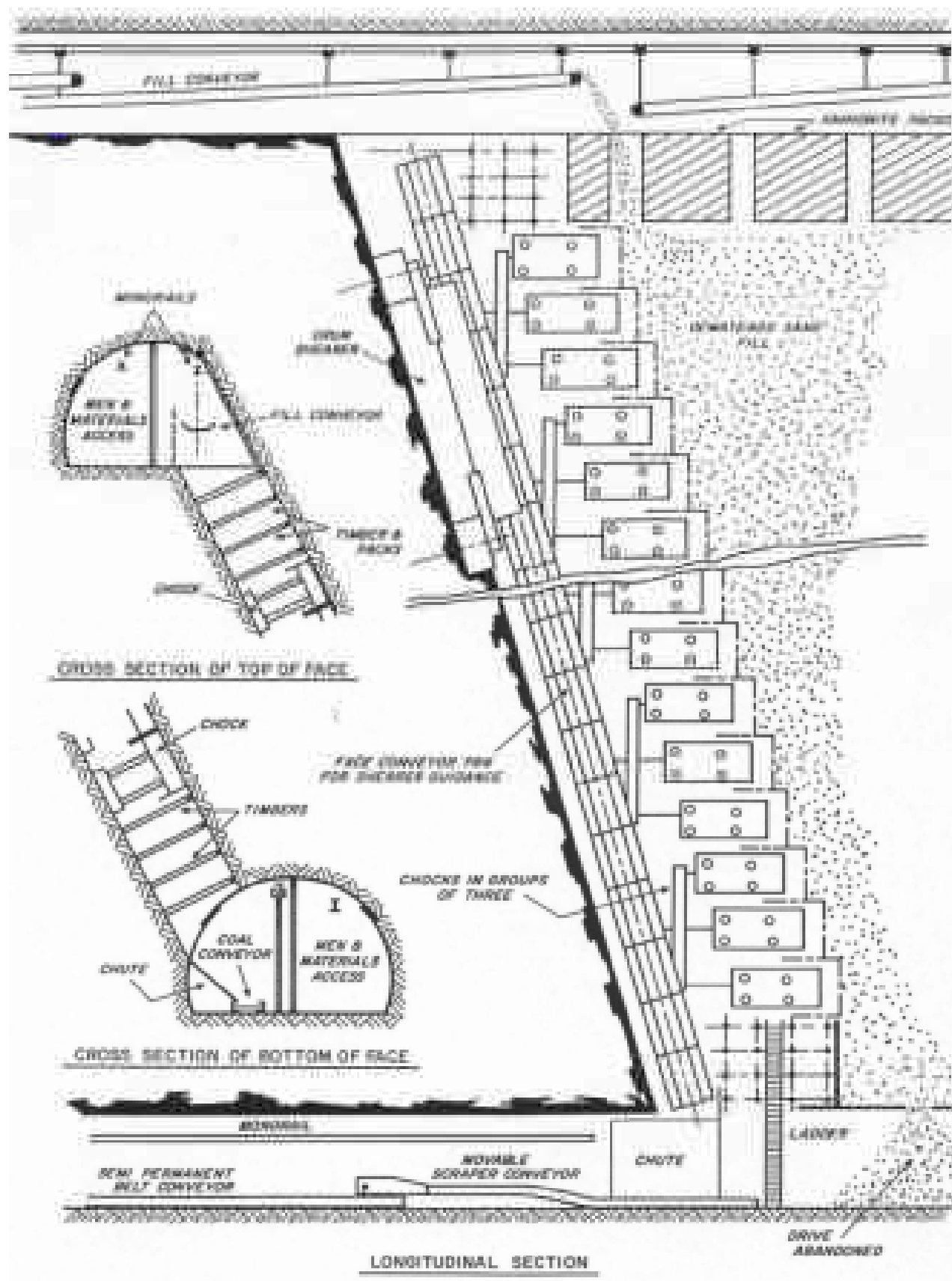
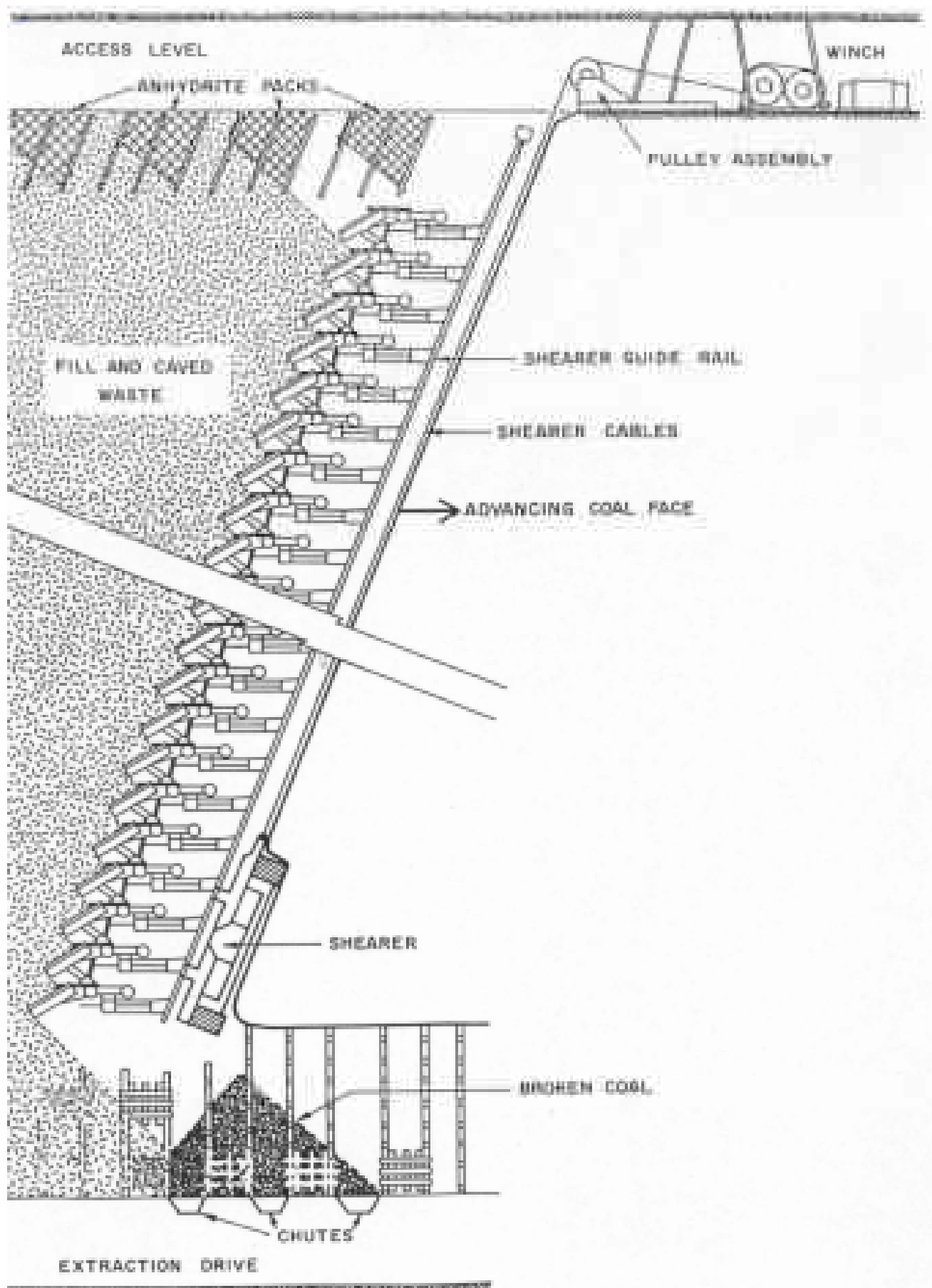


Figure 24 – Hunosa – Fondon Colliery – Longwall advancing along strike



Downdip Longwalling

Hunosa are using a Russian system for mining down dip at the San Antonio Shaft (Figure 25).

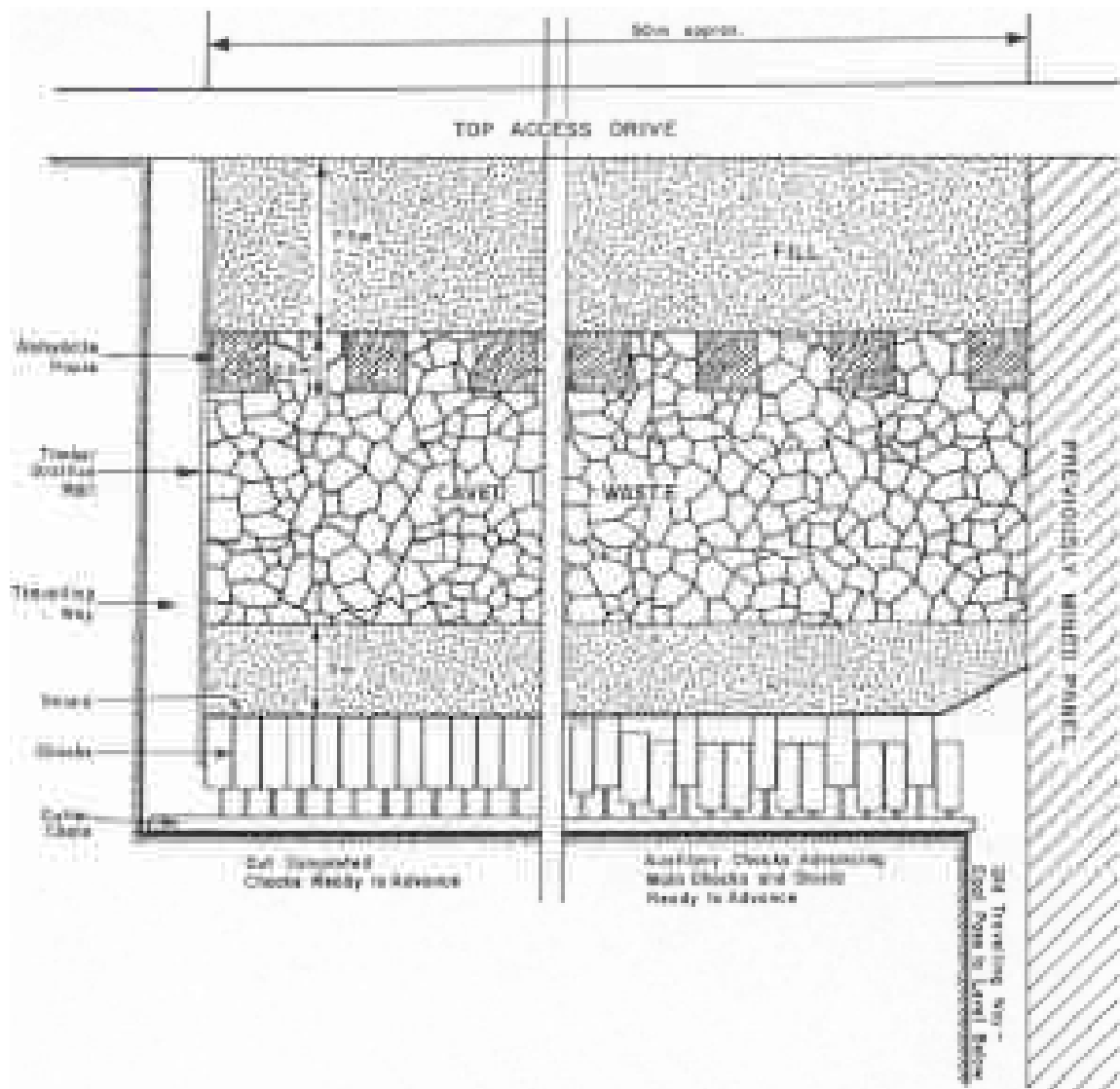
The panels being mined are 40-50m long, 1.5-2.0m wide and 100m high. Chocks are installed in groups of three; one main with two auxiliaries. One auxiliary in each group carries the hydraulic control panel for the group. Coal is cut by a continuous chain on which are mounted cutters, and the coal is swept along the face to the pass to the level below by the movement of the cutter chain along the face. The beam carrying the cutter chain is pushed forward by a ram on the main chocks. When a cut is completed, the auxiliary chocks are advanced individually, the main chocks remaining in position. All main chocks are then advanced simultaneously. The shields retaining the fill in the goaf are attached to and move down with these main chocks.

As the face is advanced downwards, a travelling way is established at one end of the face. The travelling way from the previous panel becomes the pass for broken coal to the level below.

At the commencement of mining the panel, a 5m layer of fill is placed on top of the chocks. For protection of the top access drive, a 7.5m layer of fill is placed on top of 2.5m high anhydrite packs. The chocks are initially installed immediately below the top access drive, and the top of the panel is supported with timbers until sufficient mining has taken place for the required packs and fill to be placed. Thereafter, the goaf is allowed to cave. At the bottom of the panel, a 5m pillar of coal is left for protection of the drive. This is recovered on the final retreat from the area.

Mining of a panel takes approximately four months with a crew of three men per shift in the face. Installation of the system takes approximately one month. Space is made for a set of three chocks by moiling out the coal below the top access drive with a hand held pneumatic pick. The cutter chain is extended as each set of chocks is installed to sweep the moiled out coal to the pass at the opposite end of the panel. Installation of the equipment in a new panel is done by three shifts of four men, with a crew of similar size used for concurrent dismantling in the completed panel.

Figure 25 - Hunosa San Antonio Colliery – Longwall advancing down dip – Long section



The face labour productivity during the production phase is quoted at 13 t/ms. The overall productivity, including dismantling, reassembly, and placing of timber, packs and fill at the top of the panel, is about half this figure.

Equipment for Narrow Vein Mining

Most of the equipment used in the narrow vein mining seen by the author is already available and in use in Australia. The main exceptions are the Franceloader CT 500 HE Microscoop, and its companion drilling jumbo, manufactured by L'equipement Minier. With respect to alternative rock breaking methods for use with adapted mechanized coal methods, the Chamber of Mines Research Organisation's development of an impact ripper based mining system was also investigated in South Africa.

The CT 500 HE Microscoop and Drills

L'equipement Minier is a French mining equipment manufacturer based in Paris. The spare parts and after sales service sections are located in Nancy, and the factory is located 35km north west of Metz, near the village of Briey. The company employs 180 people and has an annual turnover of some 100 x 10⁶ French Francs, split 50:50 between equipment and spare parts sales. Approximately two thirds of its production is exported, and its equipment is now working in over 35 countries. The factory produces between 7 and 10 underground vehicles per month. The majority of these are load-haul-dump units, trucks, and service vehicles. Both flameproof and non-flameproof models are available.

A range of load-haul-dump units (LHD's) from 0.6t to 15t capacity is manufactured. The smallest of these, the CT 500 HE (Figure 26) has a bucket capacity of 600kg, and an overall width of 850mm. It is, to the best of the author's knowledge, the smallest LHD on the market. The manufacturer is hoping to establish a presence in Australia with this machine, part of its "Microgeneration" range of equipment specifically designed for narrow vein operations.

As noted in Section 3 above, a number of mines visited by the author are using this equipment. The largest number are operating in Peru. Cardenas (1983) reports their use at the Huaron mine in stopes as narrow as 1.2m. The manufacturer indicated that this width has since been reduced to as little as 1.0m.

Generally, remarks by both operating and maintenance staff at mines using the equipment were very favourable. The most controversial item relates to the driver's seating and protection. The manufacturer recognizes that, in a machine such as this, built to a strict size specification, some compromises are necessary. The original seat supplied was a motorbike type of seat, straddled by the operator, which proved to be uncomfortable in use. The seat now provided is shaped to suit a driver's bottom, and curves up the back slightly to provide some lower lumbar support (see Figure 26).

The absence of a seat back relates to the philosophy adopted for driver protection. The manufacturer has worked on the premise that a properly designed and constructed canopy would increase the width of the vehicle at head height to such an extent that the consequent increase in mining width in a non-vertical orebody would be untenable. (The geometry of the machine is such that, at approximately 65° dip, the clearances to the hangingwall at both the driver's head and the top outer corner of the machine are equal.) It is contended that, at the slow tramping speed (7.2

km/h maximum), the risk of the driver striking his head is low, and the potential severity of an injury so sustained is also low. Furthermore, the size and shape of a canopy truly able to protect the operator in a serious fall of ground is out of all proportion to the size of the machine.

The manufacturer has therefore adopted the position that, since construction of an effectual, robust canopy is impractical, the operator should be able to escape from the machine as quickly as possible. To this end, the seat does not have a high back, so that the driver can retreat with minimal impediment. However, the driver's position has been ergonomically designed to maximize his comfort.

Other concerns related to stability and cable reel protection. With respect to the former, one operator found the machine to be unstable, a concern not expressed by others. The manufacturer reports static overturning tests conducted with the machine jackknifed. Instability occurred on lateral gradients as follows.

Bucket down	- empty	- 27.5° measured
	- full	- 23.2° calculated
Bucket up	- empty	- 15.8° measured
	- full	- 11.4° calculated

With respect to cable reel protection, one operator had problems with muck from the floor packing around, and impeding, the cable reel and its operating mechanism, which are under the driver's seat. Fitting of a protection plate under the vehicle did not solve the problem.

Both these problems appear to relate to operating practices at the mine in question, where floors were uneven, and a sharp transition from declined access ramp to horizontal stope floor permitted the tail of the vehicle to scrape the floor.

Operators in Canada found it necessary to make modifications to the earth leakage protection of the trailing cable system to meet local legal requirements. The same may apply in Australia. The cable reel is designed to hold 85m of a light 25mm diameter cable on a drum with a 300mm minimum diameter. Compliance with Australian Standard AS 2802-1985 may have an adverse effect on cable reel capacity. Prospective Australian users would need to follow through these aspects with the relevant regulatory authorities. The author has not pursued these matters further.

Availability figures were provided by two mines. Differences in operating practices, maintenance policies, and methods of calculating figures make comparisons difficult. However, from these raw figures, the ratio of operating hours: maintenance hours was 1.44:1 for the one case and 5.21:1 for the other. In the former case, operating conditions were difficult, all maintenance was done by maintenance crews and the machines were both operated and maintained by a contractor. In the latter case, operating conditions were good, some basic preventative maintenance was done by operators, and was not recorded as maintenance time, and the machines were operated and maintained by the owner.

L'equipement Minier also manufactures a variety of drilling jumbos to complement the Microscoop, based on the Microscoop chassis. The "Minifore" is a single drilling boom which mounts on the bucket of the Microscoop, and can be supplied for either vertical or horizontal drilling. The fitting of the boom can be accomplished by the LHD operator without

Figure 26 – CT 500 HE Microscoop (reproduced from manufacturer's brochure)

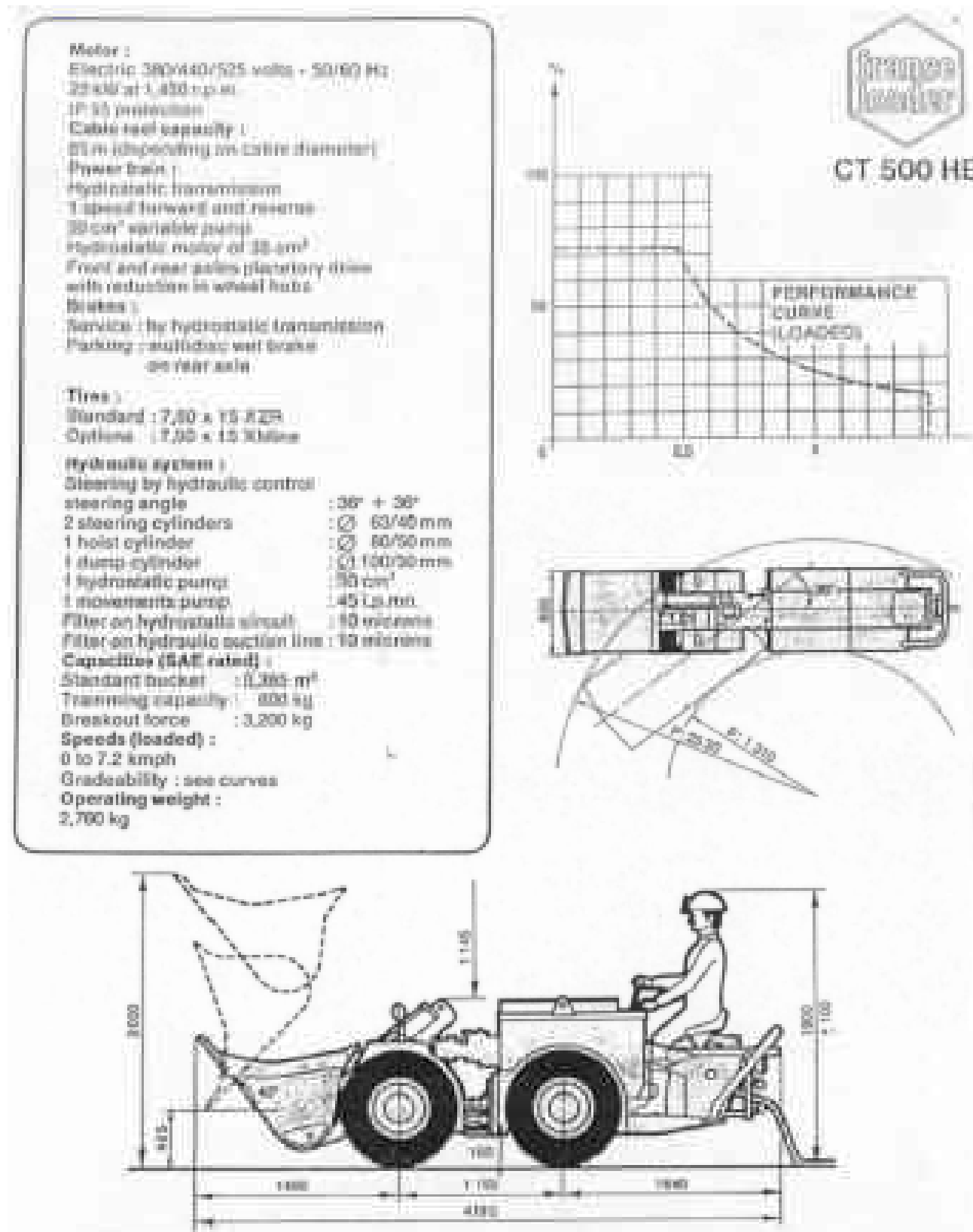
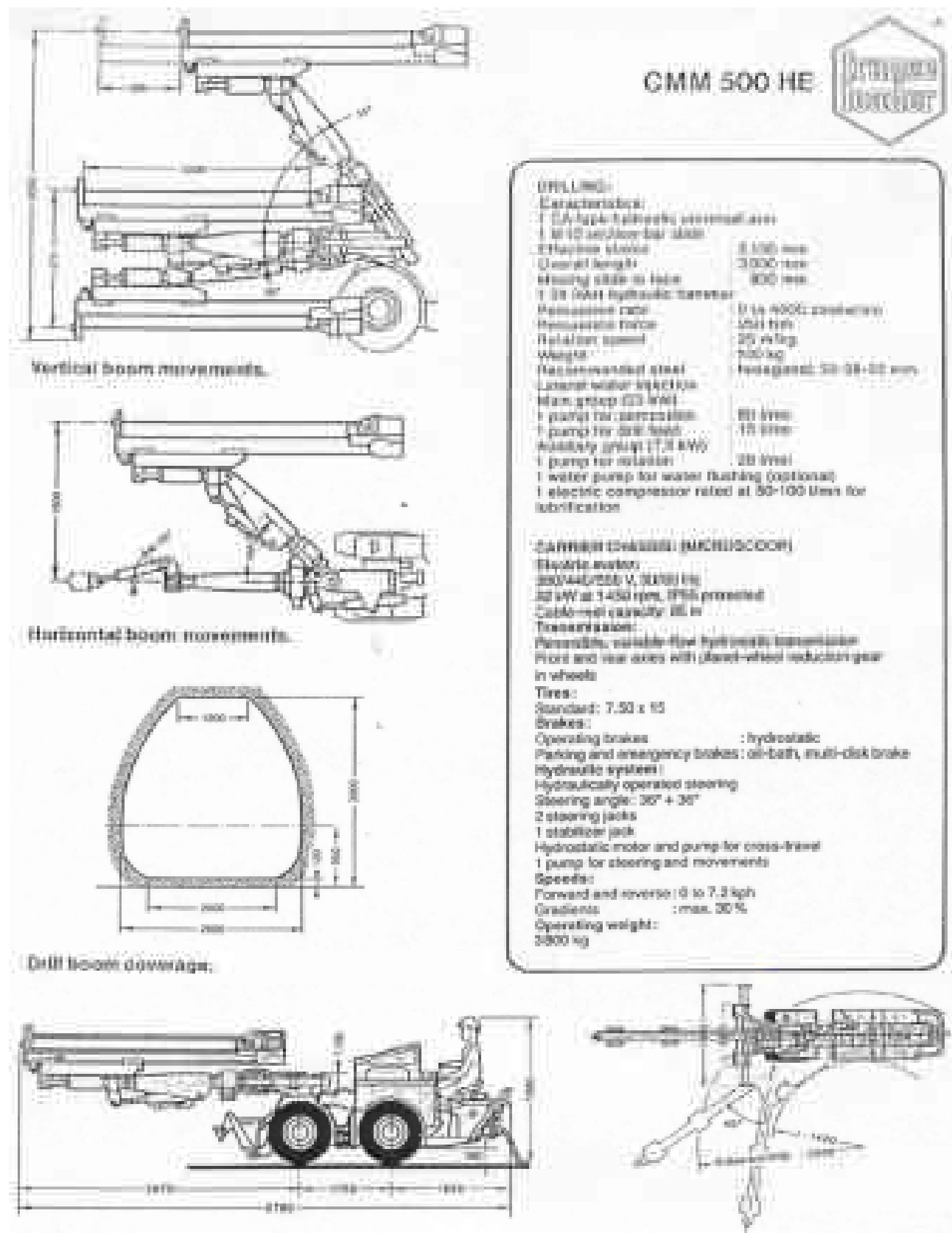


Figure 27 – CMM 500 HE Hydraulic Microdrill (reproduced from manufacturer's brochure)



assistance. An auxiliary hydraulic connection on the Microscoop must be provided to power the drill boom.

The Microdrill CMM 500 HE and MAC 500 HE, mounting hydraulic and pneumatic drills respectively, are carried on a Microscoop chassis with the bucket replaced by the drilling boom and stabilizing jack. Some details and capabilities of the CMM 500 HE are shown in Figure 27. The MAC 500 HE has similar dimensions and capabilities, the only difference being in the drifter.

These drills are comparatively new to the "Microgeneration" range, and no details from operating mines were available to the author. Dome Mines were preparing to take delivery of a MAC 500 HE at the time of the author's visit.

The Impact Ripper at Doornfontein Gold Mine

The Chamber of Mines Research Organisation in South Africa has been experimenting with non explosive rock breaking techniques in the gold mines for a number of years. The author visited the Impact Ripper Project at the Doornfontein Gold Mine to complement visits to coal mines where mechanized methods would require an alternative rockbreaking system for use in hard rock mines.

The present impact ripper investigation has evolved over some 15 years from research into a variety of non explosive rock breaking techniques. The machine currently in use is the third pre-prototype experimental machine (Figure 28). It is operating in a stope 1.1m high at a dip of 23°.

The hammers have been designed from scratch because of the unsuitability of the geometry of existing hammers - comparatively long and slender - in the South African gold mine stope situation. This would not necessarily be the case in a steeply dipping vein. The hammer developed is 1260mm x 450mm, and delivers a blow energy of 3500 J at 200 blows per minute.

It is important to note that the hammer is classed as a ripper, not a breaker. It is not designed to fracture the rock, but merely to break it out on pre-existing fractures. In the case of this

operation, the relevant fractures are stress induced, ahead of the advancing face. Broken rock is swept by the hammer onto a reciprocating flight conveyor installed along the face, which is 64m long.

The present machine is operated by a 95:5 water:oil emulsion. It is planned that the next version, probably to be classed as the prototype production model, will be built to operate on 100% high pressure water. This is in line with the trend towards "hydropower" in the deep South African mines, where the pressure of service water in the mains from surface is used to operate machinery.

The unit, weighing 5t, is moved by hydraulic rams along a toothed rack on the edge of the conveyor pan, on which the whole machine is mounted. Much of the current machine comprises the hydraulic powerpack, which would be eliminated by the move to 100% water power with the pressure derived from the head of water from surface.

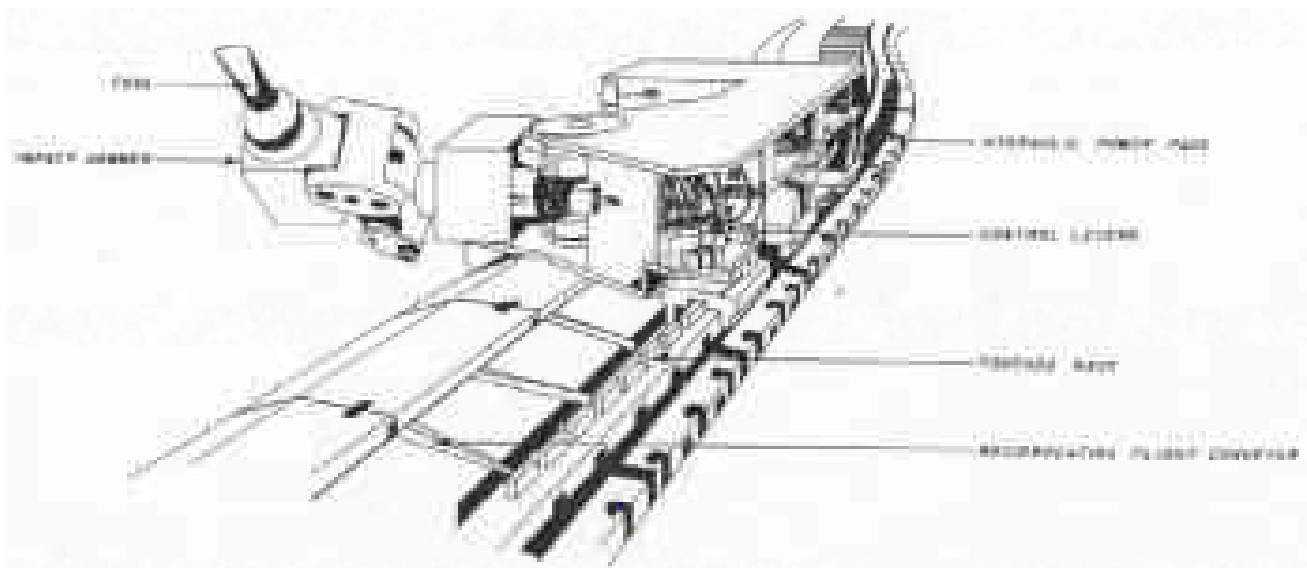
Productivities of the latest developmental machine have been very encouraging, being in the upper end of the range of productivities for South African gold mines. The potential for the system has been demonstrated by the achievement, in individual six hour shifts, of face advances between 5 and 6m, which is the industry average for a month.

Ground Support Practices

The support of sidewalls of narrow stopes presents a need for compromise. If it is desired to rockbolt sidewalls, it has to be accepted that, with existing technology, easy rockbolt installation is limited to a depth not greater than the stope width. The use of flexible or extension drill steels and flexible rockbolts or cables removes this restriction, but complicates the ground support installation. The alternative is to place stulls across the stope, but these, depending on the support density required, may impede access for mobile equipment.

It was noteworthy on the author's visits that all metalliferous mines with mechanized systems had competent sidewalls, which were able to be adequately supported either with

Figure 28 – South African Chamber of Mines Research Organisation Impact Ripper



minimal, random rockbolting, skewed into the sidewalls where necessary to permit installation, or with occasional random stulls installed above the height of the mobile machinery. No mechanized systems were observed where a closely spaced regular pattern of support was required in the sidewalls. Where closely spaced support was necessary in the narrowest stopes, stulls or packs were used in conjunction with scrapers or gravity for muck removal.

Conclusions

1. Technically feasible methods exist for mining steeply dipping narrow veins down to less than 1m wide, and could be applied to Australian deposits.
2. Rockbreaking in metalliferous stopes currently appears to be exclusively by drill and blast methods, although other methods such as road headers and impact rippers may have application as technology develops.
3. Drilling in stopes less than 2m wide appears to be the exclusive domain of handheld drilling equipment. The introduction of mechanized drilling rigs is in its infancy at these widths.
4. Support of stope sidewalls may present problems in mechanizing operations in narrow stopes. However, where stope width and sidewall ground support requirements permit, there is a move to mechanization of the mucking cycle by the introduction of small electric LHD's capable of operating in stope widths as narrow as 1m.
5. A number of narrow steep dip coal mining methods conceptually can be adapted to metalliferous operations, especially where the rock is amenable to non-explosive rock breaking techniques.
6. The economic viability of any of a number of technically feasible methods will be highly dependent on such factors as local conditions, traditions, cost structures, and regulatory requirements and constraints.

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