J. S. Afr. Inst. Min. Metall., vol. 85, no. 2.
Feb. 1985. pp. 41–49.

Cut-and-fill mining of base metals at Black Mountain Mine

by P.J. KINVER*

SYNOPSIS

This paper deals with the development of the cut-and-fill mining method at the Broken Hill orebody of Black Mountain Mineral Development Company (Pty) Limited, at Aggeneys in the north-western Cape Province of South Africa. Lead, copper, zinc, and silver are produced.

Production started in January 1976, being confined to blasthole production methods in the upper, steeply dipping areas of the orebody. This blasthole reserve is 70 per cent depleted, and pillar extraction is in progress. The balance of the production tonnage, 60 kt per month, is produced from three cut-and-fill stopes situated in the flatter-dipping areas of the orebody.

The cut-and-fill mining method at Black Mountain is a result of research and planning activities that included the use of modelling, laboratory tests, mathematical analyses, *in situ* tests, and experience gained from other mining operations.

These investigations and the resulting developed methods and designs are summarized in the paper.

SAMEVATTING

Hierdie referaat handel oor die ontwikkeling van die sny-en-opvul mynbou-metode van die Broken Hill Ertsliggaam van Black Mountain Mineral Development Company (Pty) Limited te Aggeneys wat geleë is in die noordwes Kaap Provinsie van Suid-Afrika. Lood, koper, sink en silwer word geproduseer.

Produksie het begin in Januarie 1976, en was beperk tot die skietgat ontginningsmetode in die steilhellende areas van die ertsliggaam. Hierdie steilhellende gedeelte van die ertsliggaam is alreeds 70 persent ontgin en pilaarontginning word tans gedoen. Die balans vir die addisionele tonnemaat wat sowat 60 kt per maand beloop word geproduseer deur drie sny-en-opvul afbouplekke wat geleë is in die laerhellende gedeeltes van die ertsliggaam.

Die sny-en-opvul mynboumetode te Black Mountain is die resultaat van navorsing en beplanningsaktiwiteite wat die gebruik van modelle, laboratoriumtoetse, matematiese-analise, *in situ* grondtoetsing en ondervinding wat opgedoen is by ander mynbou operasies, insluit.

Hierdie navorsing en die gebruikte ontwikkelingsmetodes en ontwerpe word saamgevat in hierdie referaat.

Introduction

Black Mountain Mineral Development Company (Pty) Limited is producing annually 1,1 Mt of ore containing lead, copper, zinc, and silver from the Broken Hill orebody, which is situated at Aggeneys, in the north-western Cape Province of South Africa. The annual production of concentrate is lead 130 kt, copper 20 kt, and zinc 48 kt, with silver reporting in the lead and copper concentrates.

The ore reserves at Broken Hill are contained in two orebodies extending over a total strike length of 1600 m, and plunging from a surface outcrop in the west down to aproximately 800 m in the eastern extremity (Fig. 1). The two orebodies run conformably and are separated by an intermediate schist, which varies in thickness between 5 and 30 m. The upper, eastern portions of the orebodies are steeply dipping at 55°, and the upper orebody has a true thickness of 20 to 30 m. Where present, the lower orebody has a true thickness of 3 to 6 m. In the lower, western portion of the orebody, the dip varies from almost flat to 40°, and the true thickness of the upper orebody varies from 6 to 30 m and that of the lower orebody between 3 and 10 m.

The stratigraphy (Fig. 2) consists primarily of footwall schists, which contain little or no water. In the footwall of the orebody is a weak zone 3 m thick that comprises graphite- and mica-rich ground, which is unconsolidated and presents strata-control problems when exposed. The orebodies consist essentially of high-grade massive sulphide lenses situated on a geological footwall, with low-grade magnetites in the hanging wall. The two orebodies are separated by intermediate schists with an average uniaxial compressive strength of 100 MPa. The geological hanging wall of the upper orebody consists of intermixed schists, quartzites, and ferruginous quartzite. The orebodies and the hanging-wall quartzites contain water that is associated with fissures and cracks.

Choice of Mining Method

At the start of full production in 1980, it was felt that emphasis should be placed on the blasthole mining area, where high extraction rates with low working costs could be achieved. This blasthole production tonnage had later to be replaced, and a mining method for the flatter-dipping areas was investigated. Because of the high grade of the orebody, it was considered of prime importance that the mining method used should permit maximum extraction.

The second main parameter that was considered was grade control. Unlike the footwall contact of the orebody, the hanging wall has an economic cut-off, which is based on the combined economic value of the constituent metals. It was observed that the distribution of the metals varied considerably, and that the actual mining limits as defined by the 20 m spaced underground diamond drill holes often changed as the development of the orebody proceeded. The proposed mining system would have to be flexible and

* Underground Manager: Production, Black Mountain Mineral Development Company (Pty) Ltd, Private Bag X01, Aggeneys, 8893 Cape Province.

SA ISSN 0038–223 X /$3.00 + 0.00.

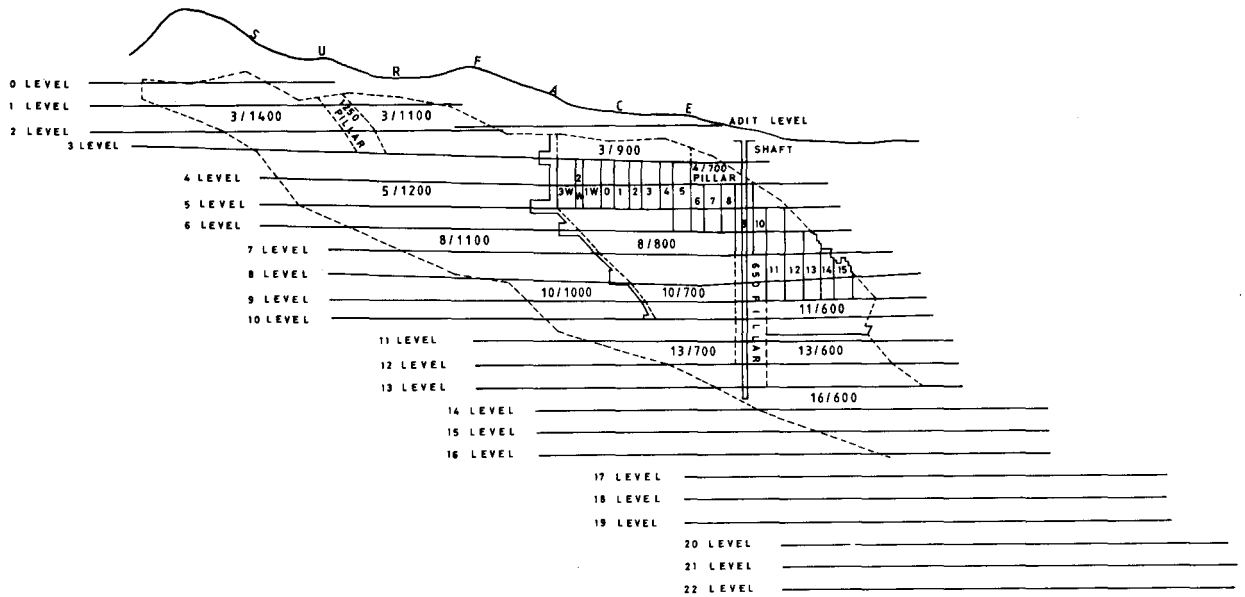

Fig. 1—A vertical stope projection at Broken Hill Mine.

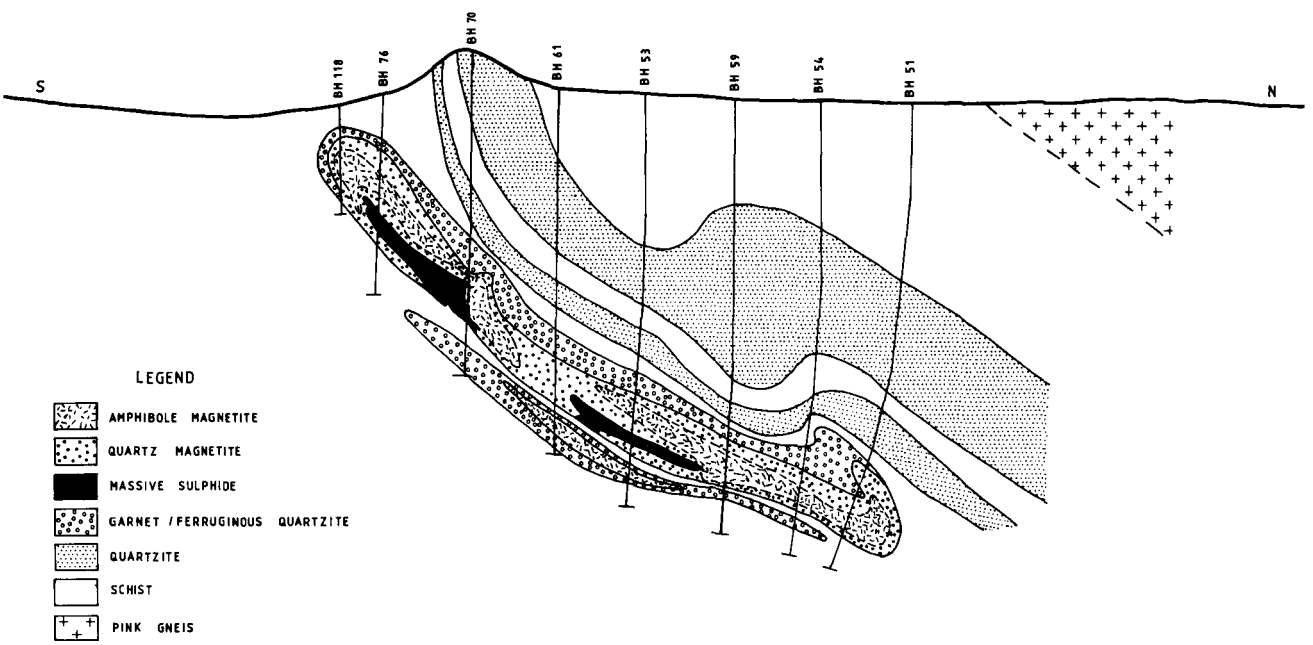

Fig. 2—A generalized section through the Broken Hill orebody.

would have to allow for the removal of ore from the hanging wall or footwall.

The flatter dip of the orebody (5 to 40°) excludes the use of a mass mining method involving large-diameter blastholes, since the blasted ore will not 'run' without assistance at this flat angle. Therefore, cut-and-fill and open-stope methods of mining were investigated.

The main reason for the choice of cut-and-fill mining as a method for investigation was its flexibility. Once a face is established, it can follow an economic hanging-wall or footwall contact, and a change in face width can easily be made. An open-stope method with production ring drilling requires a greater level of diamond-drilling information. As the dip of the orebody varies considerably over short distances, scrapers would have to be used in many areas with an open-stoping method.

Planning of the Stopes

Before any detailed planning of the proposed cut-and-fill stopes was undertaken, an investigation was made of rock-mechanics and strata-control aspects.

Rock Mechanics and Strata Control

The orebody had been diamond-drilled from surface to 40 m centres. From this information, a three-dimensional model and a computer model were produced. The following aspects were investigated:

(1) the possible requirement of major regional stability pillars, and their size and shape;

(2) the effect of backfill on the overall mine stability.

Before these questions could be answered, information was required on the strength and characteristics of the orebody and adjacent strata, the *in situ* stress values at various portions of the orebody, the characteristics of the fill, the proposed production rates from the cut-and-fill stopes, and the expected extraction rate of the major pillars.

With this information, and with the assistance of the Besol Boundary Element computer program PS002 for various extraction sequences and stope configurations, it was proposed that the overall optimum stability of the mine could best be achieved by the leaving of three main stability pillars. The computed width of 40 m was increased to 50 m after discussion and further investigation with recognized authorities of this branch of rock mechanics.

The rock-mechanics and strata-control investigations indicated that mining widths of up to 25 m and strike lengths of 150 m could be excavated at the proposed depths without any major local support.

The geological and economic hanging wall comprises rocks with a uniaxial compressive strength of approximately 150 to 200 MPa. The tensile strength, although unknown, is taken as one-tenth the compressive strength, that is 17 MPa. The *in situ* stress measurements indicated that the principal stress acted normal to the orebody, with a horizontal stress in the direction along its strike, intermediate stresses in the north–south direction, and the lowest stresses in the vertical direction. These findings agree with the results of similar investigations at other orebodies in the mining district.

Tonnage Requirements

From an examination of a financial model for the extraction of the orebody, an optimum mining rate and the areas from which this tonnage should be mined were derived. It was apparent that, as the blasthole ore reserve became depleted, five cut-and-fill stopes each producing at least 16 kt of ore per month were required.

Calculations indicated that 16 kt of trammed ore per month from a cut-and-fill stope would require four faces per stope, since at any one time 25 per cent of the faces would not be available as a result of backfilling. It was assumed that the average face width would be 20 m and the lift height 3 m. Once the size and shape of the regional stability pillars and the tonnage requirement for the proposed cut-and-fill blocks had been established, the stopes could be positioned (Fig. 1).

Layout and Design of Stopes

As indicated by Fig. 1, the orebody was divided regionally into stoping blocks of various sizes according to the pillar configurations, tonnage requirements, and grade

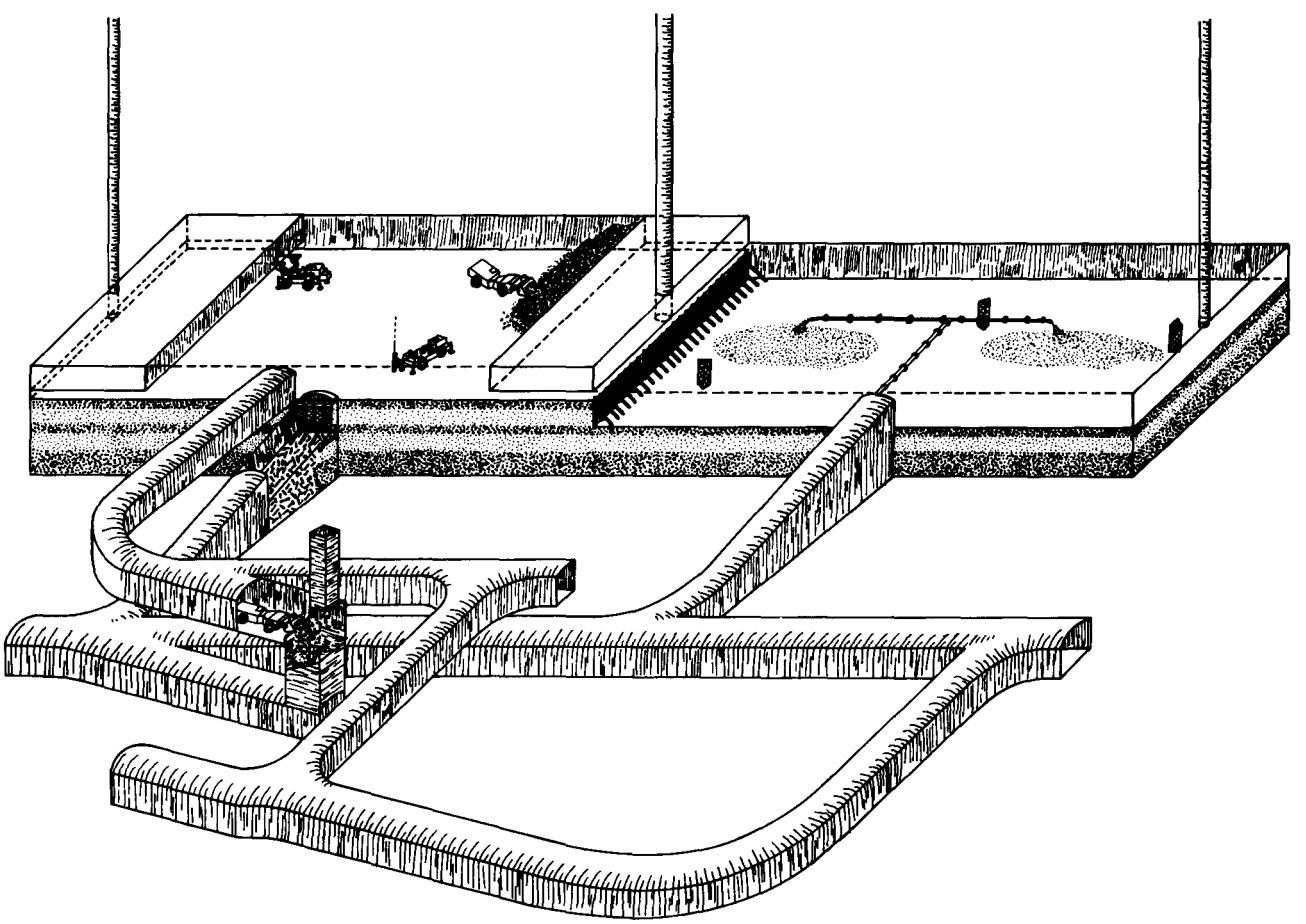

Fig. 3—A cut-and-fill stope.

valuations. The extent of the blocks on dip are not all the same since the length of back sometimes covers a distance between three main levels and sometimes between two main levels. The strike length varies from 150 to 200 m for the stope block. The main level spacing is 35 m.

Cut-and-fill stoping includes numerous methods of some variation, and the choice of any particular method depends upon the local conditions of the block of orebody to be stoped. At Black Mountain Mine, there are currently three cut-and-fill stoping blocks in full production, and two in the planned development layout. Primarily, the valuations of the stoping blocks were compiled from an analysis of surface core-drilling samples taken at 80 m intervals along the strike, which was followed by a more precise definition and valuation of the existing stoping blocks by the examination of samples from underground core drilling at 20 m intervals. These sections, which indicate the outlines of the economic orebody as delineated by the geological staff, formed the basis for all the preparatory development, planning, and stoping layouts. In addition to giving the outlines of the orebody, these sections include the outlines of the weak zone, showing its position in relation to the footwall of the orebody, which was a factor of major importance in the preparation of the planned development layout, affecting the entire stope block.

Fig. 4 shows a typical section, indicating the elevations of the core-drill sites from which the entire stope block was covered by orebody-definition drilling at 20 m intervals. The hanging-wall drive on the upper level of the block was established primarily as being convenient for core drilling, and its position north of the orebody hanging wall was predetermined through interpretation of the geological information, being vertically above the orebody horizon at the elevation of the bottom level. Its second purpose was to serve the stope as a vent return airway by linking the three upcast vent raises of the stope via vent crosscuts with the main upcast ventilation system. Access to the hanging-wall drive, linking with the upper-footwall drive, was made by the development of a crosscut in the centre of the major stability pillar that formed the western boundary of the stope block. The hanging-wall drive was also required to provide a means of access into the stope at the final stage for removal of the sill pillar below the upper stope block. The footwall drives were developed at 35 m vertical intervals, and were established at a suitable safe distance below the footwall of the weak zone so that there would be sufficient interval distance between the hanging wall of the weak zone and the footwall of the orebody to allow for maximum ramping of the stope access crosscuts, thereby making provision for only one intermediate level between the footwall drives.

The elevation of the sill-drive footwall was determined from the mean average elevation of the bottom footwall drive, which was scribed off on each section across the orebody. The outlines of the orebody were scaled off from Access to the sill drive was linked to the footwall drive by the development of two access-ramp crosscuts: one to serve the western half of the stope block, and the other to serve the eastern half. The stope block was divided into two almost-equal points on strike between the upper and lower sill levels. On the lower sill elevation, each half block on strike between the scribed centre line and the east–west stope boundary was again divided into two almost-equal halves, the midpoints providing a forward direction point between the footwall drive and the sill drive for the centre-line bearings of the access-ramp crosscuts, which were developed from the footwall drive in order to establish the sill drive that had been planned previously (Figs. 5 and 6).

Three upcast vent raises that had been planned to ventilate the stope were achieved by the linking of the lower sill drive with the upper hanging-wall drive by raise boring from the respective collaring points. Access to the collaring points was achieved by the development of 4 by 4 m vent access crosscuts off the hanging-wall drive towards the hanging-wall outline of the orebody. Vent access crosscuts were developed off the sill drive towards the footwall of the orebody to link the holing points of the vent raises with the sill drive. Fans were installed in the vent crosscuts at the top of vent raises to exhaust into the hanging-wall vent return airway, inducing the clean air to enter this sill drive–stope via the two access-ramp crosscuts of the footwall drive (Figs. 7 and 8).

After the silling out had been completed, the first backfill was admitted and made to build up the initial 3 m vertical lift above the elevation of the sill floor. Access to the new floor elevation on top of the consolidated fill was made by the sliping down of the roof in each of the access-ramp crosscuts to build ramps not exceeding a maximum gradient of + 20 per cent. The planned layout had been designed for this procedure to be repeated for the following lift of

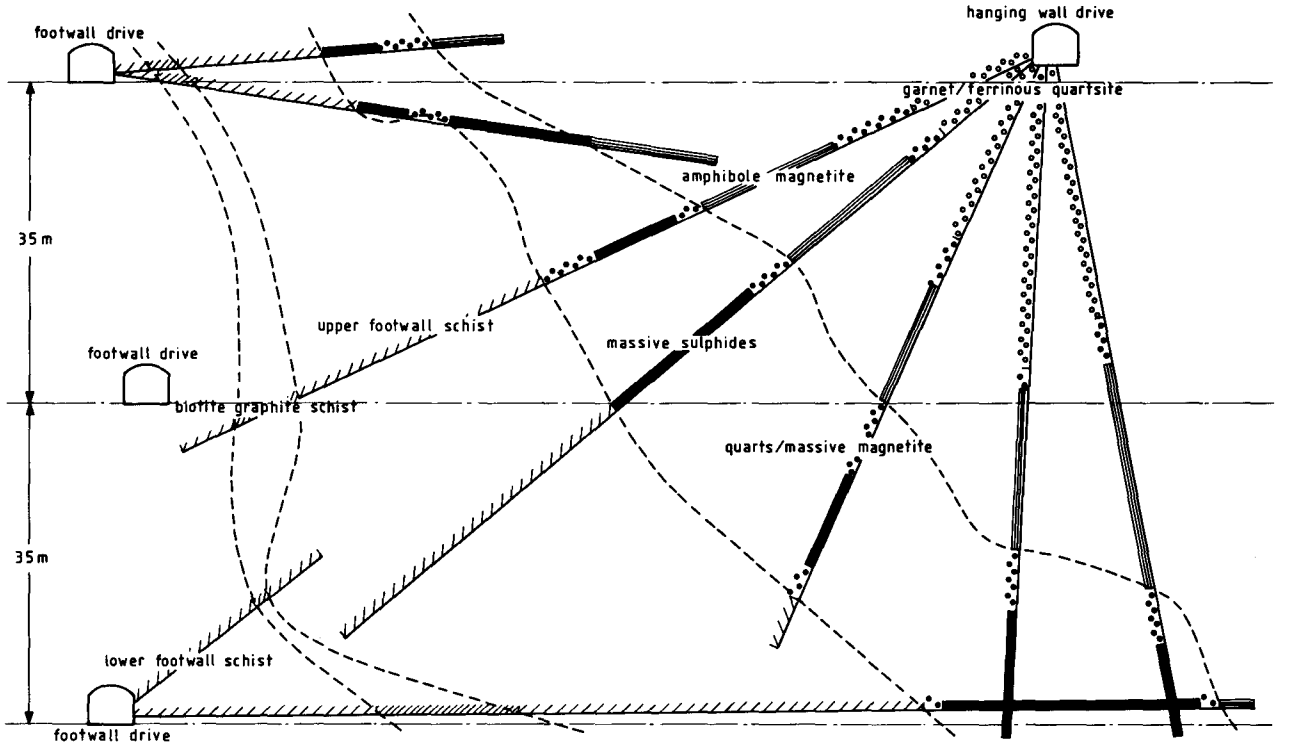

Fig. 4—A generalized section showing the required diamond-drilling information prior to stope development.

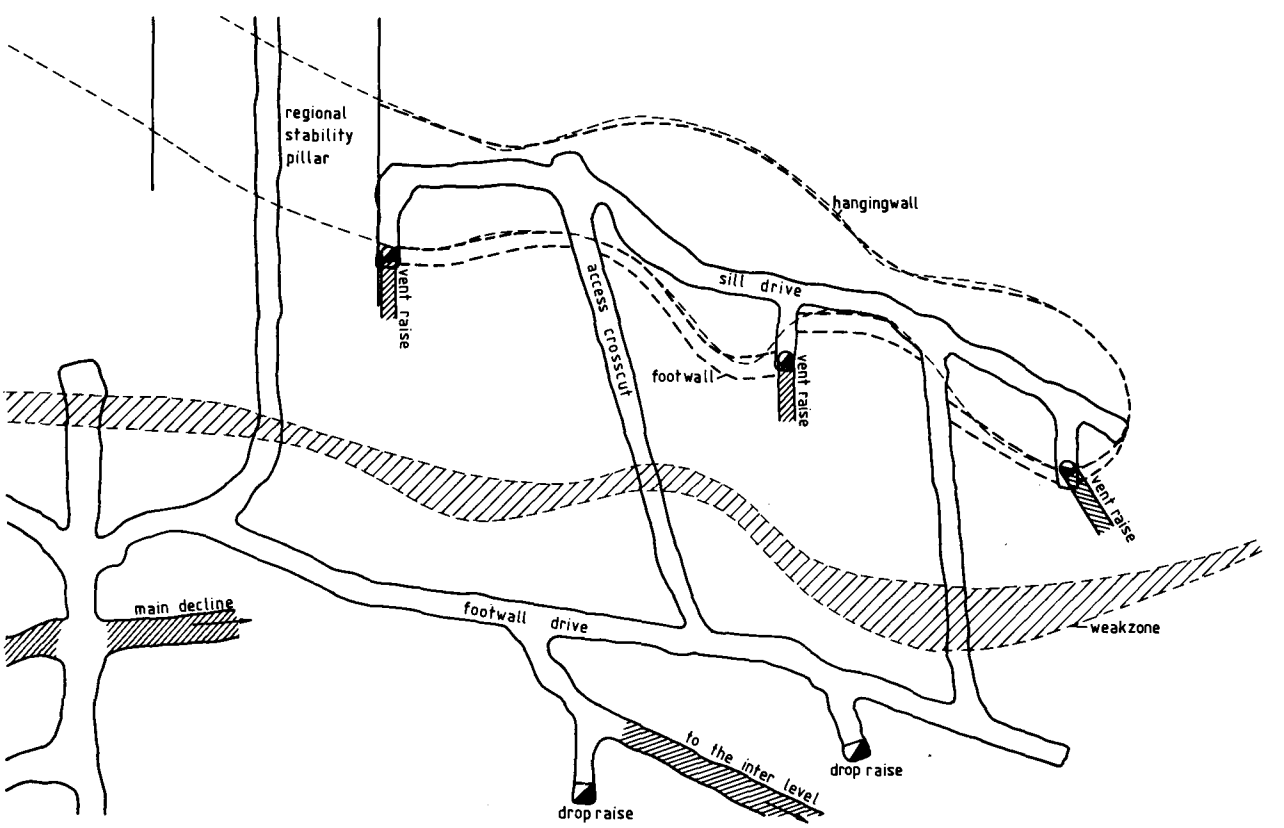

Fig. 5—Initial stope preparation.

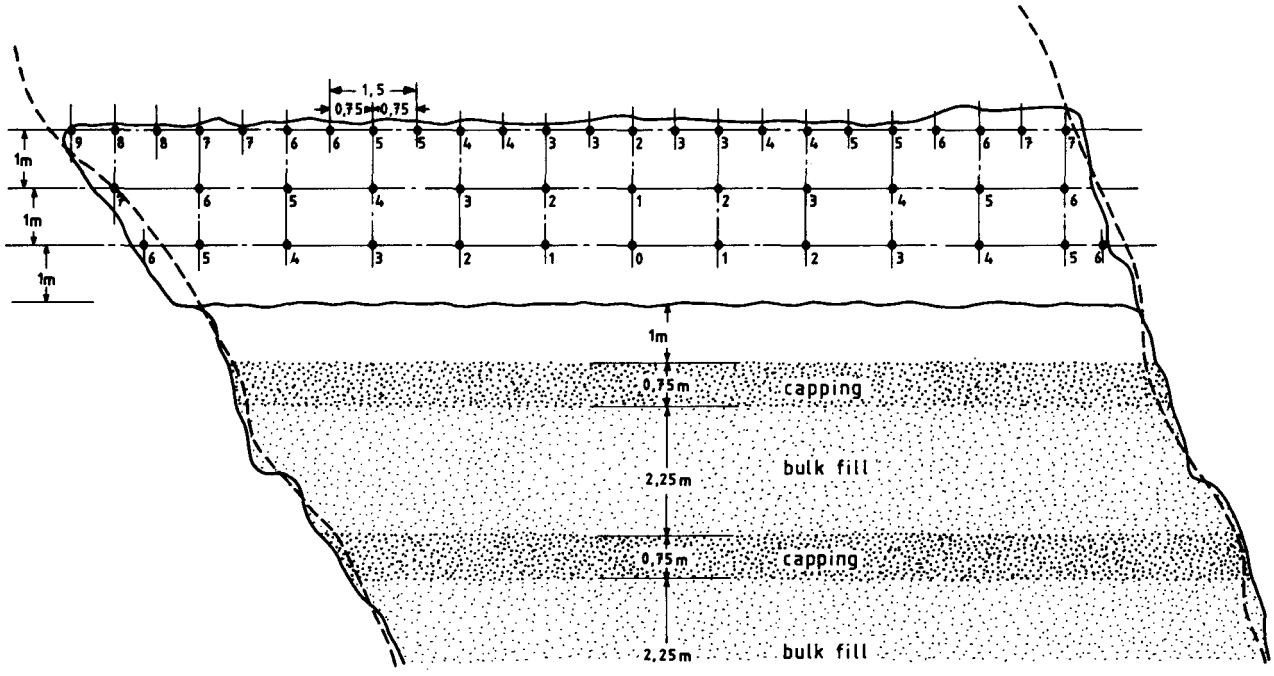

Fig. 6—The drilling layout for a cut-and-fill stope.

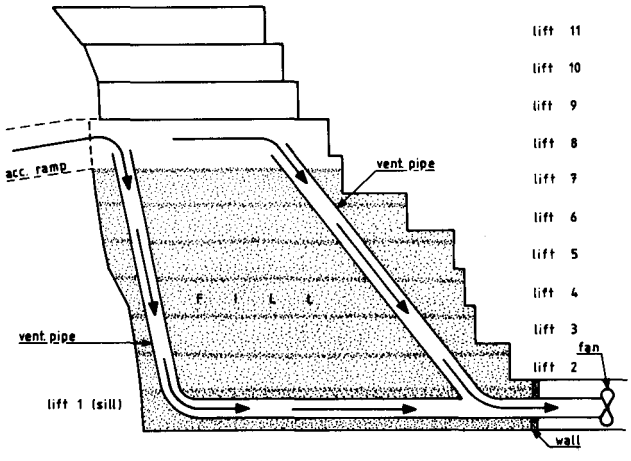

Fig. 7—Downcast method of ventilation for a cut-and-fill stope.

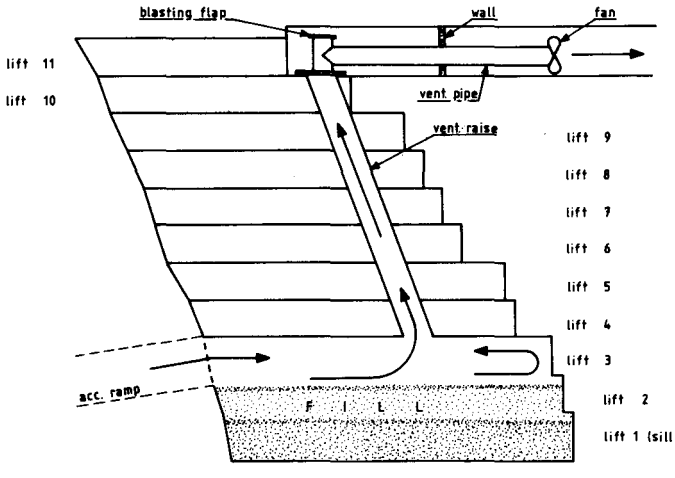

Fig. 8—Upcast method of ventilation for a cut-and-fill stope.

backfill off the lower footwall drive. The fill height at this stage was 6 m above the original floor elevation of the sill, which required a horizontal distance of 30 m to be within the limits of the permitted maximum gradient (Fig. 9).

The required number of interlevel footwall drives was dependent upon the available interval between the weak-zone hanging wall and the footwall of the orebody as determined through sections on the proposed centre line of the access-ramp crosscut. With reference to one such typical section, the procedure was to scribe a mark on the footwall elevation of the access-ramp crosscut perpendicularly below the intersection between the + 6 m horizontal line and the footwall outline of the orebody. The minimum horizontal distance of 30 m was scaled off towards the weak zone from the mark, which fell short of the weak-zone hanging wall, thereby making it feasible for a maximum of two lifts without interference of the weak zone. By trial and error, only one interlevel on the + 18 m elevation was assumed, and the scribing of a mark on this elevation line vertically above the point of intersection between the footwall of the orebody and the + 9 m elevation line required a horizontal distance of 45 m from the mark to fall short of the weak zone hanging wall, and at this point the access-ramp crosscut off the interlevel footwall drive would commence on a gradient of − 20 per cent. Another mark was also scribed on the same + 28 m elevation vertically below the + 27 m elevation intersection with the orebody footwall, which required a horizontal distance of 45 m to fall short of the weak zone, and at this point the access-ramp crosscut would complete its final lift on a + 20 per cent gradient. This procedure was repeated by progressing up each section in the manner described (Fig. 9).

Access linking of the main footwall drives with the interlevel footwall drives was achieved off the planned layout by the development of a ramp established in the footwall. Two

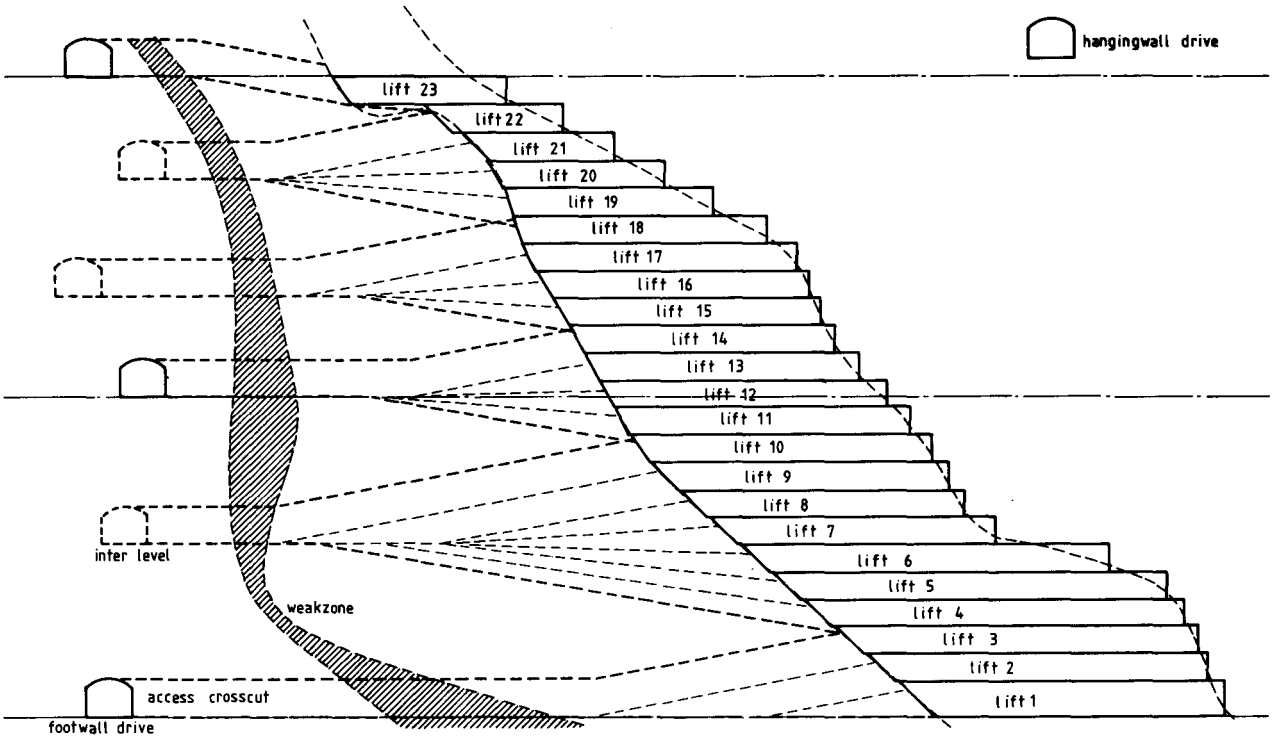

Fig. 9—Generalized section showing the proposed lifts and interlevel development.

vertical-drop ore silos were planned to link the interlevel footwall drives with the lower main-level footwall drive via drawpoint crosscuts for the transfer of ore off the interlevels.

Production Method of a Cut-and-Fill Stope

Geological information had indicated that, in the proposed cut-and-fill areas, the horizontal ore widths in general did not exceed 30 m. At that width, a longitudinal, as opposed to a transverse, cut-and-fill method, was developed. As it is not envisaged that a transverse method will be used in the extraction of the Broken Hill orebody, it is not discussed in this paper.

Three cut-and-fill stopes (8/800, 10/700, and 11/600) have been silled, and each is producing from 16 to 24 kt of ore per month. The earliest cut-and-fill production came from 8/800 stope during May 1981, and to date five and seven lifts have been mined on the western and eastern sides respectively. The 10/700 cut-and-fill stope started production in January 1982, and four lifts have been completed. The 11/600 cut-and-fill stope commenced in September 1982 and has completed two lifts. The 8/1100 cut-and-fill stope was due to come into full production during August 1983. The total tonnage removed from cut-and-fill stopes since their commencement in 1981 is 600 kt (Fig. 1).

The sequence for extraction consists initially in the silling out of the 4 m high sill drive over the entire extent of the orebody on lift one. Through-ventilation must be established with stope ventilation raises. This cut is then filled with 3 m of cemented backfill, allowing a space of 1 m between the fill surface and the back for the passage of ventilating air through the stope. Access is gained to lift two by the slashing down of the hanging wall of the stope-access crosscuts, the sliped waste forming an artificial ramp on which the equipment can climb. The successive lifts are mined *en echelon*, the stope access being used for two lifts and then the access from the interlevel above. Blasted ore from lift one and two is trammed to the station orepasses, ore from successive lifts being trammed to the drop raises.

Silling Method

The silling operation (Fig. 5) of the 11/600 cut-and-fill stope is described, but the procedure was similar for the 8/800 and 10/1000 cut-and-fill stopes.

Sliping of the two access crosscuts and the sill drive was achieved by means of a Tamrock two-boom Pneumatic Paramatic drill rig, which drilled a 3,8 m round with button bits of 51 mm diameter. Faces of 10 to 20 m were established on either side of the original sill drive. The entire sill, as laid out from the diamond-drilling information, was silled out to the height of the original sill drive (4 m). After mining to the geological outlines, additional payable areas of ore were searched for in the footwall and hanging wall.

The drilling pattern was essentially a sliping round, four lines of blastholes were drilled parallel to the roof or back and the floor. The back holes had a reduced spacing of 0,8 m, and the remaining holes had burdens and spacing of 1 to 1,2 m. The back holes were charged with 20 by 100 mm 60 per cent gelignite cartridges placed inside plastic Omega tubes, these charges being connected by detonating cord. The remaining face holes were changed to within 1 m of their collars with 580 by 38 mm 60 per cent Dynagel cartridges. The holes were bottom-primed with short-period-delay electric detonators, and the round was initiated from surface with the centralized blasting system. Blasting was achieved on two shifts with blasting taking place at the end of shift: day shift 07h00 to 15h00, and night

shift 19h00 to 03h00. Fragmentation was generally good, and has improved since the use of electric blasting instead of safety fuse. The blasted ore was mucked out by the use of dipping fractures. Where a mining width exceeded 25 m, a 10 by 10 m pre-drilled pillar was left *in situ*. This was recovered before the sill cut was filled. Calculations showed that, at a face width of 25 m, the stress field in the back changed and a state of tension existed.

Filling of the Sills

A sill pillar is formed below the sill cut, and the stress levels in the sill pillar increase as the cut-and-fill stope below progresses upwards. Additional measures were taken when the sill cut was backfilled.

Prior to filling, a layer of 8-gauge mesh was laid on the floor of the stope. It was considered that the mesh would not assist in the overall stability of the cemented backfill but would perhaps consolidate the blocks of backfill, which would be fractured owing to a build-up of stress and to adjacent blasting operations.

A 2 m layer of high-cement fill was poured. This consisted of a 50:50 sand-and-tailings slurry mixed with Portland cement in the ratio 1:6. The unconfined compressive strength of this fill was higher than 2 MPa, and it was considered that it would greatly assist in the overall stability of the sill pillar during its extraction stage. A 1 m thick capping was placed on top of the base layer, consisting of classified tailings and cement in a ratio of 10:1. This strong capping prevented blasted ore from penetrating the fill, and also prevented dilution caused by the breaking and mucking out of fill by heavy equipment. Approximately 2 per cent by mass of cemented backfill was removed during the extraction of the second lift. Details of the filling operations are given later.

Lift Two and Successive Lifts

After the backfill had been placed, its surface elevation was surveyed as an indication of the possible amount of capping that could be removed as the next lift was mined out.

Access was gained to lift two by the sliping of 3 m of hanging wall from each access crosscut, the drill rig using the blasted rock for a ramp. Additional grouted rebars were installed in the ramp-access hanging wall. A box cut, 4 by 4 m, was developed across the orebody from both access ramps, and the box cut was mined from the geological footwall to the hanging wall as defined by the lift layout. From each box cut a 3,8 m sliping round was blasted to form two faces on each side, which gave a total of four faces. These faces were mined towards the central, eastern, raises. The ventilation requirements were based on the quantity of air that was needed to dilute the noxious gases caused by the mucking and other equipment. Calculation showed that 0,116 m^3/s of ventilation per kilowatt of engine power was required to dilute the noxious gases to within the legal safe limit. A 196 kW engine of a Wagner ST8 LHD required 23 m^3/s of ventilation to dilute the noxious gases to within the legal limits.

The ventilation raises were drilled with a 61R Robbins raise-boring machine. The shallowest hole that this machine can comfortably ream is at 45°; therefore, where the dip of the orebody was very shallow and a raise-bored hole could not be kept in the orebody, a downcast system of ventilation was used.

Production Method

The breast face had a stoping width of 4 m, that is 4 m from the fill surface to the back. This face consisted of 3 m of ore (the lift height) and a 1 m space for ventilation. The back of the stope was maintained at the correct elevation by means of grade lines.

The drilled blasthole pattern was basically a sliping round of 3,8 m, which was drilled with an Atlas Copco Buffalo rig. The rig had two booms, each fitted with a hydraulic COP 1038 drifter. Three lines of 51 mm-diameter blastholes were drilled, and the roof holes were drilled at a spacing of 0,75 m, with a second and third row of holes drilled with a burden and spacing of 1 m and 1,5 m respectively. The roof holes were charged with 20 by 100 mm 60 per cent gelignite cartridges spaced at 100 mm intervals in Omega Clips, each cartridge being connected with detonating cord. The face holes were charged with 560 by 38 mm diameter 60 per cent Dynagel cartridges. Each hole was bottom-primed with a short-period-delay electric detonator. Sequential detonators were used to form a chevron-type blast. Blasting was achieved on day shift and night shift, with a re-entry period of 4 hours between shifts.

A Tamrock automatic roofbolting rig maintained the roof support by installing sixty 2,3 m by 16 mm fully grouted rebars each day in a regular 2 by 2 m pattern. The bolting is achieved on alternate faces without interfering with the production cycle.

The extraction of the second lift was completed and the

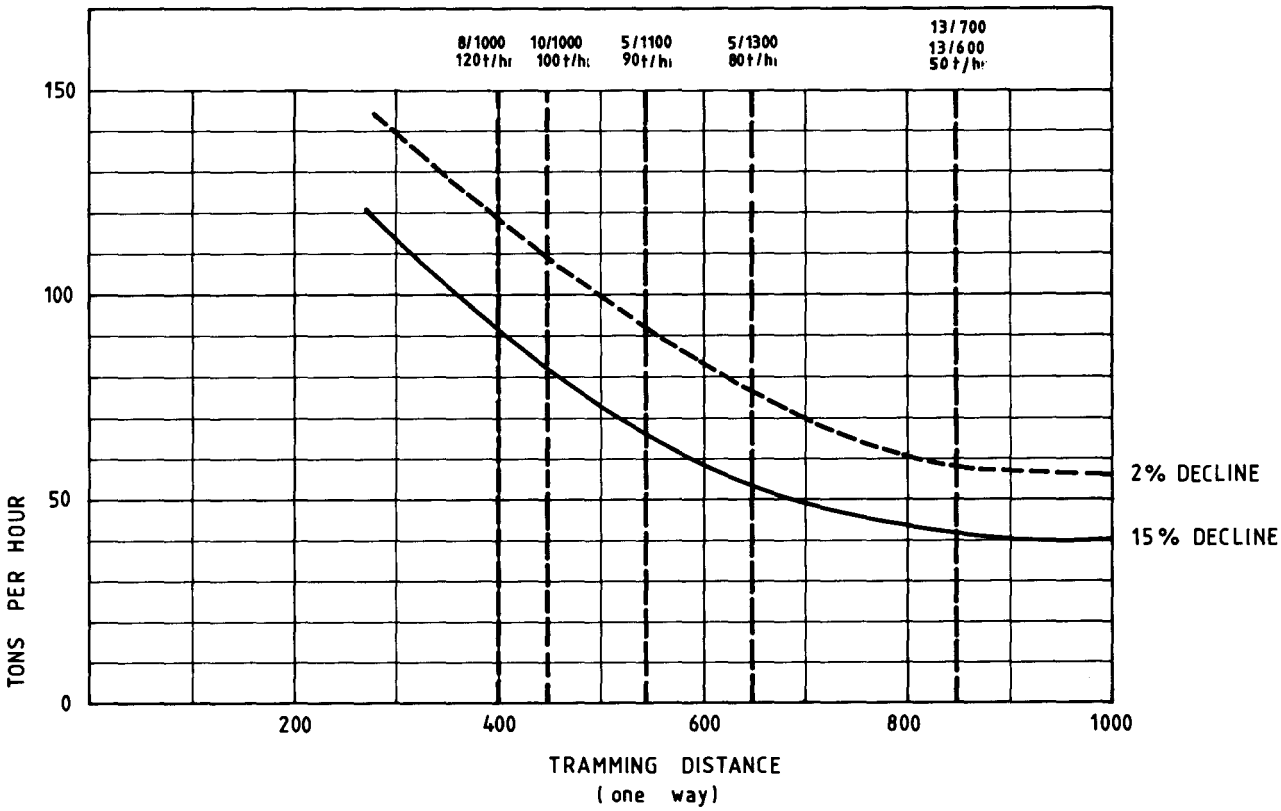

Fig. 10—Decline of output with tramming distance for a Wagner MT 425 diesel-powered haultruck.

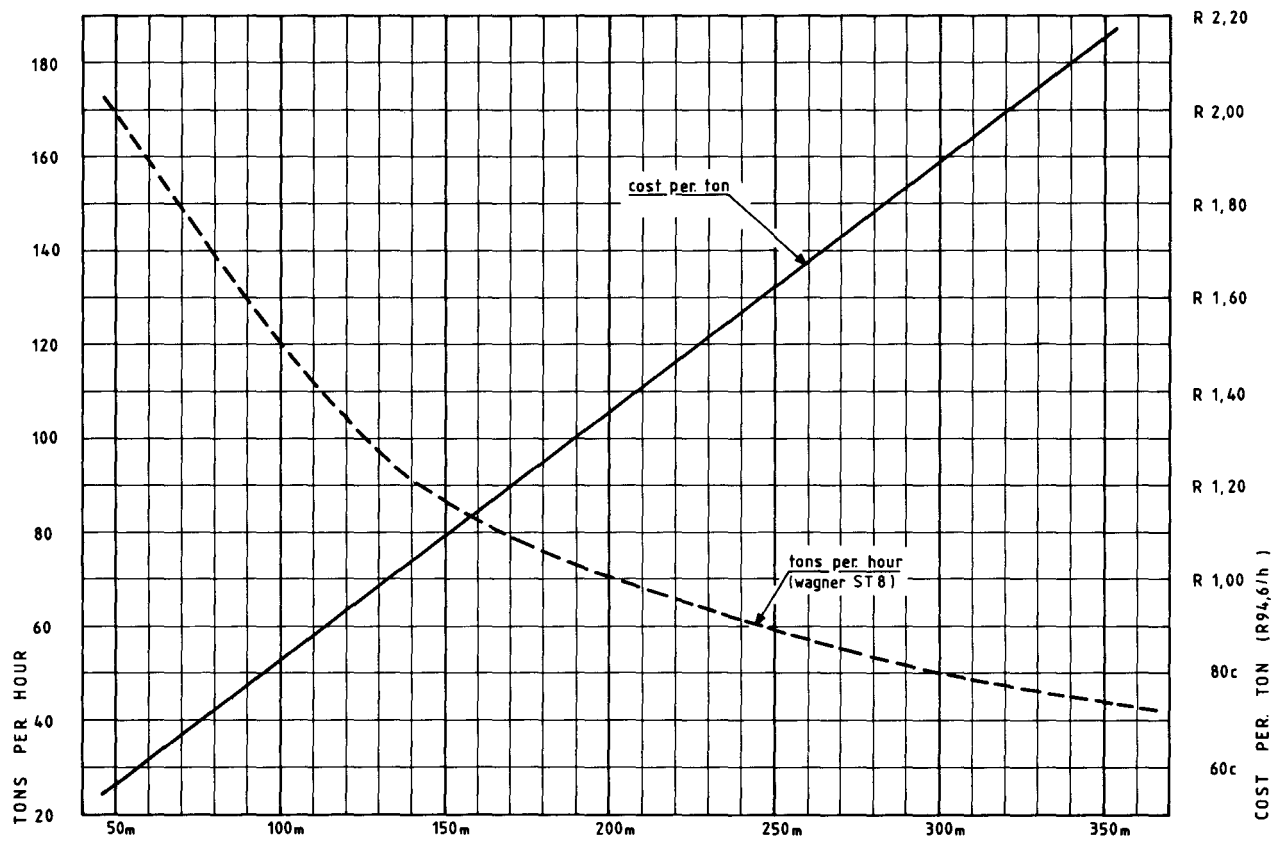

Fig. 11—Cost and output of a Wagner ST8 load–haul–dump truck with distance.

third lift commenced, access being gained from the inter-level, which splits the main level spacing of 35 m.

Wagner MT 425 25 t diesel-powered haultrucks will be used to assist in cleaning the blasted ore from lift three. The tonnage output for haultrucks reduces with the distance trammed. The relationship of tonnage to distance is shown in Figs. 10 and 11. To facilitate the use of haultrucks in the stopes, the 3 m lift will be increased to 4 m, giving a floor-to-back height of 5 m. It should be noted that the increase in lift height should reduce the overall production costs since fewer roofbolts are installed and fewer backfill cappings have to be placed.

Cyclic Mining Sequence

As the cut-and-fill mining at Black Mountain is a highly mechanized, equipment-intensive mining method, it is vitally important that a well-planned cyclic mining sequence should be followed. The tabulation-cyclic mining shows an average, and not an idealized, cycle. The ideal cycle would be one in which, as one side of the stope is near depletion, the other side has been filled and is ready to produce again, with no delays in backfilling. In practice, deviations occur from the idealized production cycle. The reasons for this are that equiment cannot always be replaced while it is being serviced, and that there are periods of engineering downtime due to breakdowns.

While backfilling of the blasthole stopes and pillars is in progress, there is a strict extraction sequence and tonnage requirement. Occasionally a shift may be lost during backfilling because a blasthole pillar or stope requires urgent completion. Another cause for delays is the time required to blast and clean the occasional tongue of ore that runs into the hanging wall. These tongues of ore usually yield relatively small tonnages, but can delay the filling of a stope by as much as 12 shifts.

Average Cut-and-Fill Production Cycle

As shown in Fig. 12, if one face is filled and the other has a small amount of remaining tonnage, the average lift tonnage is 54 kt and requires 120 shifts to be placed.

Backfilling Methods and Developments

The fill in cut-and-fill mining is used for several reasons: it provides a working surface from which the machinery and operators can extract the ore; it supports the hanging wall and, to a lesser extent, the footwall; and it enables maximum extraction of ore to take place.

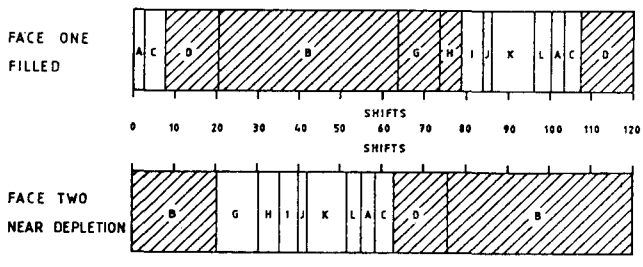

Fig. 12—Average cut-and-fill production cycle. A=Wait for capping to set; B=Produce ore; C=Slipe and bolt hanging wall of ramp access; D=Slipe and bolt box cut; E=Develop box cut; F=Examine depleted stope for payable tongues of ore; G=Blast and clean tongues of ore; H=Final clean up of stope; I=Install drain towers and barricades; J=Delay for backfill; K=Place base layer of fill; L=Place capping. The hatched areas indicate periods of ore production.

Various materials are available for backfilling purposes at Black Mountain Mine. Waste or blasted quarry rock is not used since there is no economic source of this material. Sand is available in two forms: abundant dune sand, and river sand from dry river beds. Classified mill tailings at a rate of 60 kt per month are also available.

The Backfill Material

With reference to particle size, the fill materials vary considerably. Prior to the placement of backfill in the cut-and-fill stopes, investigations suggested that the percolation rate of dune sand, unlike that of the classified tailings, would make it a suitable material for uncemented fill. However, when placed in large quantities in the stopes, the material segregated and formed large areas that remained liquefied and did not drain. Therefore, it was essential that a backfill material should be found that would support heavy equipment and prevent loss of ore, and that would be justifiably economic. The percolation rate for river sand was 0,25 mm/s, for deslimed mill tailings 0,0005 mm/s, and for dune sand 0,0006 mm/s (Fig. 13). Experience has shown that uncemented fill requires a minimum percolation rate of 0,01 mm/s for it to be suitable as a bulk fill without the danger of liquefaction or the formation of perched water tables.

At present, a 2,25 m base layer of cemented backfill is placed. This layer consists of slurry and Portland cement in a ratio (by mass) of 30:1, the slurry comprising half dune sand and half mine classified tailings. Within 24 hours this fill becomes strong enough to walk on, but not strong enough for the heavy mucking equipment to load on unless intolerable amounts of fill are removed. The unconfined compressive strength of this base layer is 200 to 400 kPa. To provide a strong surface for the equipment, a capping with a thickness of 0,75 m is placed. This capping consists of classified mill tailings and Portland cement in a ratio of 10:1. Very little dilution is caused by loss of capping, less than 2 per cent by mass of the capping being removed by the mucking equiment.

Deposits of river sand in extinct river beds have been found in close proximity to the Mine. The characteristics of this sand make it suitable for uncemented backfilling, and recently a cut-and-fill stope had its base layer filled with this uncemented river sand. It drained well, and quickly formed a competent surface on which a 0,75 m layer or normal cemented capping was placed. A large cost saving was made: 30-to-1 cemented backfill has a total cost of R5 per ton placed, and river sand R3 per ton placed. It is envisaged that all bulk filling of the base layers will eventually employ river sand.

Pumping Characteristics and Flowrates

Table I indicates the flowrates, percentage solids, and velocities in a pipe of 150 mm inside diameter, together with the density of various fills.

Minimum pumping velocities were calculated for the materials. River sand requires a higher velocity to keep the larger particles, 19 mm material, in suspension. When river sand is pumped, there is evidence of a sliding bed occurring

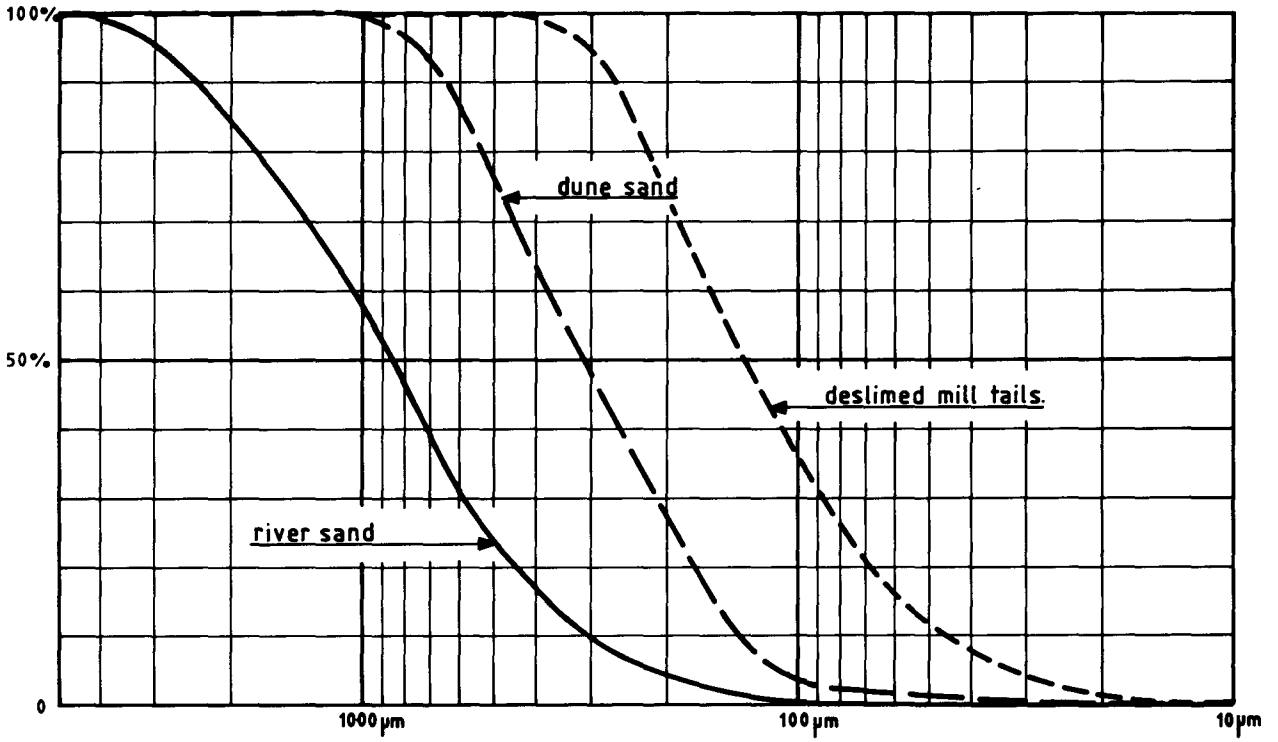

Fig. 13—Size analysis of fill material (% passing)

Material	d_{50} µm	CU	Perc. rate mm/s
Deslimed mill tailings	120	3,29	0,0005
Dune sand	320	2,82	0,0006
River sand	860	3,36	0,2500

d_{50}=size passing 50% by mass
CU=coefficient of uniformity d_{60}/d_{10}

TABLE I
FLOWRATE, SOLIDS CONTENT, AND VELOCITY OF VARIOUS BACKFILLS

Type of fill	Total cost per t placed R	Unconfined compressive strength 7 days MPa	Density of fill ($\times 10^3$ kg/m^3)	Solids %	Solids pumped t/h	Velocity in pipe m/s	Moisture content 7 days %
River sand	3	0,02	1,6	69	220	3	11
Classified tailings and dune sand 50:50 with 1:30 Portland cement	5	0,02	1,9	70	400	3	20-25
Classified tailings 1:10 Portland cement	12	≈1	2,05	73	250	2	20-25
Classified tailings and dune sand 50:50 with 1:6 Portland cement	19	≈2	2,0	73	400-450	3	20-25

in the flat sections, which will considerably increase pipe wear. However, if a high relative density can be maintained, the wear rate appears to decrease. To date a minimal amount of wear has been observed.

The Backfill Plant

The designed backfill-mixing plant is situated in the shaft area so that it can discharge directly to the near-vertical boreholes leading from surface to the various levels underground, without any significant pumping of the final backfill material. The design of the plant was based on an initial requirement of 2,5 kt of solids per day. Up to 7 kt of solids have been produced in two 8-hour shifts.

Underground Reticulation and Placement System for Backfill

From surface, pairs of fill holes 165 mm in diameter were drilled to 3, 4, 5, and 6 Levels. At the base of each hole, a puddle pipe is grouted into place and pinned into position with two lengths of rebar. Short lengths of flexible-rubber armoured hose are used to connect the puddle pipes to high-density polyethylene fill pipes of 150 mm inside diameter, and to two holes from 6 Level to the deeper sections of the mine.

For cut-and-fill stopes, the fill-pipe column is brought down the stope access ramp from the level above, and pipes are extended to the east and west. As fill is required, the pipes are lifted, broken, or swung from place to place to ensure that filling is carried out to the correct elevation. To date, the greatest length of backfill transport has been 350 m horizontally, with a 12 m increase in vertical head.

The method used to place backfill depends on the material being used for the fill. River sand has been found to form steep beaches, while tailings, or sand and tailings, spread around the stope. Pig-tail eyebolts are installed to form suspension points for the fill pipes. Surveyors give the grades to which the base and the capping are to be poured, and these lines are extended round the stope to give the fill crews a reference point.

Drainage of Backfill

Water drains from the backfill by percolation and decantation, the former in cemented slurries accounting for only 20 per cent of the water leaving the fill. For river sand, the ratio of percolation to decantation is higher. It is therefore essential that both types of drainage should be provided for.

It is vitally important that the drainage should be good since the unconfined compressive strength of a material is proportional to its final moisture content. Over-hydration of the Portland cement can occur if large amounts of water are present (Fig. 15).

Before filling starts, a series of drain towers is installed at 30 m centres in the stope. The towers consist of a steel lattice structure, usually discarded drill steel, covered with weldmesh having 100 mm openings and with coconut matting. Various types of drainage cloths have been tried, and the drainage rates and solids losses have been measured. So far coconut matting has been found to be the best economically, and achieves good drainage rates without serious blinding and with minimal losses of cement or fines.

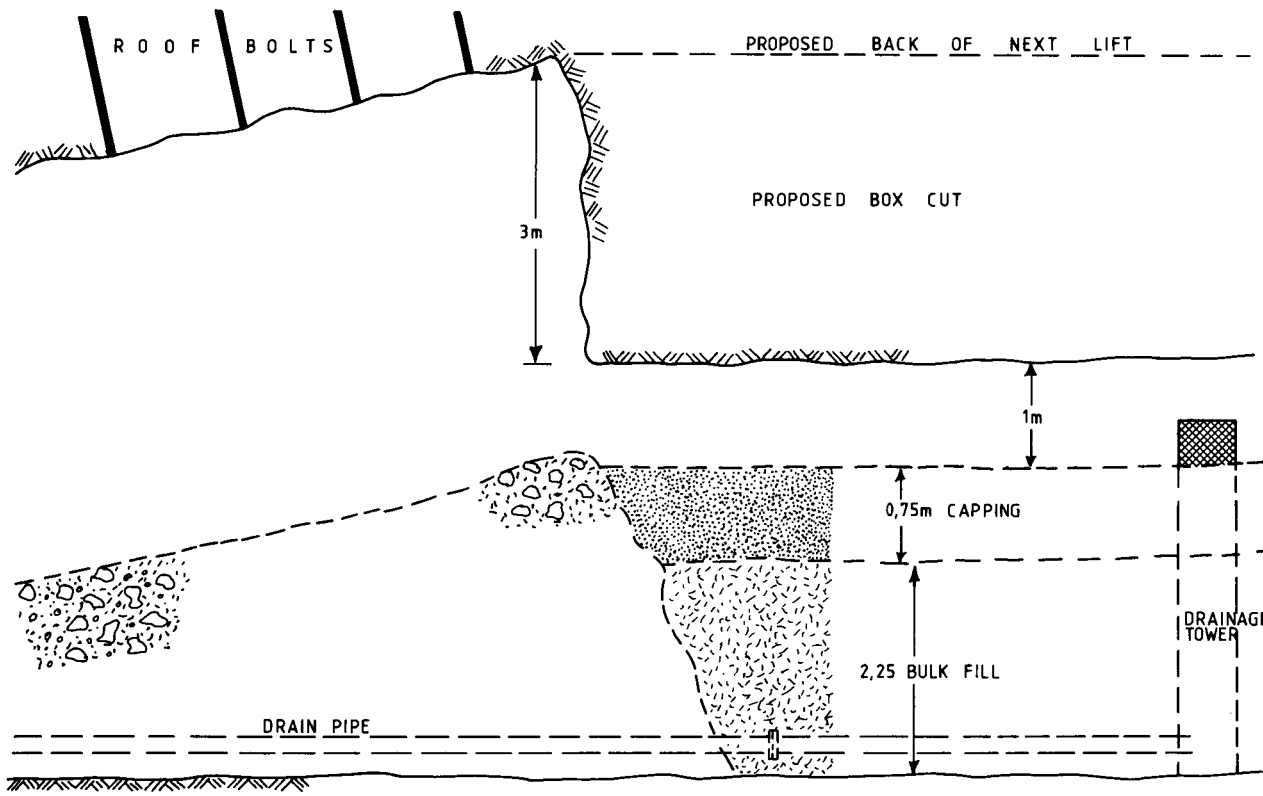

Fig. 14—Cut-and-fill waste barricade.

Backfill Barricades

Two types of drainage barricades are used in the stope access ramp: the waste wall, and the barricade of timber, mesh, and cloth. The waste wall (Fig. 14) comprises development waste from another area or waste from a hanging wall when it is blasted for the preparation of the next lift. Prior to blasting or placing of the waste, discard pipes 150 mm in diameter are laid down on the footwall to ensure that the drainage water will pass through. The other barricade consists of three mine poles 200 mm in diameter fastened at either end by means of chain and eyebolts. Against the mine poles is attached a layer of weldmesh having 100 mm openings on which a layer of coconut matting is placed. The timber barricade has a calculated strength of 18 kPa, which correspsonds to a hydraulic head of 1,5 m. Therefore, to ensure that an acceptable factor of safety is maintained, it is imperative that the daily pumping rate does not increase the level of the fill by more than 1 m. This is provided that stabilization of the backfill is quick enough and that drainage of the backfill material is adequate.

Effect of Moisture on the Backfill

During the testing of fill samples taken from the second-stage mixing tank, it became apparent that the moisture content, and therefore the drainage of the fill, has a considerable effect on the unconfined compressive strength (Fig. 15). This effect has been seen underground where poorly drained fills in cut-and-fill stopes have given low drop-cone penetrometer values.

Testing and Quality Control of Backfill

At the backfill plant, the densities, flowrates, and ratios of component materials are monitored automatically. Daily random samples of backfill are taken from the second-stage mixer, and values of unconfined compressive strength and moisture contents are obtained at 24 hours, 48 hours, 7 days, and 28 days.

After the fill capping has been placed, the elevation is surveyed so that the amount of material removed as a result of the new cut can be evaluated. A period of 48 hours is normally left after filling before the first blast is taken in the access crosscut. The unconfined compressive strength of the capping is checked randomly by means of a simple drop-cone penetrometer.

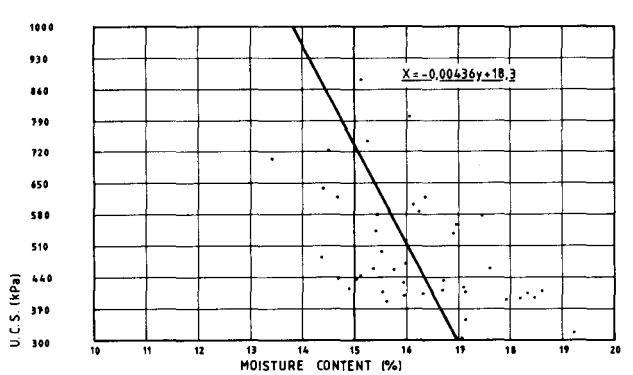

Fig. 15—Effect of moisture on the 28-day unconfined compressive strength of a filling consisting of a tailings–sand slurry and Portland cement in the ratio 20:1 by mass.

Cut-and-Fill Equipment

The mucking equipment in the cut-and-fill stopes consists of Wagner ST8 machines, which were originally used for tramming ore from blasthole drawpoints to the nearby orepasses. The Mine has a fleet of eight ST8s, and at present four are being used in cut-and-fill, the remainder being used in blasthole, training, and major overhauls. Their availability and utilization is 95 per cent and 90 per cent respectively, with an average of 63 t per hour trammed. The distance of cut-and-fill stopes from the main orepass system is considerably greater than for the blasthole stopes. The economic limit for the Wagner ST8 machine is approximately 200 to 250 m, depending on the grades and the roadway conditions; at greater distances, a pair of Wagner MT425 25 t haultrucks is used.

Tamrock drilling equipment was purchased initially for the cut-and-fill stopes, but a Copco Electro Buffalo two-boom rig fitted with hydraulic COP 1038 drifters has now been purchased. The utilization and availability for the cut-and-fill production drilling equipment are 60 per cent and 85 per cent respectively.

A Tamrock automatic roofbolt rig fitted with a Tamrock HL438 hydraulic drifter is used solely for installing roof or back support in the cut-and-fill stopes. This rig drills holes for and installs 30 grouted rebars each shift.

Each cut-and-fill stope has a Rocor Rand scissors truck. These diesel-driven units have an elevating platform and are used to assist in examination, making-safe, and charging-up procedures.

Conclusion

Although cut-and-fill mining as at Black Mountain is a relatively new method, a most productive system has been developed. For this highly mechanized mining system, an extensive training system has been introduced to enable all the mining staff to become technically proficient.

Project teams are actively involved in evaluating existing techniques, and new methods are continually being evolved to make the mining system as productive and competitive as possible.

Acknowledgement

The author thanks the Consulting Engineer and Management of Black Mountain Mineral Development Company (Pty) Limited for permission to publish this paper, and the staff of the Services Department for assistance in its preparation.

Bibliography

DE JONGH, C.L. Design parameters used and backfill materials selected for a new base metal mine in the Republic of South Africa. Symposium on the Application of Rock Mechanics to Cut and Fill Mining, Sweden, June 1980.

ROBERTZE, G.J. The blasthole stoping method. Johannesburg, Association of Mine Managers of South Africa, *Papers and Discussions* 1980–1981.

ROBERTZE, G.J. Cemented hydraulic backfill for the Broken Hill ore body. *Ibid.*

ROSS-WATT, D.A.J. Stope pillar extraction at Black Mountain using backfill. Johannesburg, Association of Mine Managers, *Circular* no. 4/82.