CYPRUS ANVIL MINING CORPORATION

VANCOUVER BRITISH COLUMBIA

DY PROJECT

PRELIMINARY STUDY FOR UNDERGROUND EXPLORATION AND MINING

PROJECT 945 MAY 1978

WRIGHT ENGINEERS LIMITED
Vancouver Canada

WRIGHT ENGINEERS LIMITED

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1444 Alberni Street, Vancouver, British Columbia, Canada, V6G 2Z4

Project No. 945-100

May 3, 1978.

Cyprus Anvil Mining Corporation,
330-355 Burrard Street,
Vancouver, B.C.
V6C 2G8

Attention: Mr. J. Olk

Gentlemen:

As authorized by you, we are pleased to submit herewith our report entitled:

CYPRUS ANVIL MINING CORPORATION
VANCOUVER, B. C.

DY PROJECT

PRELIMINARY STUDY FOR UNDERGROUND
EXPLORATION AND MINING

The concepts and cost estimates presented are as complete and accurate as the limited time and information would allow but should be sufficiently accurate for a preliminary report of this nature.

We appreciate your entrusting this study to us and would welcome an opportunity to collaborate on any further engineering requirements.

Yours very truly,

WRIGHT ENGINEERS LIMITED

N.R. Krpan, P.Eng.,
Project Manager.

NRK/sd
Enc.

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INTRODUCTION

Cyprus Anvil's Dy property is approximately 12 miles southeast of Anvil's open pit. The Dy mineralized zone has been explored by surface diamond drilling with 10,000 metres in 14 holes completed to date. A target area of 200 metres x 900 metres located approximately 700 metres below surface has been selected for more detailed investigation. This area contains a relatively flat lying deposit which may have up to 15 million tons of 17% combined lead-zinc and 38 gm per ton silver ore.

Because of the ore depth, it appears that further investigation can best be accomplished by establishing underground access for diamond drilling and bulk sampling.

Cyprus Anvil Mining Corporation has engaged Wright Engineers Limited to estimate costs for an underground exploration programme and to prepare a preliminary mining study for a 3,000 tons per day underground mine.

TERMS OF REFERENCE

A meeting was held on February 13th, 1978, at Cyprus Anvil's Vancouver office, with Mr. J. Olk, Mr. G. Davis, Mr. G. Simpson, of Cyprus Anvil, and Mr. C.T. Penney and Mr. N. Krpan, of Wright Engineers Limited, to discuss the geological and mining aspects of the Dy deposit. At this meeting Mr. Olk and Mr. Simpson gave a general outline of the scope of work and geology. Wright Engineers Limited were asked to conduct a preliminary mining study for a 3,000 ton per day underground mine. This study would include conceptual planning and order of magnitude cost estimates for underground exploration, development and production. Cost estimates would be for ore in the headframe bin and exclude surface transportation and milling.

SECTION II

SUMMARY

SECTION II

SUMMARY

GENERAL

The exploration programme consists of a 715 metre exploration shaft, 1,230 metres of lateral development and 2,350 metres of diamond drilling. Drawing No. 945-100-1203 illustrates the exploration programme.

There are only two drill holes intersecting the target ore zone. The mining concepts were therefore based on very limited drill information, and unconfirmed projections and interpretations of the ore structure. The most practical approach to mining as presently envisaged includes shaft access with room and pillar and cut and fill mining methods. Drawing No. 945-100-1204 and 1205 illustrate this approach.

Following completion of the current drilling programme, sufficient geological information may become available to justify a preliminary mining feasibility study. It is recommended that during this stage a more detailed analysis of mine development and production methods be implemented.

CAPITAL COSTS

The order of magnitude costs for underground exploration and mining are as follows:

Exploration Shaft	$3,896,000
Lateral Development - Exploration	1,538,000
Diamond Drilling	164,000
Sub-Total	$5,598,000
Capital cost for Development and Equipment for Production at 3,000 TPD ore (includes costs of above exploration work)	$31,183,000
Allowance for equipment replacement over 15 year mine life	$ 8,700,000

Does not include Salvage 15% total

Table 2 is a Schedule of the Capital and Operating Costs.

OPERATING COSTS

The operating costs are based on an estimated production performance of 12.8 tons per man shift.

Operating cost per year for 780,000 tons at $15.70 is approximately $12,300,000.

Room + pillar
If possible
could go
to 20 T/MS

SECTION III

EXPLORATION PROGRAMME

GENERAL - SHAFT VERSUS DECLINE

The footwall of the main ore zone is approximately 700 metres below surface. Access for underground diamond drilling and bulk sampling can be obtained by shaft sinking or by decline developed with trackless diesel equipment.

It is desirable to situate this initial heading so that maximum utilization can be obtained in the event of a production decision.

A decline could be collared in relatively shallow overburden at the 900 metre contour, approximately 900 metres southeast of the ore zone. The length of drive at -15% to reach the ore footwall at the 400 metre elevation would be 3,333 metres. This positioning is advantageous for service and ventilation for future production. The decline could be shortened to 2,666 metres by intersecting the projected ore zone at the 500 metre elevation. However, to be useful for future production, it would require eventual extension outside the ore zone to the footwall.

A shaft collared in shallow overburden approximately 125 metres south of the assumed ore contact could provide access for exploration and could serve for future service, ventilation and possibly production. Shaft depth to serve for exploration is 715 metres. This would be extended to 780 metres if a decision is made to produce ore. Alternatively the shaft could be positioned in the area of hole 77X-06 and serve almost exclusively for exploration. It could provide some future use for ventilation and possibly a secondary means of egress. The depth of shaft in this location would be 580 metres.

An order of magnitude cost comparison for the two methods of access and related factors are as follows:

COST COMPARISON FOR METHODS OF ACCESS

	Shaft		Ramp	
a) Exclusively for exploration with limited future use for ventilation and secondary egress.	580 m @ $5,450	$3,161,000	2,666 m @ $1,310	$3,493,000
b) Located for exploration and potential future use in production.	715 m @ $5,450	$3,896,000	3,333 m @ $1,310	$4,366,000
c) Extended to serve for future production	780 m	$4,115,000	3,333 m @ $1,310	$4,366,000
d) Additional surface to orebody development required subsequent to full production.				
Fresh Air Raise	700 m @ $ 800 =	$ 560,000	700 m @ $ 800 =	$ 560,000
Return Air Raise	700 m @ $ 800 =	$ 560,000	Use Ramp	
Production and Service Shaft	Use Exploration Shaft		780 m	$4,115,000
Timber Fresh Air Raise for Escapeway	700 m @ $ 200 =	$ 140,000		
	Sub-Total =	$1,260,000		$4,675,000

Note: Drawing No. 945-100-1201 illustrates above development schematically.

The preliminary evaluation and comparison of the two methods of access show economic advantages for a shaft over a decline system for both exploration and production. A shaft system has therefore been selected for this conceptual plan.

SHAFT SINKING

Shaft sinking is most commonly undertaken by specialized mining contractors who have the experienced labour and necessary equipment such as shaft muckers and drilling equipment. The Client Company usually provides supplies and materials, including explosives, pipe, timber and may even provide some plant equipment such as power plant, hoists and compressors.

The shaft configuration and size is dependent on shaft purpose, depth and ground conditions. Exploration shafts of depth less than 300 metres may consist of a single hoisting compartment and a manway-pipe compartment and may be approximately 2.5 m x 3.75 m in cross-section. Shaft lining of timber or concrete is dictated by nature of surrounding rock.

Shafts deeper than 300 metres are usually driven with two hoisting compartments plus a manway-pipe compartment. Even though more material must be excavated and more timber or concrete placed, a higher rate of advance makes the cost only marginally more or even competitive with the smaller sized shaft.

It is recommended that three compartment timbered shafts of 2.3 m x 6.5 m in cross-section be considered for access to the orebody. This configuration is illustrated in Drawing No. 945-100-1202. Hoisting compartments are of sufficient size for lowering diesel LHD equipment used in lateral development and possible subsequent production. A double drum hoist would be used for the sinking and for removing muck from lateral exploration drives.

COST SUMMARY - EXPLORATION SHAFT

CONTRACTOR

Mobilization and Demobilization	$ 69,000
Sink Collar to 25 metres	46,000
Labour for sinking 690 metres	993,000
Station Excavation	65,000
Contractor's Plant Rental	150,000
Sub-Total	$1,323,000
Contractor's Mark-up on sub-total 15%	198,000
CONTRACTOR'S TOTAL	$1,521,000

CYPRUS ANVIL

Materials and Supplies	$1,221,000
Company Plant Purchases	$1,154,000
CYPRUS ANVIL TOTAL	$2,375,000

TOTAL - EXPLORATION SHAFT (Total Plant written off against Shaft)	$3,896,000

Some salvage value may be possible for Company plant equipment if the development does not advance beyond the exploration stage. However for purposes of this study, salvage value has not been included.

Total cost of the exploration shaft therefore is $3,896,000 or $5,450 per metre.

If a decision is made to proceed with stope development and production, Company plant equipment would be used to deepen the shaft and proceed with production.

The cost for a production shaft would therefore be $3,896,000 plus $219,000 cost for deepening from 715 m to 780 m. Deepening cost is estimated as follows:

Labour	$ 993,000 / 690 m x 65 m	=	$ 94,000
Mark-up @ 15%		=	$ 14,000
Materials and Supplies	$1,221,000 / 715 m x 65 m	=	$ 111,000
Total			$ 219,000
Total Cost for Production Shaft			$4,115,000

COST ESTIMATE - EXPLORATION SHAFT

CONTRACTOR

Mobilization and Demobilization:

Moving crews to site, setting up headframe and hoists, plus crane rentals and dismantling plant following completion of project.

2,600 man hours @ $15.00	=	$39,000	
Crane rental	=	$30,000	$ 69,000

Sink Collar to 25 Metres:

Labour - 8 men/shift, 2 shifts/day for 2 weeks.

Pay rate	$ 9.30	
Fringes	$ 1.60	(Campsite Expenses by Cyprus Anvil)
Incentive	$ 4.10	
Total	$15.00/hour	

8 men x 2 shifts x 8 hours x $15.00/hour x 12 days=	$ 23,040
Overtime allowance 10%	2,304
Sub-Total	$ 25,344
Crane Rental	$ 20,000
Total Collar - approximately	$ 46,000

Labour for Sinking:

35 men required for 21 shifts/week.

Labour cost per week

35 men x 8 hours x 5 days x 15.00/hour	= $ 21,000
Overtime allowance 10%	2,100
Sub-Total	$ 23,100

Depth of exploration shaft from collar
1,100 m elevation to 385 m elevation,
less 25 m for collar section = 690 m.

Average rate of advance = 16 metres/week

690/16 = 43 weeks required for sinking exploration shaft

Total Labour Cost - 43 x $23,100 approximately= $ 993,000

Station Excavation and Rock Pass:

Stations	6.5 m x 4.3 m x 30 m = 838 cu.m		
Loading pocket	6.5 m x 3.7 m x 3.7 m = 89		
	927 cu.m		
	927 cu.m. @ $46.00	=	$ 43,000
Rock pass	2.5 m x 2.5 m x 20 m @ $600.00	=	$ 12,000
Allowance for chute and rock pass controls		=	$ 10,000
Total			$ 65,000

Contractor's Plant Rental:

This will include the following:
Scaffold, cryderman and service winches, headframe and accessories, stage, wire rope for small winches, muck disposal gear, drill equipment.

Estimated Rentals $ 150,000

Contractor's Mark-up:

A 15% mark-up is allowed on
Contractor's labour and plant - .15 x $1,323,000 $ 198,000

CONTRACTOR'S TOTAL $1,521,000

CYPRUS ANVIL

Materials and Supplies:

Shaft timber, screen and hanging rods		$ 150,000
Drill steel, Explosives		$ 77,000
Concrete for Collar Section and shotcrete allowance for 20% of shaft footage		$ 144,000
Campsite Costs	1,750 man weeks @ $200.	$ 350,000
Power	50 weeks @ $10,000/week (Using $.07 per kWh for diesel generated power)	$ 500,000
Sub-Total		$1,221,000

Plant Equipment:

Buildings to include hoist house and its foundations, office and warehouse trailer, dry trailer, shops and compressor enclosure	$ 215,000
Hoists, compressors, fans, pumps, mobile equipment (dump truck, front-end loader, pick-up truck).	$ 395,000
Water supply, pipe, ropes and cables, miscellaneous electrics, workshop equipment.	$ 144,000
Diesel generator sets 3 units @ 600 kw each	$ 400,000
Sub-Total	$1,154,000
CYPRUS ANVIL TOTAL	$2,375,000
GRAND TOTAL FOR CONTRACTOR AND CYPRUS ANVIL	$3,896,000

LATERAL DEVELOPMENT - EXPLORATION

Proposed development consists of crosscutting from the exploration shaft into the orebody and longitudinally through the ore zone. Drawing No. 945-100-1203 illustrates this development.

Total footage and estimated costs:

a)	Crosscuts:		4.3 m x 4.3 m
		in waste	125 m
		in ore	250 m
b)	Longitudinally:		4.3 m x 4.3 m
		in ore	840 m
c)	Muck Bays:		
		in ore	15 m
	Total Lateral Development		1,230 m

Crosscutting and longitudinal drives in the ore zone would provide a bulk sample of approximately 20,000 cubic metres.

Estimated cost for lateral development, including hoisting and disposal of development muck on surface:

1,230 m @ $1,250/m = $1,538,000 approx.

DIAMOND DRILLING

Drawing No. 945-100-1203 illustrates the proposed underground diamond drill programme. The costs for this preliminary programme are based on a total of 2,350 metres drilled on five sections at 150 metre intervals.

Estimated costs:

Drilling \$60/m	=	\$	141,000
Camp Costs - 46 man weeks @ \$200 approx.	=	\$	9,000
Moving Costs		\$	14,000
Total		\$	164,000

SECTION IV

MINING

SECTION IV

MINING

GENERAL

A field trip was made to the Dy and Grum properties on April 11th, 1978 to investigate and attempt to predict the rock conditions which may be encountered in an underground mine.

Drill cores from Holes 77X-05 and 77X-06 intersecting the target ore zone were studied with particular emphasis on competency of rock in hanging wall of the ore. The hanging wall material in Hole 77X-05 is classified as 4C7. Table 1 describes classification code. This, plus the classifications 5B6 and 5B1 immediately above it, are competent rock types which could probably be bolted to provide a good stope back. However, in Hole 77X-06, 500 metres to the east, the hanging wall is extremely incompetent. This material classified as 4E1 and 4C0 is badly broken through 5 metres with pyrite sand present and loss of core recorded. It is suspected that this may be a local fault. Hole 78X-01 currently being drilled 300 metres west of Hole 77X-06, should provide further data on ore and hanging wall characteristics.

The Grum deposit, partially developed by an underground decline system, was toured to observe the existing ground conditions. Back and wall support for these declines was provided through roof bolting, strapping, screening and occasional shotcreting. Some caving occurred in weak graphitic phyllite during development of the declines. Generally the surrounding rock was well supported with bolts and strapping. Widths at drift intersections and at some drill stations were 12 metres or more with no sign of deterioration or caving on walls and back.

A considerable amount of water was noticed flowing in the Grum declines. The local contractor had grouted some of the drill holes during the exploration programme to decrease the flow. It would be advantageous to have the Dy drill holes grouted before moving the drill crews off the holes.

MINE DEVELOPMENT

Mine development and production plan is based on the premise that sufficient ore exists for a 780,000 ton per year production rate for 15 years. Daily production scheduled at 3,000 tons per day.

The vertical shaft used for exploration could be deepened and subsequently used for development, ventilation and production. Other preproduction development headings include footwall service and haulage drifts, vent raises and vent drifts, ore and rock passes, crushing station, loading pocket and pump station. These are shown in Plan and Section on Drawing No. 945-100-1204 and 945-100-1205.

The shaft would be collared approximately 125 metres south of the assumed ore contact. This location has several advantages including: minimum glacial till in collar section; positioning takes advantage of gravity for haulage of ore down the limbs of the ore structure; advantage of gravity for drainage to shaft pumping station; located in waste near the orebody's centre of gravity; no ore would be lost in shaft pillars.

It is proposed that vent raises to surface be driven by Alimak. Intermediate stations at 200 metre intervals in the shaft would provide access to the raises to maintain reasonable driving lengths.

Load-haul-dump equipment, trucks and electric hydraulic drill jumbos are recommended for all lateral and inclined drift development.

Cost, development and production schedules are illustrated in Table 2.

MINING METHODS

The footwall contact, approximately 15% on both the longitudinal limbs and in cross-section as illustrated on Drawing No. 945-100-1203, suggests that a trackless method of mining would be most practical.

Stoping methods which could be applicable include room and pillar, blasthole and cut and fill. With the limited amount of geological information presently available, it is not justifiable to investigate these methods in detail. The practicality of each method is dependent on the thickness and inclination of ore zones and the strength characteristics of both the ore and surrounding rock.

A preliminary investigation of the Dy drill core and review of geological sections indicate that most of the Dy deposit could be mined by open room and pillar methods. This could be modified to blasthole stoping for thick ore sections in competent ground. There are however, some sections, as demonstrated by weak hanging wall material in Hole 77X-06, which would require cut and fill mining.

Room and Pillar Mining

This concept is based on transverse rooms and pillars with dimensions 12 metres and 9 metres respectively, developed 200 metres across the orebody. Stope development would commence with a pilot drive along the hanging wall. This drive would then be slashed to full stope width and the back bolted and screened as required. The ore below the drive would be benched out to the footwall in 3 metre lifts with bolting and screening of stope walls following the bench advance. Pillar slashing or crosscutting could be undertaken where ground conditions permit to maximize ore recovery. Drawing No. 945-100-1206 shows general arrangement for this method.

Blasthole Mining

Where ore thickness exceeds 15 metres, it may be necessary to develop blasthole stopes. Access drifts and draw points would be developed primarily in waste. Blasthole methods could contribute towards higher productivity, however dilution for this method would be higher than room and pillar methods.

Drawing No. 945-100-1207 shows General Arrangement for this method.

Cut and Fill Mining

Ore sections as demonstrated by the weak hanging wall material in Hole 77X-06 would require cut and fill methods for successful mining.

Stope and pillar dimensions and access are similar to the room and pillar method. However the ore is mined from the footwall to hanging wall with 3 metre ore slices taken off the stope back. Cemented sand or mill tailings are piped to the stope and used for support. Tailings could be obtained from Anvil's existing mill and transported by returning ore trucks. The filling cycle follows mining so that it is possible to have equipment operate off backfill and be within reach of the stope back. Drawing No.945-100-1208 illustrates General Arrangment for this method.

MINING COSTS - OPERATING

The mining costs are based on estimated productivity factors for the various mining methods proposed. Also assumptions are made as to the extent of tonnage to be mined by each method.

Calculations:

Average underground labour rate	$ 9.30/hr.
Fringe benefits at 30%	$ 2.78
Bonus estimated at 30%	$ 2.78
Sub-Total	$14.86
Overtime allowance 10%	$ 1.49
Total Labour Cost per hour	$16.35

Total Labour Cost per shift: $16.35 x 8 = $130.80

Labour portion of the total cost is estimated to be 65%.

Therefore total cost - $\frac{\$130.80}{0.65}$ = $201.23

Mining methods and estimated performances which could be applicable include the following:

Room and Pillar	24 tons/man shift
Blasthole - Shrinkage	10 tons/man shift
Cut and Fill	7 tons/man shift

The thickness, width and competency of the ore zone and surrounding rock will dictate the method of mining. Examination of drill core and tour of the adjacent Grum workings indicate that Room and Pillar methods could be the most common method at the Dy deposit. However, there are indications of weak hanging wall material in the area of Hole 77X-06 which could require Cut and Fill methods. There is also a possibility that blasthole methods may be applicable.

The estimated distribution of mining methods is as follows:

Room and Pillar	60%
Blasthole	10%
Cut and Fill	30%

With this distribution, average mine performance for 780,000 tons per year, or 3,000 tons per day mining:

$$.60 \times 780{,}000 \times \frac{1}{24} = 19{,}500 \text{ man shifts}$$

$$.10 \times 780\ 000 \times \frac{1}{10} = 7{,}800$$

$$.30 \times 780{,}000 \times \frac{1}{7} = \underline{33{,}429}$$

$$780{,}000 \text{ tons} = \underline{60{,}729} \text{ man shifts}$$

Performance: <u>12.8 tons/man shift</u>

With cost per man shift at $201.23 and performance at 12.8 tons per man shift, cost per ton of ore -

$$\frac{\$201.23}{12.8} = \$\ \underline{15.72}$$

Operating Cost per year

$15.72 x 780,000 tons = approximately $<u>12,300,000</u>

MINING COSTS - CAPITAL

An order of magnitude cost for the mine has been developed on the basis of telephone quotations for major equipment and preliminary estimates for cost of rock work.

The estimated capital cost $31,183,000 consists of the following:

Surface Plant Equipment	$ 2,631,000
Permanent Underground Equipment	$ 1,075,000
Underground Service Equipment	$ 2,172,000
Mine Ventilation	$ 3,444,000
Permanent Underground Development	$21,861,000
Total	$31,183,000

Table No. 2 is a schedule listing this expenditure, plus replacement equipment for the mining period. The various categories of capital costs are further described as follows:

Surface Plant Equipment

Production Headframe	$ 250,000
Modify Sinking Hoist Installation	$ 100,000
Compressor House and Equipment	$ 370,000
Skips, Cages and Ropes	$ 100,000
Addition to Sinking Power Plant	$ 250,000
Surface Buildings	$ 1,035,000
Sub-Total	$ 2,105,000
Plus 10% Engineering and Supervision	$ 210,000
Plus 15% Miscellaneous and Contingencies	$ 316,000
Total	$ 2,631,000

Permanent Underground Equipment

Crusher Room and Crusher	$ 600,000
Pumping Station, Excavation, Equipment	$ 260,000
Sub-Total	$ 860,000
Plus 10% Engineering and Supervision	$ 86,000
Plus 15% Miscellaneous and Contingencies	$ 129,000
Total	$ 1,075,000

Underground Service Equipment

6	- Scooptrams (5 cu.yd.)	@ $110,000	$ 660,000
1	- 2-Boom Development Jumbos	@ $ 90,000	$ 90,000
3	- 3-Boom Production Jumbos	@ $110,000	$ 330,000
1	- Scaling and Bolting Rig	@ $ 90,000	$ 90,000
2	- 20-Ton Telescopic Dump Trucks	@ $120,000	$ 240,000
2	- Utility Vehicles	@ $ 40,000	$ 80,000
1	- Scissor-Lift Truck	@ $ 45,000	$ 45,000
1	- Mine Mobile	@ $ 22,000	$ 22,000
1	- Stope Drill Wagon	@ $ 30,000	$ 30,000
4	- Long-Hole Drills	@ $ 15,000	$ 60,000
18	- Stopers and Jacklegs	@ $ 2,600	$ 47,000
2	- Air Pumps	@ $ 1,500	$ 3,000
4	- Tugger Hoists	@ $ 4,000	$ 16,000
4	- Auxiliary Ventilation Fans	@ $ 1,000	$ 4,000
2	- 30 HP Slushers	@ $ 10,000	$ 20,000
Sub-Total			$ 1,737,000
Plus 10% Engineering and Supervision			$ 174,000
Plus 15% Miscellaneous and Contingencies			$ 261,000
Total			$ 2,172,000

Mine Ventilation

Ventilation Fans, Electrics and Installation		$ 235,000
Fresh Air Raise	700 m @ 800	$ 560,00
Return Air Raise	700 m @ 800	$ 560,000
Ventilation Drifts (3.6 m x 3.6 m)	1,400 m @ 1,000	$ 1,400,000
Sub-Total		$ 2,755,000
Plus 10% Engineering and Supervision		$ 276,000
Plus 15% Miscellaneous and Contingencies		$ 413,000
Total		$ 3,444,000

Permanent Underground Development

Shaft	$ 4,115,000
Exploration Headings and Underground Drilling	$ 1,702,000
Loading Pocket and Installation	$ 250,000
Ore Pass Development and Chutes	$ 350,000
Level Development	$11,072,000
Sub-Total	$17,489,000
Plus 10% Engineering and Supervision	$ 1,749,000
Plus 15% Miscellaneous and Contingencies	$ 2,623,000
Total	$21,861,000

MINEABLE RESERVES AND RECOVERIES

At the time of report preparation there were only two drill holes, 77X-05 and 77X-06 intersecting the proposed mining area. The data from these two holes plus geological interpretation provided by Cyprus Anvil geologists have been used for the development of conceptual mining plans.

The target mining area is approximately 200 m x 900 m and contains ore of variable thickness. For the purposes of this conceptual study, a mineable reserve of approximately 12 million tons (i.e. 780,000 tons per year for 15 years) is assumed to exist.

The grade of the mineable ore is based on assays from the two drill holes and from assumed dilution factors.

Estimated grade is as follows:

Hole No.	Pb %	Zn %	Ag gms/ton	Pb-Zn%	Intersection
77X-05	5.15	7.13	95.0	12.28	2.0
	6.05	8.67	119.0	14.72	2.0
	5.33	7.85	115.0	13.18	2.0
	3.83	6.31	83.0	10.14	1.0
Average	5.27	7.66	105.86	12.93	-
77X-06	8.23	8.24	14.7	16.47	1.9
	7.11	13.72	16.0	20.83	2.0
	4.76	11.15	8.0	15.91	2.0
	1.61	4.80	1.8	6.41	1.0
	10.49	18.99	18.0	29.48	1.4
	10.41	21.33	18.7	31.74	2.0
	8.76	21.20	18.0	29.96	0.9
	3.96	6.05	6.6	10.01	1.6
	4.11	11.99	8.4	16.10	2.0
	3.75	9.33	7.4	13.08	2.0
	7.81	14.23	16.0	22.06	1.3
Average	6.44	12.67	~~12.13~~ 121.3	19.11	-
77X-06 / 77X-05	6.11	11.27	38.27	17.39	

The recovery and dilution factors estimated for each mining method and for the total production are as follows:

RECOVERY AND DILUTION FACTORS

Mining Method	Proportion of Tonnage by Method	Tonnage by Method	Recovery Factor	Dilution Factor
Room & Pillar	60%	468,000	75%	5%
Blasthole	10%	78,000	85%	15%
Cut and Fill	30%	234,000	95%	5%
Total Per Year		780,000	80%	6%

Recovery in room and pillar mining is estimated to be 75%. This factor is based on 12 m and 9 m width dimensions for rooms and pillars and 9 m wide crosscuts at 21 m centers. Ground conditions and varying heights will result in varying recoveries for this method. The small post pillars left behind are assumed unrecoverable. Dilution from the hanging wall and footwall should be minimal because short hole drilling by jumbos and wagon drills should allow reasonable control of dilution. Estimated dilution factor = 5%.

In blasthole mining with delayed filling of stopes and pillar mining using the recently developed vertical crater retreat method should result in recoveries to 85%. The only loss of ore would be the unrecoverable tonnage between draw points and in isolated pockets or faulted zones. The proportion of ore loss is dependent on mining heights with the higher stopes giving higher recoveries. Dilution could be high because of irregularities in the ore zones, difficult control with long holes and a greater possibility of slough from the hanging wall. Estimated dilution factor = 15%.

The cut and fill mining method allows for very high recoveries. Only small isolated pockets of ore and faulted areas may be unrecoverable. Pillars are usually totally recoverable. Recovery is estimated to be 95%. Dilution is very low and could occur from extremely irregular ore boundaries and from adjacent filled stopes during pillar mining. Estimated dilution factor = 5%.

SECTION V

RECOMMENDATIONS

SECTION V

RECOMMENDATIONS

It is recommended that the geological information from the current exploration programme be used as a basis for a preliminary feasibility study.

The structure and competency of the ore and surrounding rock will dictate the eventual mining methods and hence the costs. Additional drill information plus geotechnical evaluation will allow for a more realistic assessment of stope-pillar dimensioning and mining methods.

A decision for any underground exploration should be accompanied by a test stoping programme to provide valuable geotechnical data for mine planning.

TABLES

DY DEPOSIT AREA

TABLE NO. 1

LITHOSTRATIGRAPHIC CODE

L. U. CRETACEOUS — INTRUSIVE ROCKS

Intrusive Rocks

Unit 11

11 A	Muscovite-biotite granodiorite	Anvil Batholith
B	Porphyritic, biotite-quartz monzonite	
C	Quartz monzonite pegmatite dikes	
D	Equigranular Hb-bio quartz diorite	Dioritic Suite Related to Anvil Batholith
E	Porphyritic Hb-bio quartz diorite	
F	Sticky quartz-feldspar porphyry	
G	Pyroxenite and serpentinized equivalents	
Q	Bull quartz vein, pod	
1	Gneissose LS - tectonite	Anvil Batholith Varieties
2	Gneissose L - tectonite	
3	Equigranular	
4	K - feldspar phenocrysts	
5	Garnet-quartz-monzonite	
6	Plagioclase phenocrysts	Diorite Porphyry Varieties
7	Plagioclase + biotite phenocrysts	
8	Plagioclase + biotite + hornblende phenocrysts	
9	Strongly altered	
0	Normal	

Intrusive Contact

KECHIKA GROUP

Vangorda Formation

Unit 5

5 A	Variably calcareous, graphitic phyllite (hosts Unit 4)
B	Calcareous muscovite-chlorite ± biotite phyllite
C	Metabasite
D	Laminarly banded, variably calcareous, chloritic phyllite
E	Phyllitic marble and silicated marble
F	Chloritic phyllite
G	Variably calcareous, graphitic phyllite
H	Amygdaloidal chloritic phyllite
I	Carbonaceous to graphitic argillite/phylite
1	Siliceous
2	Carbonaceous
3	Calcareous
4	Altered, pyritic (white mica envelope)
5	Banded/laminated
6	Non-calcareous
7	Tuffaceous
8	Chloritic
9	Sulfide-bearing
0	Normal

Conformable Contact

DY Deposits

Unit 4

4 A	Sulfide-bearing, ribbon-banded, graphitic quartzite	Banded
B	Pyrite-free quartzite (may contain base metal sulfides)	
C	Base metal-poor, pyritic quartzite	
D	Base metal-bearing, pyritic quartzite	
E	Massive pyritic sulfides	
F	Buckshot facies, massive sulfides	Massive
G	Baritic facies, massive sulfides/sulfates (>10% $BaSO_4$)	
H	Pyrrhotitic facies, massive sulfides	
J	Non-pyritic, massive sulfides/oxides	
K	Carbonate-bearing, massive pyritic sulfides	
L	Sulfide-bearing siliceous to tuffaceous exhalite	
1	Siliceous	
2	Course, porphyroblastic pyrite-bearing	
3	Fine pyrite/marcasite-bearing	
4	Sphalerite and/or galena-bearing	
5	Carbonaceous	
6	Barite-bearing	
7	Pyrrhotite-bearing	
8	Magnetite-bearing	
9	Chalcopyrite-bearing	
0	Normal	

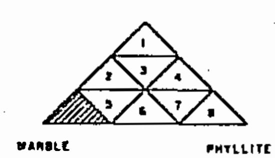
Conformable Contact

Mt. Mye Formation

Unit 3

3 A	Transition zone with Unit 1
B	Chloritic phyllite/schist
C	Metabasite
D	Calc-silicate phyllite/schist
E	Graphitic phyllite/schist
F	Marble and silicated marble
G	Non-calcareous, muscovite-chlorite ± biotite phyllite/schist, undifferentiated
H	Tuffaceous calc-silicate phyllite/schist (associated with 3D)
I	Graphitic quartzite in non-calcareous phyllite/schist
1	Siliceous
2	Non-calcareous
3	Calcareous
4	Altered, pyritic (white mica envelope)
5	Banded/laminated
6	Sulfide-bearing
7	Tuffaceous
8	Chloritic
9	Carbonaceous
0	Normal

CALC-SILICATE PHASES



M. CAMBRIAN/M. ORDOVICIAN

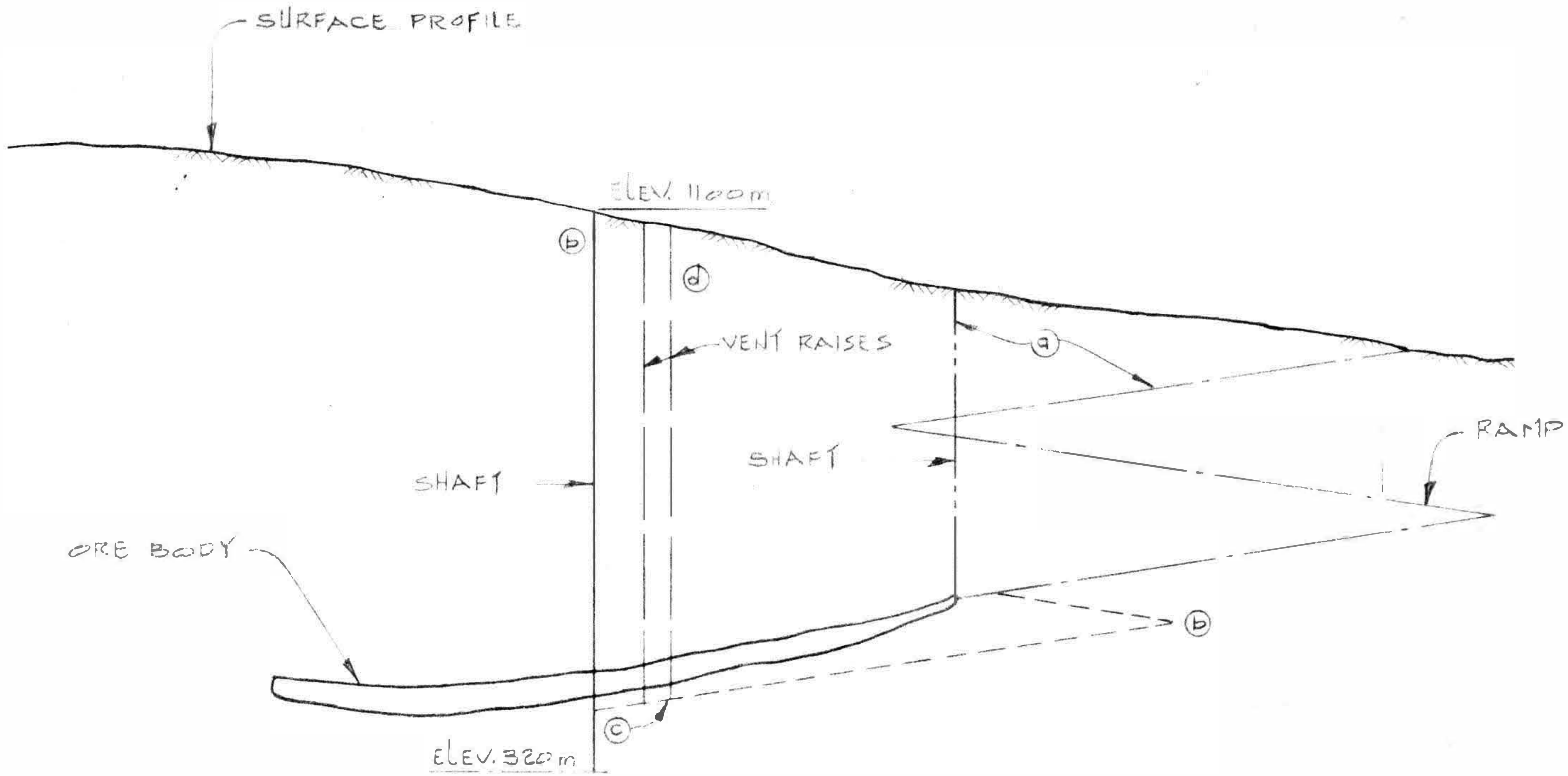
TABLE NO. 2

SCHEDULE OF CAPITAL AND OPERATING COSTS

($ x 1,000)

	Year -3	Year -2	Year -1	Year 1 to Year 15
Surface Plant Equipment		1,315	1,316	
Permanent Underground Equipment			1,075	
Underground Service Equipment		1,086	1,086	
Mine Ventilation		1,722	1,722	
Permanent Underground Development	4,870	3,151	4,840	$ 600 per year
Mine Equipment Replacement	(Allowance $8,700,000 over mine life)			
Capital Costs ($31,183,000)	4,870	7,274	10,039	$ 600 per year
Estimated Operating Cost (780,000 tons per year ore to shaft bins)				$12,300 per year Year 1 to Year 15

DRAWINGS



DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	SL						

CYPRUS ANVIL MINING CORPORATION
DY PROJECT

METHODS OF ACCESS

WRIGHT ENGINEERS LIMITED
VANCOUVER CANADA

SCALE: N.T.S

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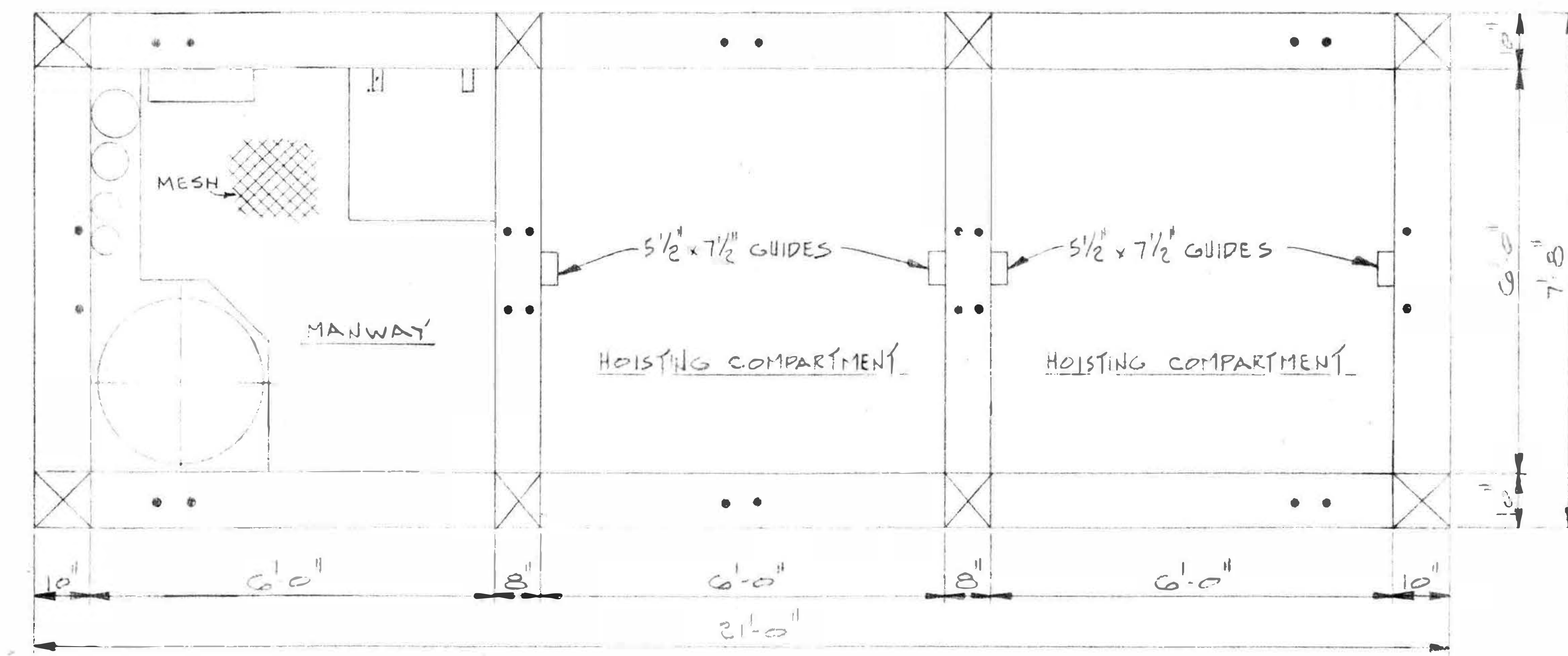
REV.



STANDARD SHAFT SET

DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	S.L.	WK					

CYPRUS ANVIL MINING CORPORATION
DY PROJECT

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EXPLORATION SHAFT-PLAN

SCALE: 1/2" = 1'-0"

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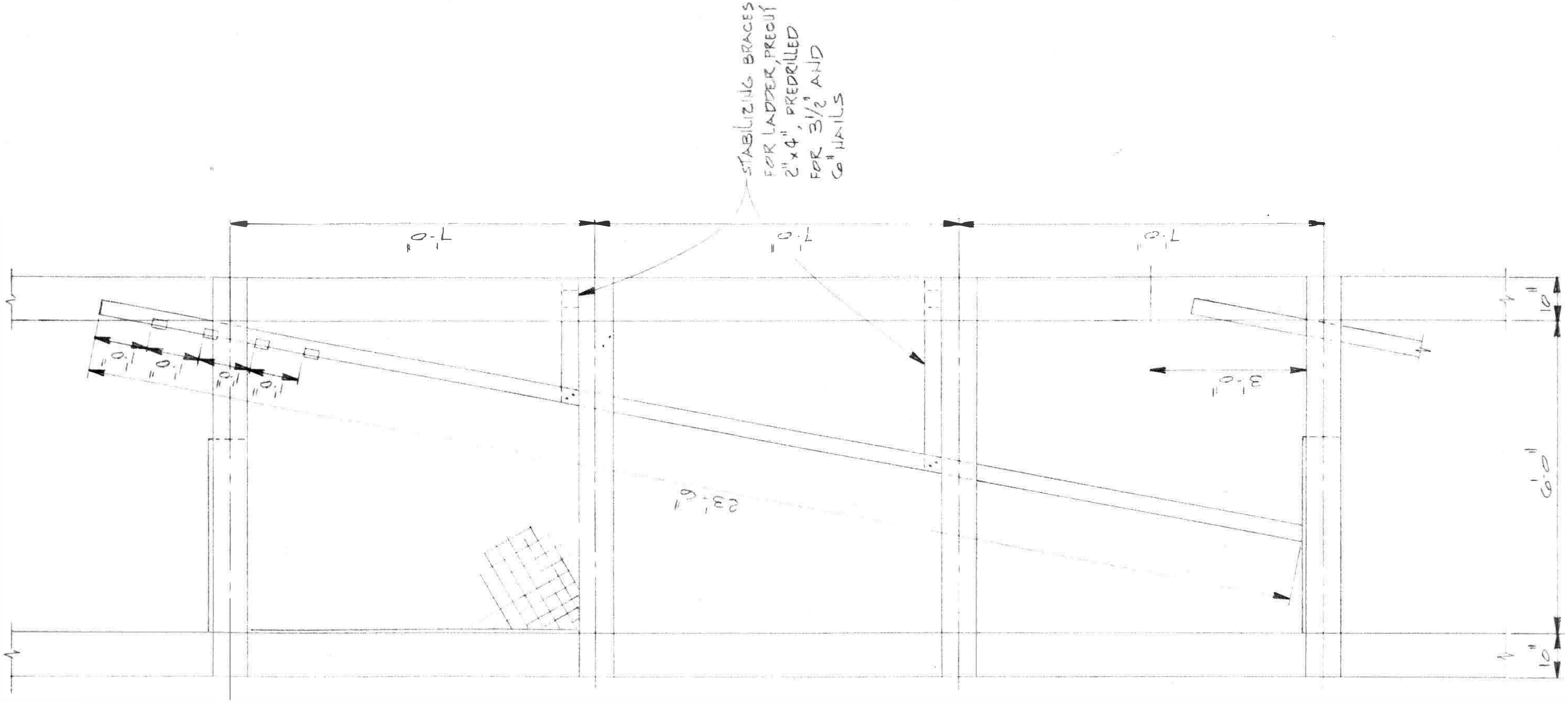
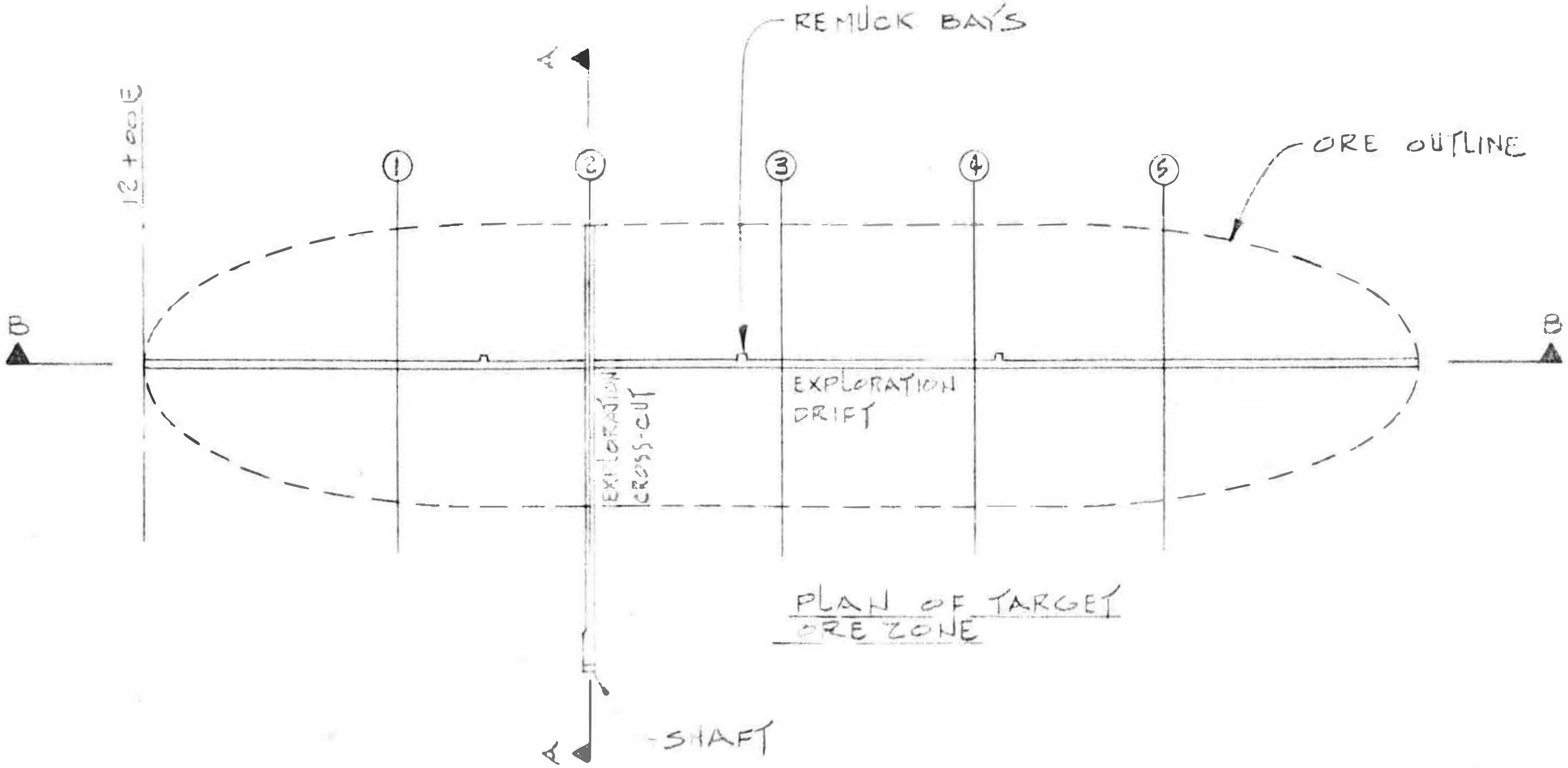
REV.



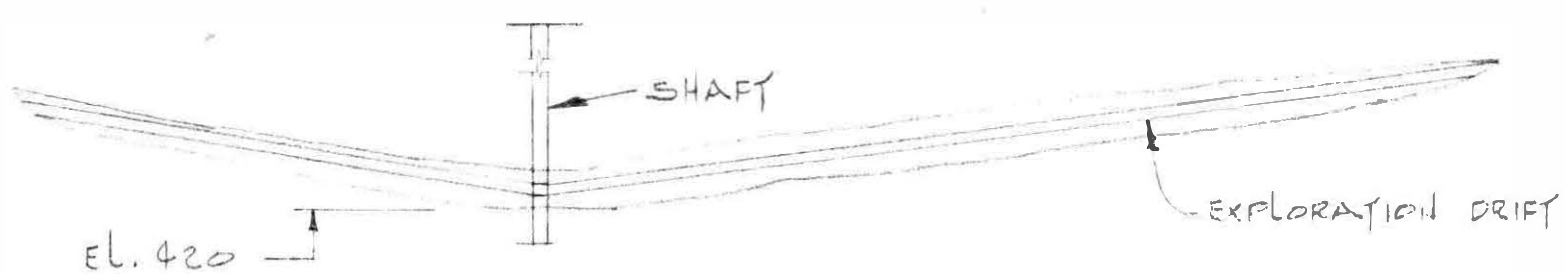
SECTION THRU MANWAY COMPARTMENT

DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	S.L.	WRL					

CYPRUS ANVIL MINING CORPORATION DY PROJECT	EXPLORATION SHAFT - SECTION
WRIGHT ENGINEERS LIMITED VANCOUVER CANADA	SCALE: ½" = 1'-0"
DRAWING No. B 945 100 1202B	REV

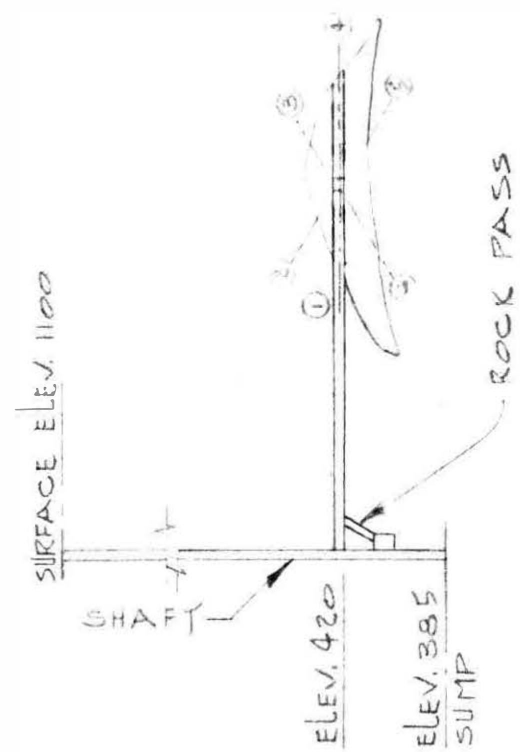


PLAN OF TARGET ORE ZONE



SECTION B-B

TYPICAL DRILL SECTION
6 HOLES TOTAL FOOTAGE
PER SECTION 470 m.



SECTION A-A

DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	S.L.	MRK					

CYPRUS ANVIL MINING CORPORATION
DY PROJECT

EXPLORATION DRIFTING & DRILLING

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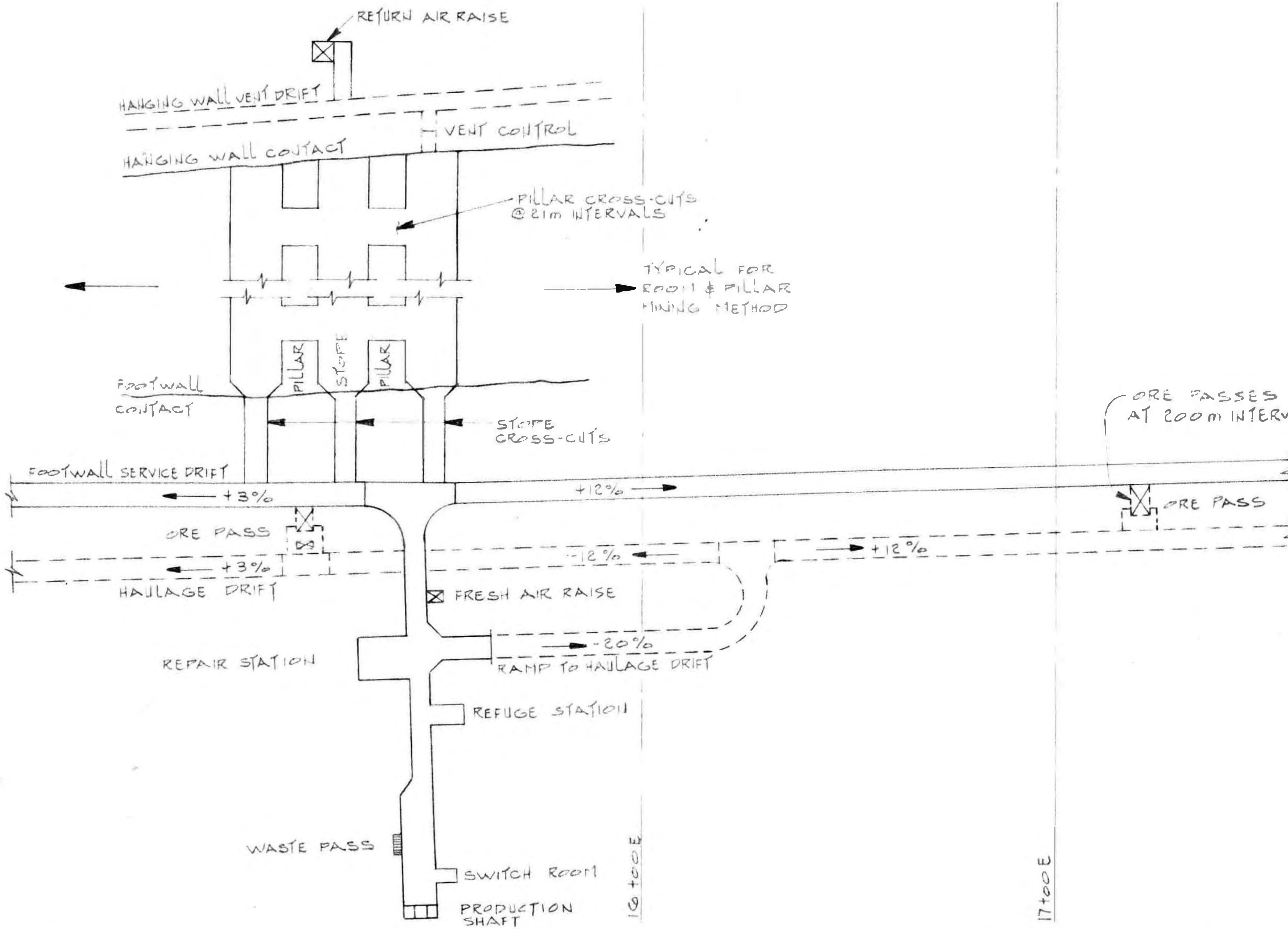
SCALE	DRAWING No.	REV.
N.T.S.	B 945 100 1203	



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	S.L.	NRK					

CYPRUS ANVIL MINING CORPORATION
DY PROJECT

WRIGHT ENGINEERS LIMITED
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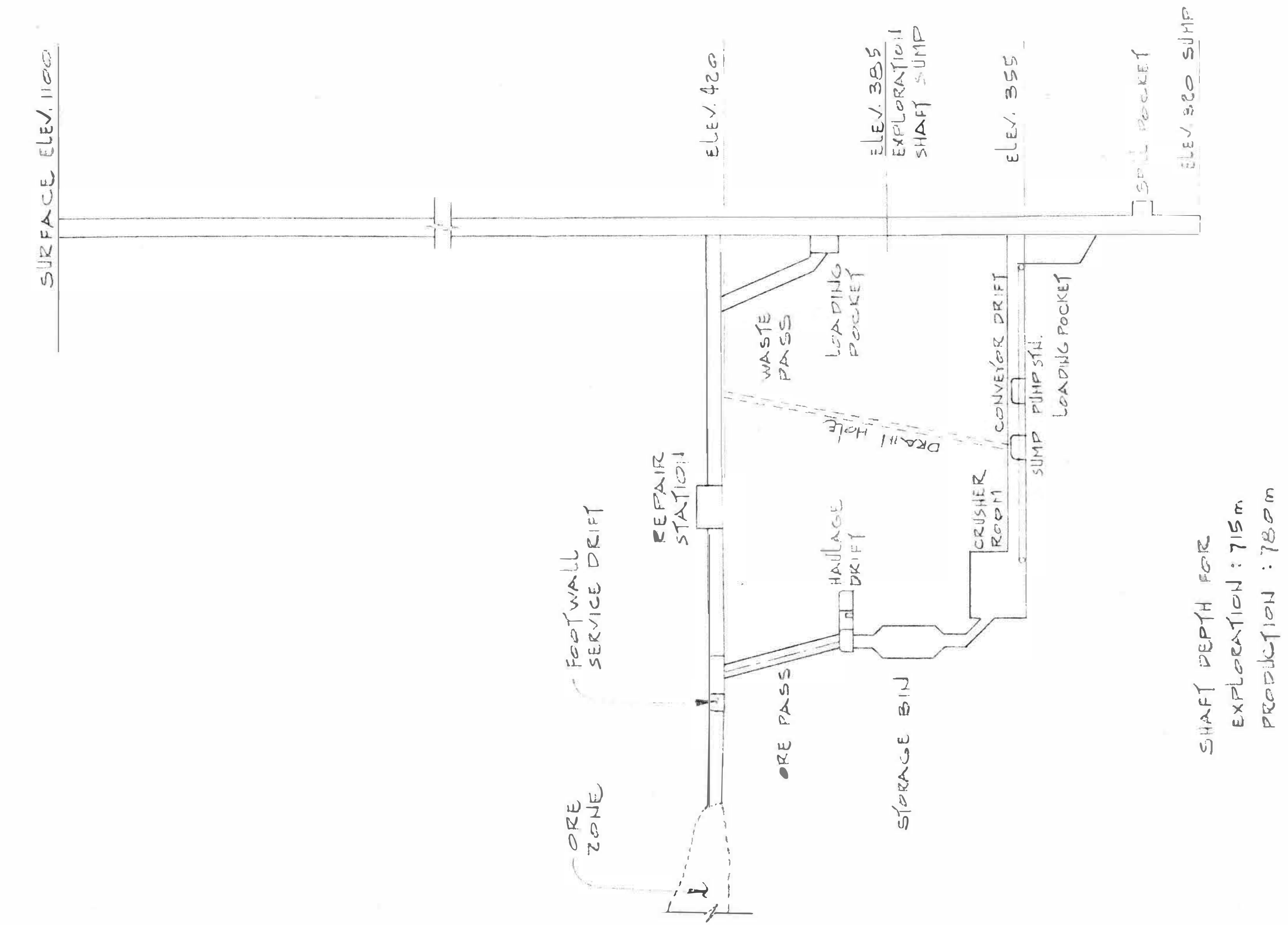
PLAN OF MINING LEVEL

SCALE: N.T.S.

DRAWING No. B 945 100 1204

REV.



DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	S. L.	KRL					

CYPRUS ANVIL MINING CORPORATION
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SECTION THROUGH SHAFT AND PERMANENT DEVELOPMENT

SCALE: N.T.S.

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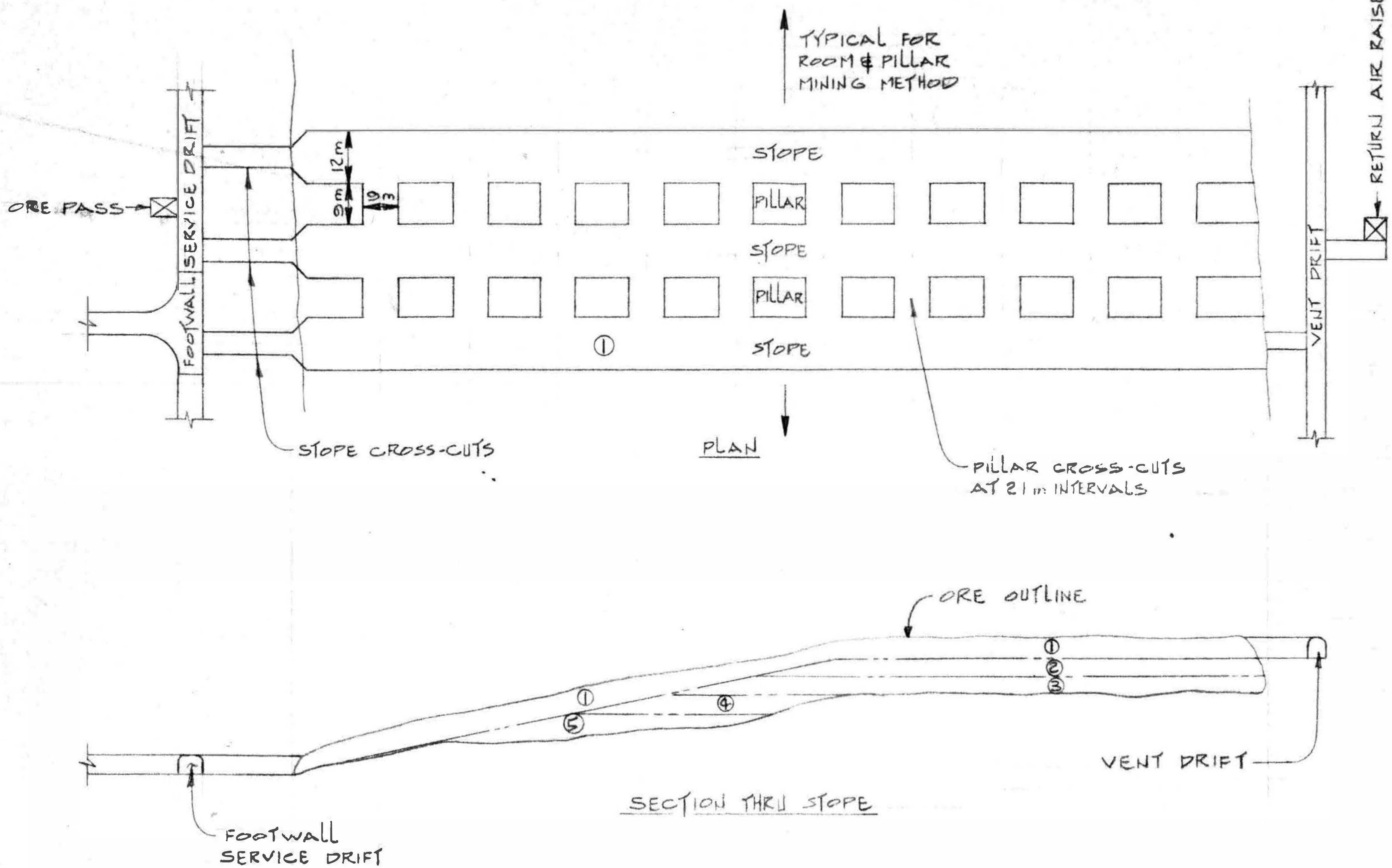

MINING SEQUENCE

① DRIVE H.W. DR., SLASH TO 12m, BOLT AND SCREEN BACK, CONNECT TO VENT DR., CROSS CUT THRU PILLARS FOR VENTILATION AND ORE RECOVERY.

② TO ⑤ MINE OUT 3m BENCHES, BOLT & SCREEN WALLS AS REQ'D, CROSS CUT THRU PILLARS.

DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
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CYPRUS ANVIL MINING CORPORATION
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ROOM AND PILLAR MINING METHOD

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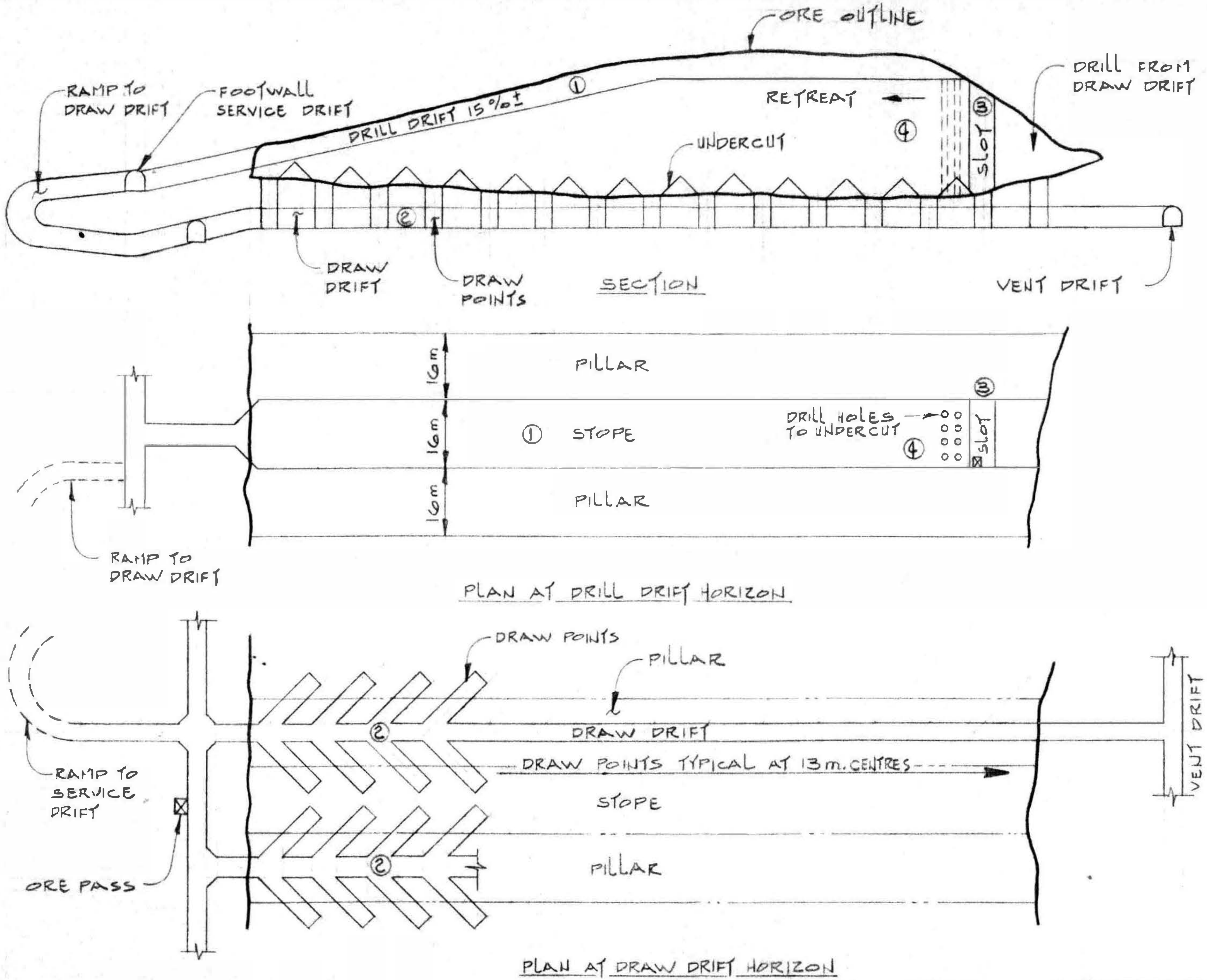
SCALE: N.T.S.

DRAWING No. B 945 100 1206

REV.



MINING SEQUENCE

① DRIVE H.W. DR. AND SLASH TO 16m, BOLT AND SCREEN BACK
② DRIVE DRAW DR. AND DRAW PTS.
③ OPEN SLOT ACROSS STOPE.
④ DRILL AND BLAST LONG HOLES.

DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
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CYPRUS ANVIL MINING CORPORATION
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BLAST HOLE MINING METHOD

WRIGHT ENGINEERS LIMITED
VANCOUVER CANADA

SCALE: N.T.S.

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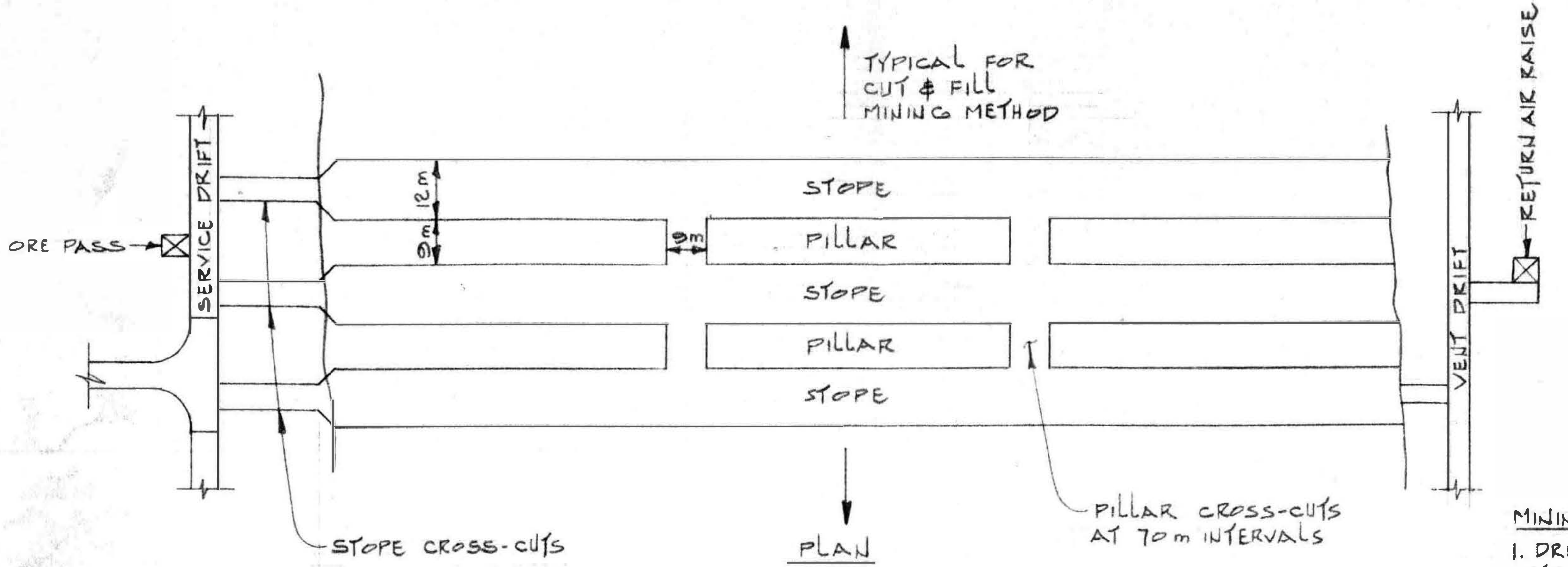
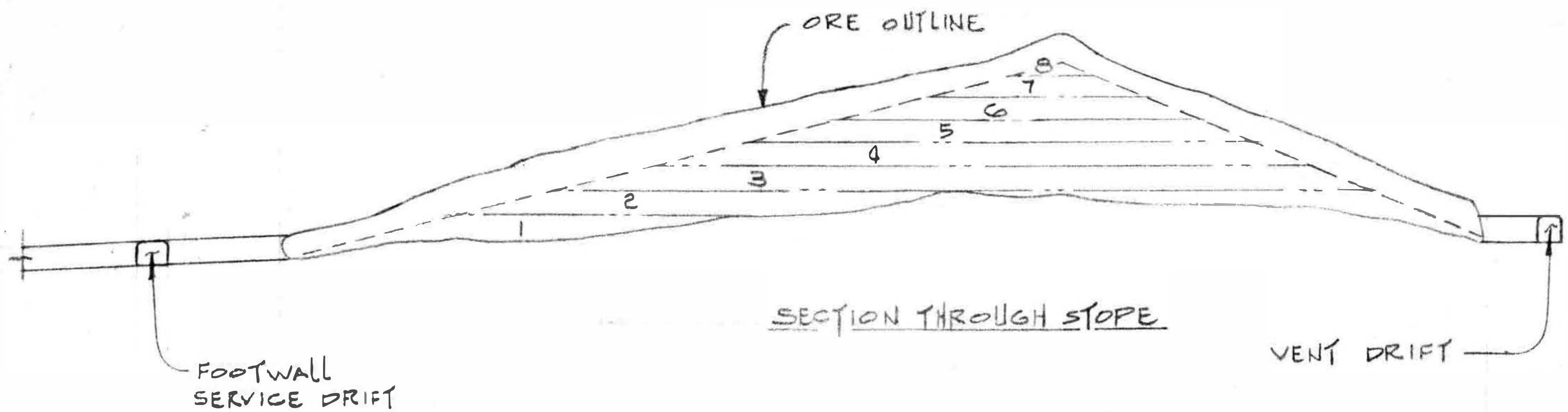


MINING SEQUENCE

1. DRIVE F.W. DRIFT, SLASH TO 12 m. WIDE
2. DRILL AND BLAST BACK AND CONNECT TO VENT DR. FILL 1 TO DASH LINE
3. RAMP ON FILL IN 1 AND MINE SLICE 3. FILL 2 TO DASH LINE
4. RAMP ON FILL IN 2 AND MINE SLICE 4

REPEAT PROCEDURE THRU 8 TO COMPLETION.



DSGN.	DRAWN	CHECK	APPR.	ISSUED FOR	DATE	REV.	DESCRIPTION OF REVISION
	S.L.	MRK					

CYPRUS ANVIL MINING CORPORATION DY PROJECT	CUT AND FILL MINING METHOD
WRIGHT ENGINEERS LIMITED VANCOUVER CANADA	SCALE: N.T.S. — DRAWING No. B 945 100 1208 — REV.

SUBMITTED BY:

WRIGHT ENGINEERS LIMITED

N. R. KRPAN, P.ENG.

VANCOUVER, B. C.
MAY, 1978.