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DESIGN GUIDELINES FOR MULTI-SEAM MINING AT ELLIOT LAKE

D.G.F. HEDLEY

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DESIGN GUIDELINES FOR MULTI-SEAM MINING AT ELLIOT LAKE

by

D.G.F. Hedley*

ABSTRACT

Multi-seam mining was practiced in some of the Elliot Lake uranium mines in the 1960's prior to their closing down. With the current expansion in uranium mining, multi-seam mining could again be practiced in the near future. Information on the dimensions of stopes, pillars and parting zone was gathered from plans and sections of the relevant closed mines. Discussions were held with personnel familiar with these mines to establish instances of pillar, roof, and parting zone failures. Design guidelines are formulated for stope and pillar dimensions in multi-seam mining for a range of orebody configurations using past practice in a back-analysis approach.

Besides achieving pillar and parting zone stability, the layout of multi-seam mining has to take into account geometrical factors such as dip and seam thickness. These factors affect the choice of mining equipment. The jackleg drills and slushers used in the 1960's are being replaced by jumbo drills and load-haul-dump units. Constraints imposed by dip and seam thickness on the choice of equipment and mining layout are evaluated.

An attempt is made to bring together the engineering aspects, including rock mechanics, of multi-seam mine design with uranium recovery and other economic factors for three alternative mining layouts: single-seam mining; double-seam mining; and seams-and-parting mining. A series of examples are worked through, showing how the design guidelines can be applied for typical orebody configurations.

Key words: Uranium, Elliot Lake, Mine Design, Multi-Seam Mining, Rock Mechanics, Economic Factors.

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DIRECTIVES DE CONCEPTION POUR L'EXPLOITATION MINIERE A COUCHES MULTIPLES

par

D.G.F. Hedley*

RESUME

L'exploitation minière à couches multiples a été mise en pratique dans quelques-unes des mines d'uranium d'Elliot Lake durant les années 60 avant leur fermeture. L'expansion actuelle de l'exploitation minière de l'uranium favoriserait sans doute l'exploitation minière à couches multiples dans un avenir rapproché. L'information sur les dimensions des gradins, des piliers et les zones d'intercalation a été recueillie à partir de plans et de profils des mines fermées. Les personnes connaissant ces mines se sont entretenues afin d'examiner les cas de défaillances au niveau des piliers, du toit et des zones d'intercalation. Les directives de conception ont été élaborées sur les dimensions des gradins et des piliers dans des exploitations minières à couches multiples et pour une série de configurations de corps minéralisés en se basant sur les expériences du passé.

A part l'obtention d'une stabilité au niveau des piliers et de la zone d'intercalation, la disposition de l'exploitation minière à couches multiples doit tenir compte des facteurs géométriques tels que l'épaisseur de la couche et de l'inclinaison. Ces facteurs influencent le choix de l'équipement d'exploitation. Les marteaux perforateurs et les râteleurs employés durant les années 60 ont été remplacés par des mèches géantes et des appareils de chargement-roulage-déversement. On évalue dans le présent rapport les contraintes imposées par l'épaisseur de la couche et de l'inclinaison sur le choix de l'équipement et de la disposition d'une mine.

On essaie de combiner le point de vue de la science appliquée, y compris la mécanique des roches, du concept des mines à couches multiples avec la récupération de l'uranium et les autres facteurs économiques afin d'en arriver à trois concepts différents d'exploitation minière: exploitation minière à une couche, exploitation minière à double couche et exploitation minière par couches et par intercalations. Une série d'exemples sont inclus dans le rapport démontrant comment ces directives pouvaient être appliquées pour les configurations des corps minéralisés typiques.

Mots clés: Uranium, Elliot Lake, concept de la mine, exploitation minière à couches multiples, mécanique des roches, facteurs économiques.

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INTRODUCTION

The uranium orebodies at Elliot Lake are on the north and south limbs of a broad syncline. In the northern limb, five uranium-bearing conglomerate reefs or seams* exist, separated by quartzite beds from a few feet to over 100 ft thick. In the southern limb, three uranium reefs are separated by quartzite beds up to 30 ft thick. These reefs are not continuous and overlapping occurs only at some mining properties.

Originally there were twelve mines in this area, of which only two are at present in operation. Rehabilitation of some of the closed mines is planned. A stope or room-and-pillar mining layout was used at all mines. The Nordic, Lacnor and Milliken mines of Rio Algom Ltd., all on the southern limb, practiced multi-seam mining during the 1960's prior to closing down. Stanrock mined two reefs, one above the other although actually it was the same reef displaced by a low angle thrust fault. Denison also left the parting zone between reefs in a few stopes.

With the current expansion of uranium mining by Rio Algom Ltd., Preston Mines Ltd., and Denison Mines Ltd., it is envisaged that multi-seam mining could again be practiced in the near future. Unfortunately, no technical articles were published on multi-seam mining at Elliot Lake during the time this method was used. Also, most of these mines are now partially flooded and underground conditions cannot be observed. To document all available information, plans and sections of the relevant closed mines were examined to determine dimensions of stopes, pillars, and parting zones. Instances of pillar, roof, and parting zone failures were collected and discussions held with personnel who worked at these mines. Using this information, guidelines are formulated in this report on design layout for multi-seam mining, stability of pillars and the parting zone are evaluated, and recovery and economic factors for mining single-seam, double-seam, and both seams-and-parting zone are analyzed.

MINING BACKGROUND

Figure 1 shows the general geological sections through the northern and southern limbs. In the northern limb the mining companies have identified the various reefs by letters (i.e., A, B, C, etc.). Unfortunately Rio Algom Ltd. started at the top reef and numbered downwards, whereas Denison Mines Ltd. started at the bottom reef and numbered upwards. To avoid confusion, the major reefs have been identified by the name of the major mine operating in that reef. Names have already been given to reefs in the southern limb. The mines which operated in each reef are also identified.

During the early 1960's the stope and pillar layout at most mines in Elliot Lake were very similar and a typical plan and section of double-seam mining is shown in Fig. 2. Normally the lower reef was developed first by driving twin raises on either side of a centre pillar between haulage or sill drifts. These raises were then expanded in a series of slices until the full stope width was opened up. The broken ore was scraped down dip to a boxhole or chute at the bottom of the stope. Rib pillars were left on dip separating each pair of stopes, and sill pillars were usually left at the top and bottom of each stope. The roof was bolted with 6-ft and 8-ft bolts spaced at either 4 x 4-ft or 5 x 5-ft centres. In some mines, wooden posts were also installed between roof and floor of the lower reef. The upper seam was then developed in a similar manner, care being taken to ensure that pillars in the upper reefs were located above those in the lower reef.

Information on multi-seam mining practice at Nordic, Milliken, Lacnor, and Denison mines is summarized in Table 1. There are no reported instances of either pillar failure or parting zone collapse in the multi-seam mining area at the Nordic mine or the few two-seam stopes at Denison mine. At the Lacnor mine there was also no reported collapse of the parting zone, but there were a few instances of pillar spalling and two

* Throughout this report "reef" and "seam" are synonymous. Reef is the normal term used at Elliot Lake.

cases of roof failure when 75-ft stope spans were attempted. Only at the Milliken mine were there cases of both pillar failure and collapse of the parting zone. However, changes in stope-and-pillar design alleviated these problems.

PARTING ZONE STABILITY

From examination of individual stope plans and sections at the mines practicing two-seam mining, a data bank was collected on cases where

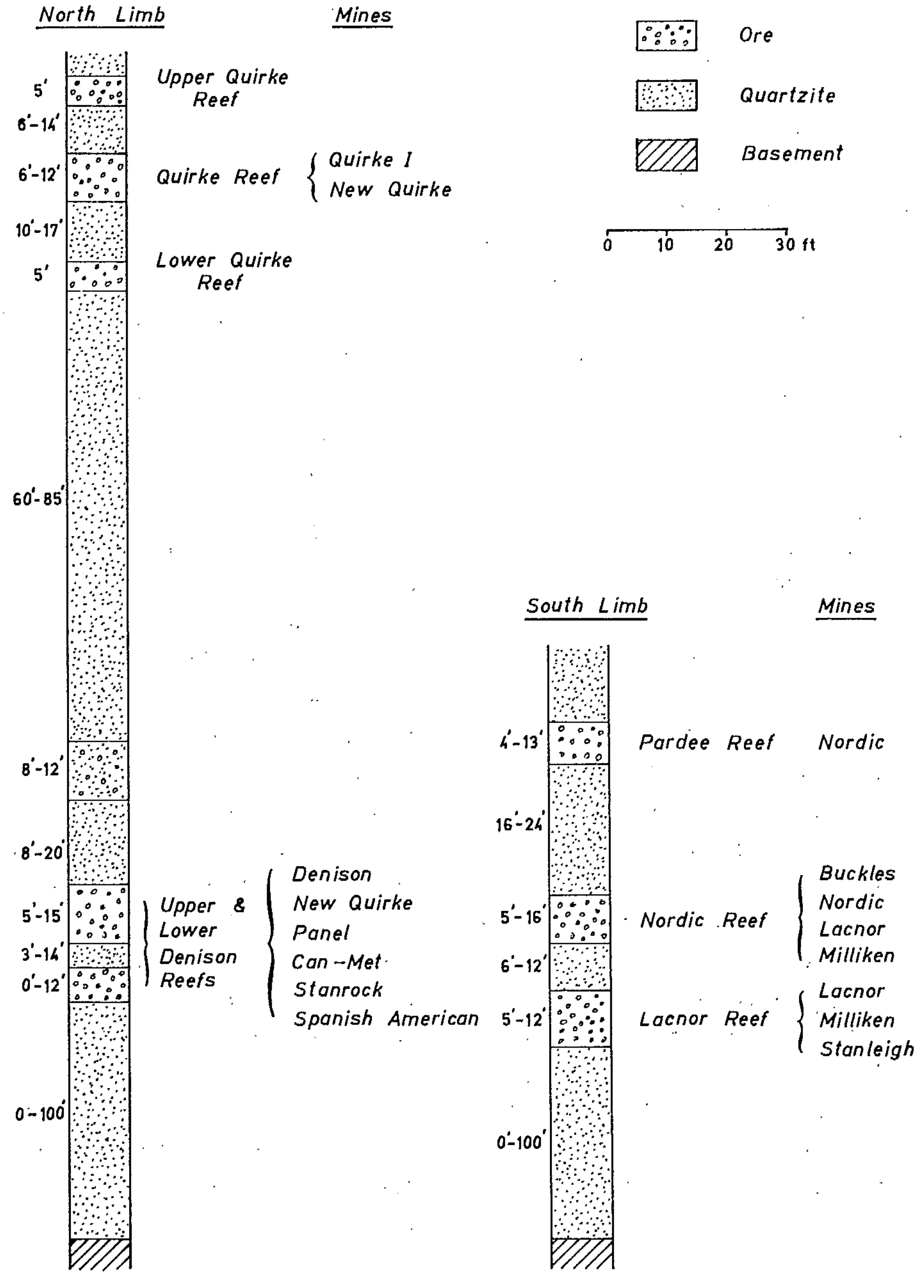
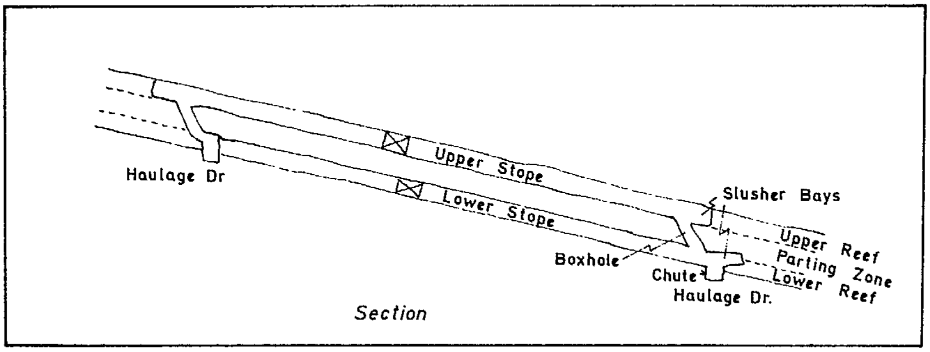


Fig. 1 - Typical geological sections of the north and south limbs



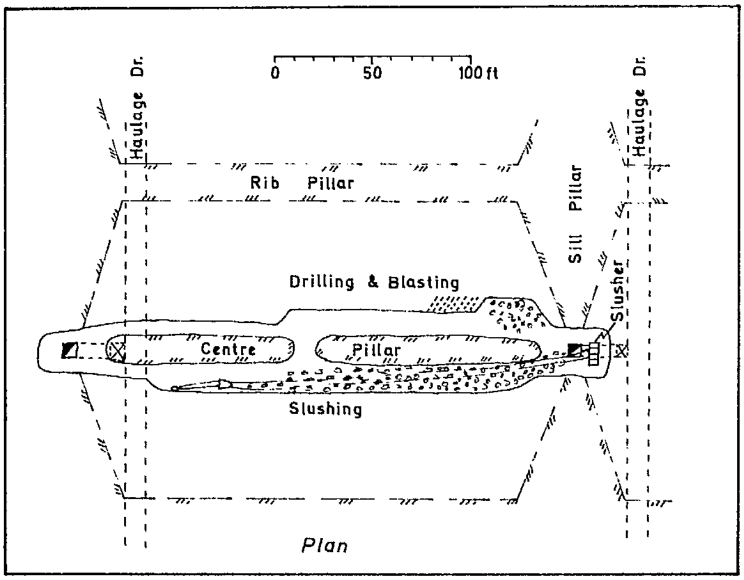


Fig. 2 - Typical stope and pillar layout for double-seam mining in the 1960's

the parting zone remained in place and on cases where it collapsed. This information is presented in Fig. 3 by plotting parting thickness against stope span. There were 36 cases at four mines where the parting zone was stable and remained in place, whereas in 7 cases at one mine the parting zone collapsed. Each point on the graph represents several stopes with similar dimensions. At the Milliken mine there were more cases of parting zone collapse but these were associated with pillar failure and are not included in Fig. 3.

A number of mechanisms which could cause failure of the parting zone include: buckling due to high horizontal forces, tensile failure at the centre of the roof in the lower stope, and shear or tensile failure next to the pillar edge. There has been no reported instance of tensile cracks running down the centre of the roof in the lower stope. An eye witness account stated that the parting zone collapsed as one unit which could suggest shear or tensile failure near the pillar edge. Also, finite element model studies on multi-seam mining at the Elliot Lake mines indicate a tensile stress in the parting zone near the pillar edge.

In the early 1960's, a wide range of stope spans from 25 to 100 ft were being used at the mines. However, present practice is limited to a

Table 1: Multi-seam mining practice

Nordic mine

Operated:	1957 to 1969		
Upper reef:	7 ft thick	Lower reef:	7 - 12 ft thick
Parting zone:	15 - 25 ft thick	Dip:	17°
Stope span:	65 ft	Stope length:	300 ft
Rib pillars:	10 ft wide	Extraction:	84%

Depth below surface of two-reef mining: 670 - 1050 ft

Number of two-reef stopes: 46 single

Rock bolts: 6 ft long at 5-x 5-ft centres

Comments: Lower reef mined first.
No collapse of parting zone.
No pillar failures.
Minor loose in roof of lower seam.

Milliken mine

Operated:	1957 to 1964		
Upper reef:	7 - 10 ft thick	Lower reef:	7 ft thick
Parting zone:	7 - 16 ft thick	Dip:	12°
Stope span:	40 - 100 ft, mainly 60 ft	Stope length:	200 and 450 ft
Rib pillars:	10 - 20 ft, mainly 20 ft	Extraction:	70 - 85%, mainly 70%

Depth below surface of two-reef mining: 2700 - 3000 ft

Number of two-reef stopes: 52

Rock bolts: 6 ft long at 4- x 4-ft centres

Comments: First experimental stope mined top seam first, but soon converted to mining lower seam first; 18 cases of parting zone collapse, 100 roof failures and 80 rib pillar failures (mainly 10 ft wide), in some cases pillar failure initiated collapse of parting zone.
Roof failures often associated with faulting.
Change to 20-ft wide pillars, 60-ft stope spans and 70% extraction alleviated pillar failures.
Lower seam posted sometimes with 2 by 4's which were used as a sag indicator.
Parting zone collapsed as one piece with cracks initiated near the pillars.
Slusher sometimes hit top of bolts in parting zone.

Table 1 (cont'd.)

Lacnor mine

Operated:	1957 to 1960		
Upper reef:	7 - 13 ft thick	Lower reef:	6 - 12 ft thick
Parting zone:	6 - 12 ft thick	Dip:	17°
Stope span:	45 ft	Stope length:	200 ft
Rib pillars:	10 ft and 15 ft wide	Extraction:	75%

Depth below surface of two-reef mining: 2200 - 2600 ft

Number of two-reef stopes: 70

Rock bolts: 6 and 8 ft long at 4-x 4-ft centres

Comments:

First experimental stope mined top seam first, but then converted to mining lower seam first.

No collapse of parting zone.

Wooden posts initially installed in lower seam, but took no weight and practice discarded.

Some pillar spalling and two cases of roof failure when 75 ft stope spans were attempted.

Slusher sometimes hit top of bolts in parting zone.

Denison mine

Operated:	1957 to date (1960 - 61 for multi-seam)		
Upper reef:	8 - 10 ft thick	Lower reef:	8 - 9 ft thick
Parting zone:	8 - 11 ft thick	Dip:	20°
Stope span:	35 - 45 ft	Stope length:	200 ft
Rib pillars:	10 ft and 25 ft wide	Extraction:	67%

Depth below surface of two-reef mining: 1900 and 2400 ft

Number of two-reef stopes: 7 single

Rock bolts: 6 and 8 ft long at 3-1/2-x 3-1/2-ft centres

Comments:

Lower reef mined first, normally two stopes with 40-ft span separated by 10-ft wide pillar.

Upper reef mined over complete width 80-to 100-ft span.

No collapse of parting zone and no pillar failures.

Slusher sometimes hit top of bolts in parting zone.

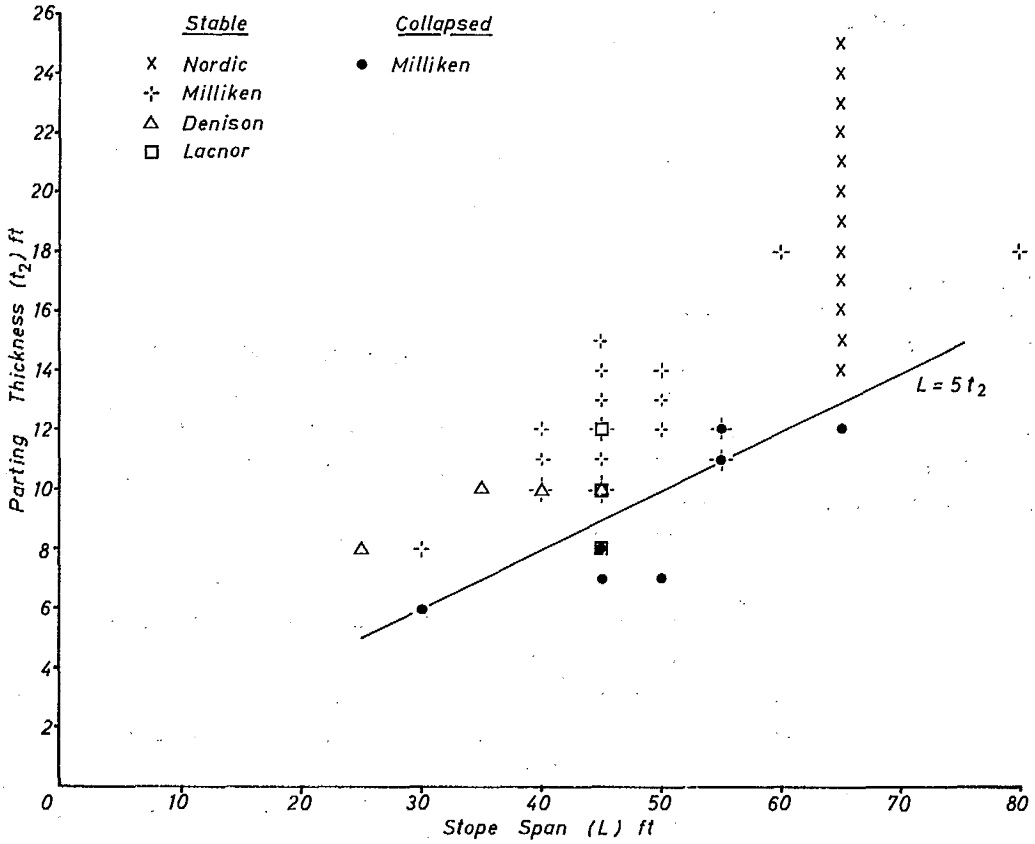


Fig. 3 - Relationship between parting thickness and stope span

range of 50-ft to 65-ft spans. Within this limited range, a straight line relationship can be used between maximum stope span for a given parting thickness. From Fig. 3, the dividing line between stable and unstable conditions can be expressed by:

$$L \leq 5t_2 \qquad (1)$$

where: L = maximum stope span, ft

t_2 = parting zone thickness, ft

A decision on whether or not to practice multi-seam mining is essentially only required when the parting thickness is between 9 ft and 13 ft. Above 13 ft, a stope span of 65 ft can be used which is present practice and multi-seam mining should be no problem. Below 9 ft, a stope span in the order of 40 ft would be required and conventional multi-seam mining would likely be uneconomical due to the extra development work required. However, in that case as described later, a system utilizing backfill could be used. Between 9 ft and 13 ft, a stope span of about 45 to 50 ft would be required which means rib pillar spacing at about 60-ft centres.

Referring back to the geological sections in Fig. 1, it can be seen that in the northern limb there is no problem in double-seam mining between the Denison and Quirke reefs which are about 100 ft apart. Similarly in the southern limb there are no problems with the Nordic and Pardee reefs which are about 20 ft apart. Consequently, the main decisions on multi-seam mining concern the upper and lower Denison reefs, and between the three Quirke reefs in the northern limb and between the Lacnor and Nordic reefs in the southern limb.

PILLAR STABILITY

A previous investigation established design guidelines for pillars in single-seam mining at the Elliot Lake mines (1). Average pillar strength and stress were defined by:

Average pillar strength,

$$Qu(psi) = 26{,}000 \frac{W^{0.5}}{H^{0.75}} \qquad (2)$$

Average pillar stress,

$$(\sigma p)(psi) = \frac{1.1\ D \cos^2\alpha + 3000 \sin^2\alpha}{1 - R} \qquad (3)$$

where: W = pillar width, ft
H = pillar height, ft
D = depth below surface, ft
α = dip of orebody, degrees
R = extraction ratio.
Horizontal stress is taken as 3000 psi.

Mining a second seam with identical layout of stopes and pillars would have little effect on the pillar stress distribution since the extraction ratio would not change. However, pillar strength would be affected and is dependent on how close the seams are to each other. To estimate pillar strength for such conditions, the concept of an effective pillar height is introduced. Essentially, this means that pillars in the upper and lower seams plus the parting zone are defined in terms of a single pillar of equivalent strength. The height of this single equivalent pillar is the effective pillar height. There is no experimental data on which to base effective pillar height and it is necessary to assume a relationship which fits logical boundary conditions. Intuitively, if the parting zone is only a few feet thick the effective pillar height should be almost the sum of the upper and lower seams. As the parting increases in thickness the interaction should decrease and eventually the pillars in the two seams could be treated independently. A simple equation which approximates this line of reasoning is:

$$H = t_1 + \frac{t_3^2}{2t_2 + t_3} \qquad (4)$$

where: H = effective pillar height, ft
t_1 = thickness of major seam (either upper or lower), ft
t_2 = thickness of parting zone,ft
t_3 = thickness of minor seam (either upper or lower), ft.

The equation indicated that when the parting zone is zero the effective pillar height is the sum of the upper and lower seams. As the parting zone increases in thickness the second term in the equation decreases significantly and the effective pillar height approaches the thickness of the major seam.

It is possible to determine the applicability of these equations to two-seam mining by comparing with conditions at three mines; Nordic, Lacnor, and Milliken. The basis for comparison is the safety factor (S.F.), defined as the ratio of pillar strength to pillar stress:

$$S.F = \frac{Qu}{\sigma p} \qquad (5)$$

For values greater than one, the pillars should be stable and, for values less than one the pillars should fail. The relevant mining parameters at the three mines as well as the calculated safety factors are listed in Table 2. At Nordic, the calculated safety factor was about 2 and the pillars were stable. At Milliken where 10-ft wide pillars were left and 80% extraction taken, the calculated safety factor was less than one and there were 80 cases of pillar failure. When the mining layout was changed to pillars 20 ft wide and 70% extraction, pillar stability was achieved with a calculated safety factor of about 2. At Lacnor the calculated safety factor was just over one, and it is reported that towards the end of the mine life, some pillars crushed when pillar recovery operations were attempted. This indicates that the pillars were fairly close to failure.

If only the major seam thickness rather than effective pillar height were used in the equations, then the calculated safety factors would be

Table 2: Comparison of calculated and actual pillar stability

Mining parameter	Nordic	Lacnor	Milliken	Milliken
Average depth	800 ft	2400 ft	2800 ft	2800 ft
Dip	17°	17°	12°	12°
Extraction	84%	75%	70%	80%
Pillar width	10 ft	15 ft	20 ft	10 ft
Major seam thickness, t_1	10 ft (lower)	10 ft	9 ft (upper)	9 ft (upper)
Average parting thickness, t_2	20 ft	10 ft	8 ft	8 ft
Minor seam thickness, t_3	7 ft (upper)	10 ft	7 ft (lower)	7 ft (lower)
Effective pillar height	11.0 ft	13.3 ft	11.1 ft	11.1 ft
Average pillar strength	13,600 psi	14,400 psi	19,100 psi	13,500 psi
Average pillar stress	6,600 psi	10,700 psi	10,300 psi	15,400 psi
Calculated safety factor	2.1	1.3	1.9	0.9
Actual conditions	pillars stable	spalling on some pillars	pillars stable	80 cases pillar failure

increased. Specifically, at Lacnor the safety factor would be 1.7 rather than 1.3 and at Milliken with 80% extraction from 0.9 to 1.03. Because a large number of pillar failures occurred at Milliken and pillars at Lacnor were observed to be nearly in a failure condition, suggests that the effective pillar height must take into account the parting zone and minor seam thicknesses.

Based on this evidence, there appears to be a correlation between calculated pillar stability in two-seam mining and the observed pillar stability underground. However, it should be noted that this analysis has been tested only over a very limited range of mining parameters. In the absence of any better method of engineering analysis, the equations for average pillar strength and stress can be used for design purposes to determine stope and pillar layouts for various depths and multi-seams dimensions.

The equations can be simplified by assigning values to a number of parameters as follows:

a. When dip of the orebody is less than 30°, the horizontal stress has limited effect on the perpendicular stress acting on the orebody, and hence the dip can be fixed at 15°.

b. For planning purposes, a safety factor of 1.3 represents the dividing line between uncertain and stable conditions.

c. Basically, three types of pillar and pillar spacing are used in normal practice, the basis for their choice being described later in the report.

   i) Rib pillars on dip or at an angle to dip are spaced at either 60-ft or 75-ft centres. The area of sill pillars at the top and bottom of long stopes is relatively small and the extraction ratio, R, can be approximated by:

$R = \frac{60 - W}{60}$ for rib pillars spaced at 60-ft centres

$R = \frac{75 - W}{75}$ for rib pillars spaced at 75-ft centres.

ii) For square pillars spaced at 60-ft centres, the extraction ratio, R, can be expressed by:

$$R = \frac{3600 - W^2}{3600} .$$

Substituting for R for each type of pillar and spacing, equations (2) and (3) and combining with equation (5) can now be expressed as:

For rib pillars at 60-ft centres,

$$1.3 = \frac{\dfrac{26{,}000\ W^{0.5}}{H^{0.75}}}{\dfrac{1.03\ D + 200}{W/60}} . \qquad (6)$$

which can be rearranged to:

$$W = [3.0 \times 10^{-3}\ (1.03\ D + 200)\ H^{0.75}]^{0.67}. \qquad (7)$$

Similarly for rib pillars at 75-ft centres,

$$W = [3.75 \times 10^{-3} (1.03\ D + 200)\ H^{0.75}]^{0.67}. \qquad (8)$$

and for square pillars at 60-ft centres

$$W = [0.18\ (1.03\ D + 200)\ H^{0.75}]^{0.4} . \qquad (9)$$

H is the effective pillar height as defined in equation (4).

Equations (7), (8) and (9) are represented in nomograph form in Fig. 4, 5 and 6 respectively, relating upper, lower, and parting thickness, effective pillar height, depth, pillar width, stope width, and extractions. These nomographs can also be used for single-seam mining by using the seam thickness as the effective pillar height and for mining both seams and parting where the combined thickness is the effective pillar height.

As an example of the use of nomographs, for a minor seam thickness, t_3, of 8 ft, a parting zone, t_2, of 10 ft and a major seam thickness, t , of 12 ft, the effective pillar height, H, is 14.3 ft. At depths of 1500 ft and 3000 ft, the following pillar widths and extractions are obtained for the three pillar spacings:

	Rib pillars 60-ft centres		Rib pillars 75-ft centres		Square pillars 60-ft centres	
Depth	Width	Extraction	Width	Extraction	Width	Extraction
1500 ft	11.4 ft	81%	13.3 ft	82%	22.2 ft	86%
3000 ft	17.4 ft	71%	20.2 ft	73%	28.5 ft	77%

These results indicate that a square layout gives 4 - 5% more extraction than a rib pillar layout. However, the occasions where square pillars can be used is limited as explained in the next section.

DESIGN CONSTRAINTS IN MULTI-SEAM MINING

Economic factors such as ore grades, mining costs, selling price of uranium, capital requirements, timing of expenditures and cash flow are the major factors in determining whether to mine a single seam, double seam or both seams and the parting zone. However, the layout of stopes and pillars is subject to other constraints including stability of pillars and parting zone as well as geometrical factors such as dip and seam thickness.

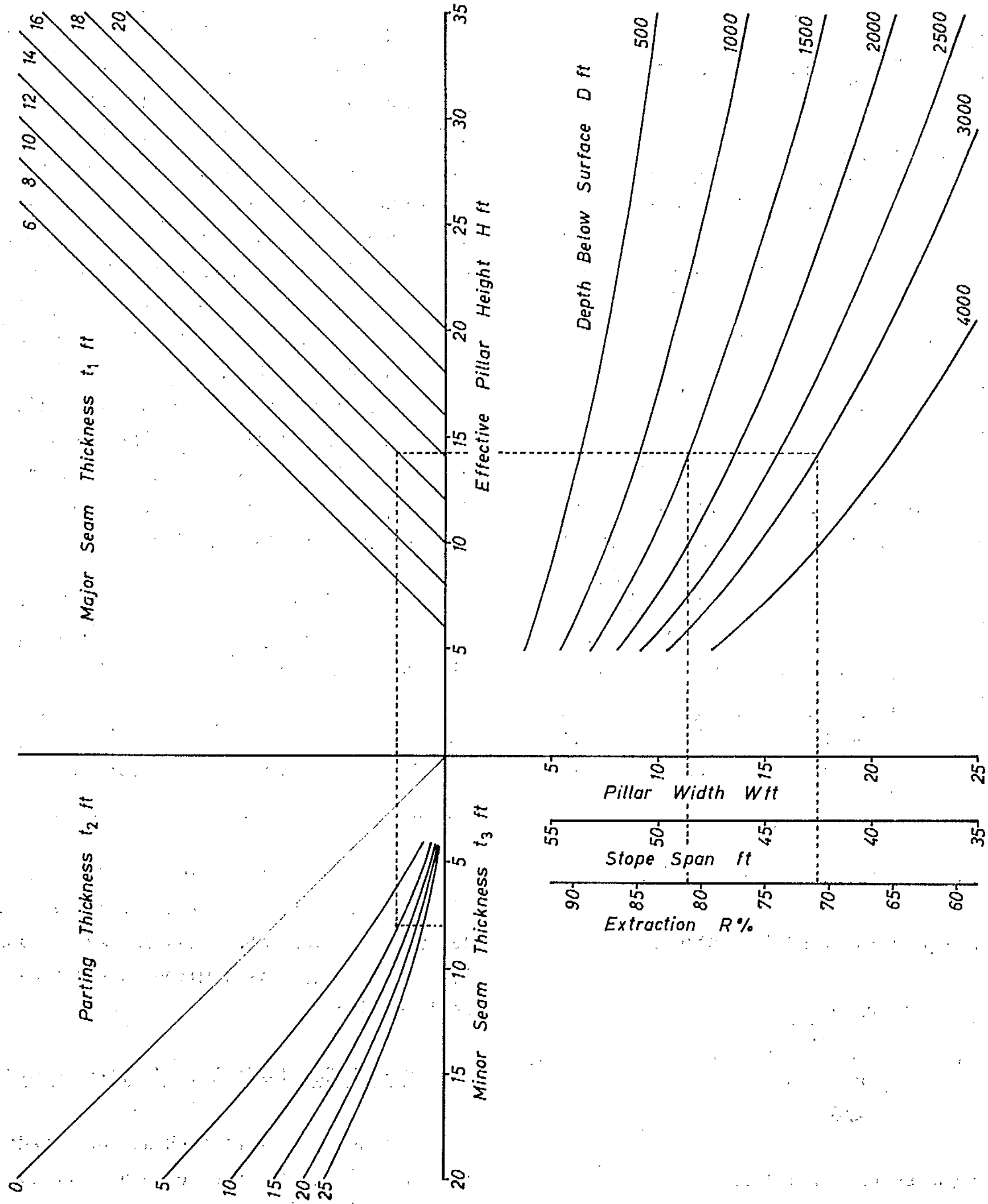


Fig. 4 - Design nomograph for rib pillars spaced at 60-ft centres

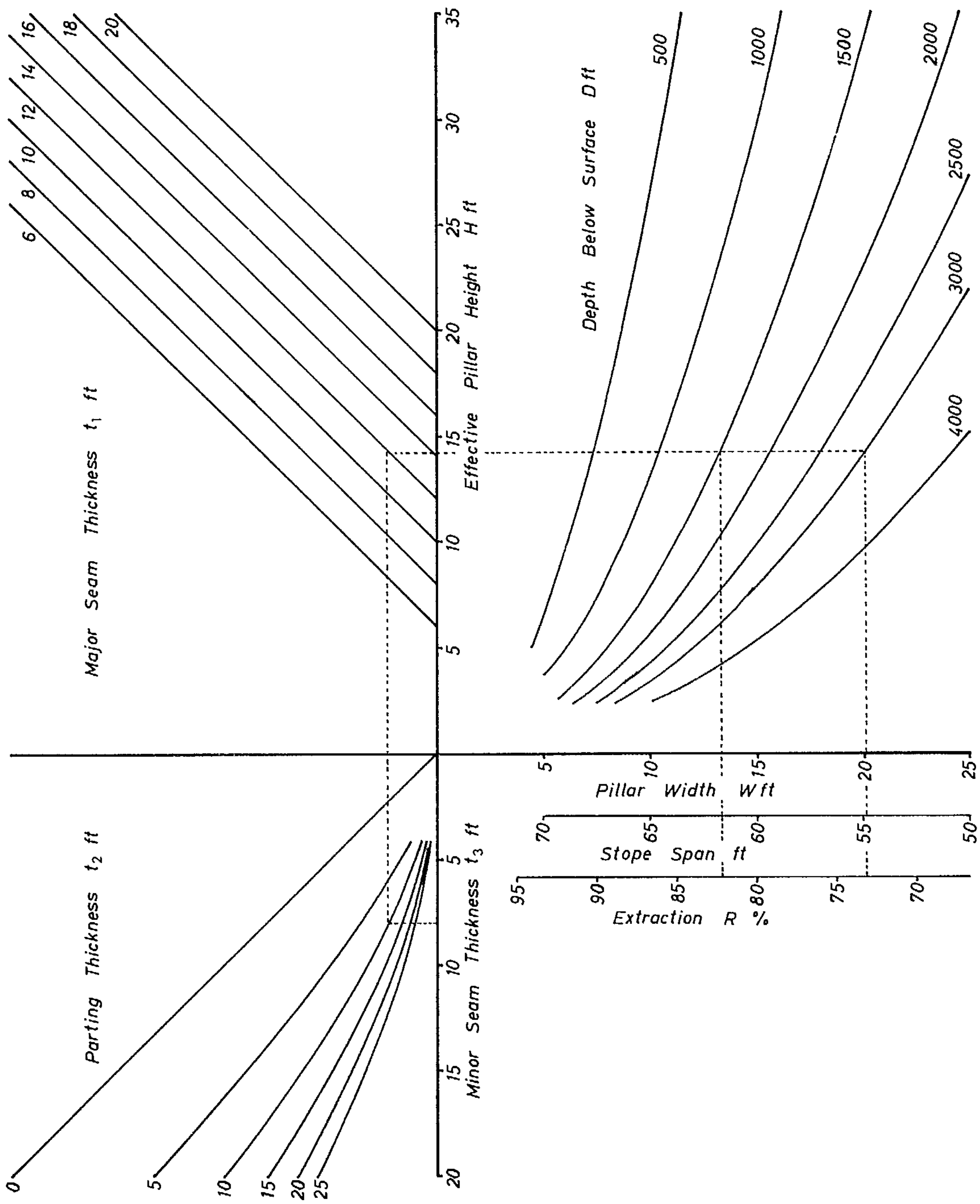


Fig. 5 - Design nomograph for rib pillars spaced at 75-ft centres

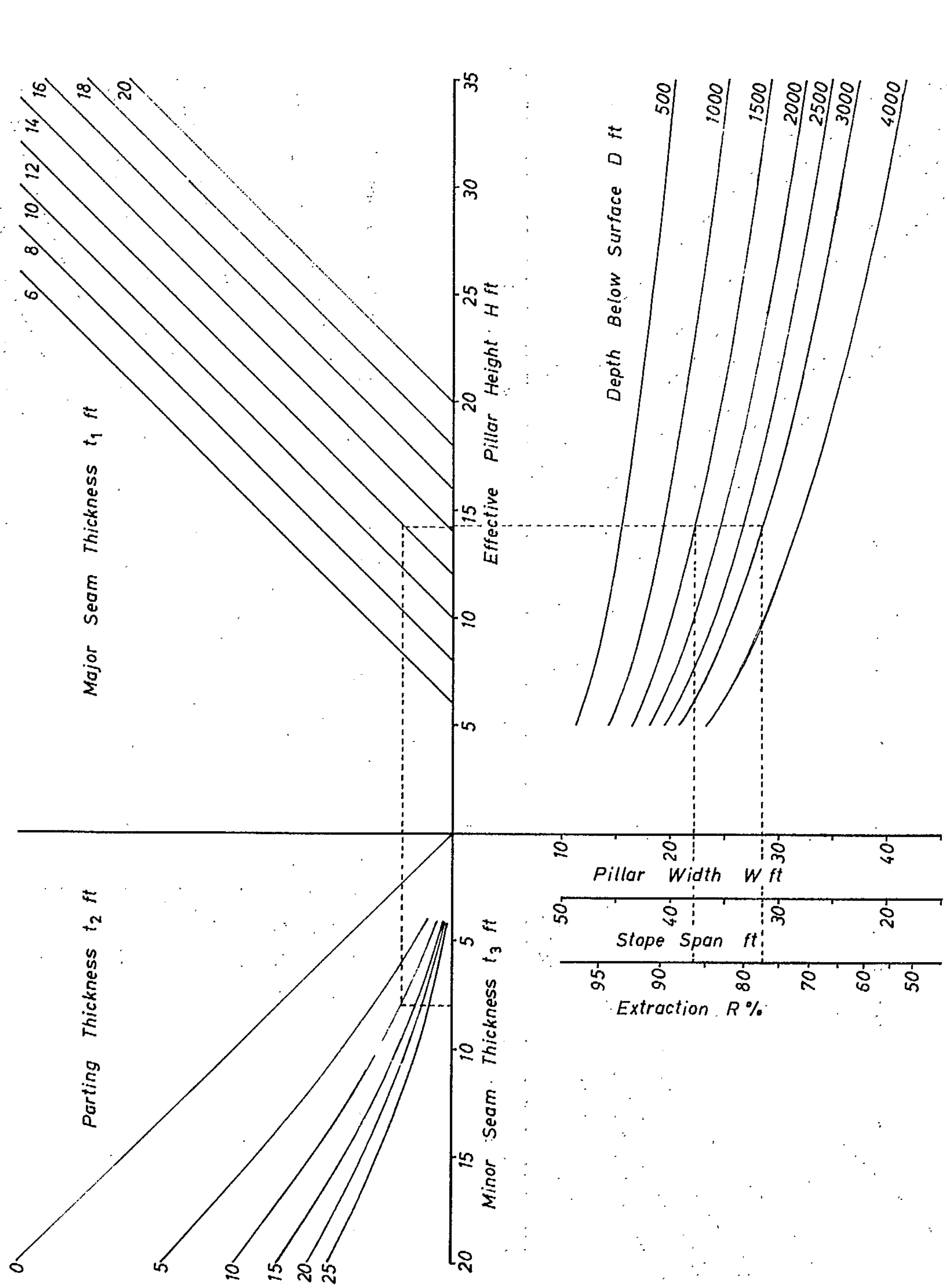


Fig. 6 - Design nomograph for square pillars spaced at 60-ft centres

The previous two sections described the design procedures for achieving stable pillars and parting zone. However, judgement is required in choosing the type of pillar and the pillar spacing, which in turn determines stope span. It has been shown that square pillars allow about 5% more extraction than rib or sill pillars, but because of trackless equipment requirements, their use is more or less restricted to areas where the dip is less than 10° and to where seam thickness is more than 10 ft.

In the 1960's, jackleg drills and slushers were the predominant methods of drilling and removing broken ore and the stopes were usually laid out on dip. In the 1970's, equipment changed and with it stope layout. Multi-drill jumbo's and long-hole drills are increasingly replacing jackleg drills. Scoop-trams and trucks are also replacing slushers. To operate this mobile equipment requires a dip of less than 10° and head room of at least 10 ft. At dips greater than 10°, rib pillars are laid out at an angle between strike and dip providing an apparent dip of 8°. Further advances in equipment design could occur which would permit operating with a head room of less than 10 ft. Slusher operations could thus be almost entirely eliminated and mobile equipment used for most orebody configurations. However, the economics of both methods as well as availability of skilled labour to operate the equipment have to be taken into account.

The effects of parting zone thickness, dip, and seam thickness on the choice of type and spacing of pillars, and on the option of trackless or slusher operations are summarized in Table 3a for multi-seam mining, and in Table 3b for single-seam and for both seams-and-parting mining.

Some basic criteria in selecting a mining method are:

a) Only rib pillars can be used in multi-seam mining to provide continuous support for the parting zone.
b) For a parting thickness of less than 9 ft, backfilling of the lower stope is required to support the parting zone and 75-ft pillar spacing can be used.
c) For a parting thickness of 9 to 13 ft, rib pillar spacing at 60-ft centres are

Table 3a: Basic design parameters for multi-seam mining

Parting zone thickness			Dip		Seam thickness		Pillar	Pillar	Mining
< 8 ft	8 to 12 ft	> 12 ft	< 10°	> 10°	< 10 ft	> 10 ft	type	spacing	requirements
X	-	-	X	-	X	-	rib	75 ft	backfill & slushers
X	-	-	X	-	-	X	rib	75 ft	backfill & trackless
X	-	-	-	X	X	-	rib	75 ft	backfill & slushers
X	-	-	-	X	-	X	rib*	75 ft	backfill & trackless
-	X	-	X	-	X	-	rib	60 ft	slushers
-	X	-	X	-	-	X	rib	60 ft	trackless
-	X	-	-	X	X	-	rib	60 ft	slushers
-	X	-	-	X	-	X	rib*	60 ft	trackless
-	-	X	X	-	X	-	rib	75 ft	slushers
-	-	X	X	-	-	X	rib	75 ft	trackless
-	-	X	-	X	X	-	rib	75 ft	slushers
-	-	X	-	X	-	X	rib*	75 ft	trackless

*rib pillars laid out at an angle between strike and dip.

Table 3b: Basic design parameters for single-seam and both-seams-and-parting mining

Dip		Seam thickness		Single seam			Both seams and parting		
< 10°	> 10°	< 10 ft	> 10 ft	Pillar type	Pillar spacing	Mining equipment	Pillar type	Pillar spacing	Mining equipment
X	-	X	-	rib	75 ft	slushers	square	60 ft	trackless
X	-	-	X	square	60 ft	trackless	square	60 ft	trackless
-	X	X	-	rib	75 ft	slushers	rib*	75 ft	trackless
-	X	-	X	rib*	75 ft	trackless	rib*	75 ft	trackless

*rib pillars laid out at an angle between strike and dip.

required.

d) For a parting thickness over 13 ft, rib pillars can be spaced at 75-ft centres.

e) For a dip or apparent dip less than 10°, trackless equipment can be used.

f) For a seam thickness greater than 10 ft, trackless equipment can be used.

RECOVERY AND ECONOMIC FACTORS IN MULTI-SEAM MINING

The parting zone between the upper and lower seams in some cases contains uranium, but of a lower grade. This requires a decision on whether to mine only the major single seam, the upper and lower seams, or both seams plus the parting zone. Since the effective pillar height is greater in two-seam mining, and especially when the parting zone is also mined, larger pillar widths are required in these two mining layouts; this in turn reduces the total amount of uranium recovered from the higher grade upper and lower seams. A decision on which of the three mining layouts is to be adopted has to be made before mining commences in a block of stopes because, if only a single seam is mined, the small pillar width could cause failure if either the other seam or the parting zone were to be mined at a later date. Also, the sequence of extraction is different: in multi-seam mining the lower seam is mined first, whereas when mining both seams and parting, the upper seam is mined first. At present, mines use a cut-off grade criterion to determine whether to mine a particular seam. Maximizing uranium recovery, minimizing costs and especially maximizing the economic return per stope are alternative methods to determine which mining layout to use. These factors are examined in the sections which follow:

Figure 7 illustrates the three alternative layouts. For the example chosen, rib pillars would be spaced at 75-ft centres for single-seam and for both-seams-and-parting mining, and at 60-

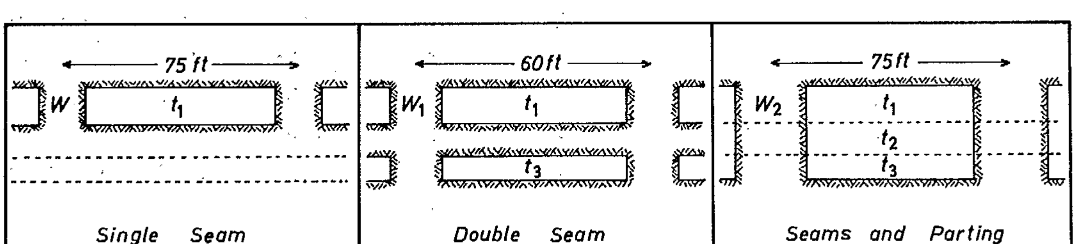


Fig. 7 - Alternative mining layouts

ft centres for double-seam mining. The upper seam, parting zone, and lower seam have thicknesses of t_1, t_2, and t and ore grades of u_1, u_2, and u_3 respectively. It is assumed that the upper seam is the major seam and would be the one mined in single-seam mining. In the following series of examples, a grade of 2 lb/ton U_3O_8 is given for u and the ore grades required in the other seam and parting zone are determined.

Maximum Uranium Recovery

The uranium recovery in each mining layout is directly proportional to extraction ratio, R, seam thickness, t, and ore grades, u, as follows :

single seam : $R_1t_1u_1$

double seam : $R_2(t_1u_1 + t_3u_3)$

seams and parting : $R_3(t_1u_1 + t_2u_2 + t_3u_3)$ (a)

This relationship can be used to determine which mining layout produces the maximum uranium recovery for various depths, seam thicknesses, and ore grades, as illustrated in the following example:

Example

A 30-ft thick combined orebody at a depth of 2000 ft has t_1 = 12 ft, t_2 = 10 ft, and t_3 = 8 ft with the ore grade, u_1 = 2.0 lb/ton. From Fig. 4 and 5, the effective pillar heights and corresponding pillar and stope widths and extraction ratios are:

	Effective pillar height	Pillar width	Stope width	Extraction ratio
Single seam	12.0 ft	14.4 ft	60.6 ft	0.808
Double seam	14.3 ft	13.5 ft	46.5 ft	0.775
Seams and parting	30.0 ft	22.8 ft	52.2 ft	0.696

Using these values in the derived relationship gives:

single seam : 0.808 x 12 x 2

double seam : $0.775(12 \times 2 + 8u_3)$

seam and parting : $0.696(12 \times 2 + 10u_2 + 8u_3)$ (b)

Expressing these values in terms of single seam mining gives:

$$1.0 : \frac{24 + 8u^3}{25.02} : \frac{24 + 10u^2 + 8u^3}{27.86} \qquad (c)$$

The uranium recovered for various values of u_2 and u_3 are plotted in Fig. 8. The results show that ore grade in the lower seam has only to be as high as 0.13 lb/ton for more uranium to be recovered by double-seam rather than by single-seam mining. If grade in the lower seam is 1.0 lb/ton, then grade in the parting zone should be more than 0.36 lb/ton before mining the parting zone would increase total uranium recovery.

Minimum Costs

Another basis for deciding on a method is to determine which mining layout gives the minimum cost per pound of uranium produced. In single-seam and double-seam mining, the seams are extracted on a one-pass system and mining and milling costs per ton should be the same for both seams. When mining both seams and the parting zone, the upper seam is usually mined first on a one-pass system and the parting zone and lower seam are then benched. The mining costs per ton for benching should be lower than for the one-pass system.

Let A be the total cost per ton for the one-pass mining system and B the total cost per ton for benching. Using the same symbols as defined in the previous example, total costs in single-seam mining are directly proportional to:

$$A\,R_1t_1. \qquad (d)$$

The recovery of uranium is:

$R_1 t_1 u_1$. (e)

Consequently, the cost per pound of uranium is:

$$\frac{A\ R_1 t_1}{R_1 t_1 u_1}. \quad (f)$$

which simplifies to:

$$\frac{A}{u_1} \quad (g)$$

Similarly the cost per pound of uranium production in double-seam mining is:

$$\frac{A(t_1 + t_3)}{t_1 u_1 + t_3 u_3}. \quad (h)$$

and in mining both seams and parting zone it is:

$$\frac{At_1 + B(t_2 + t_3)}{t_1 u_1 + t_2 u_2 + t_3 u_3}. \quad (i)$$

Expressing these relationships in a ratio format gives:

single seam : $\frac{A}{u_1}$

double seam : $\frac{A(t_1 + t_3)}{t_1 u_1 + t_3 u_3}$

seams and parting : $\frac{At_1 + B(t_2 + t_3)}{t_1 u_1 + t_2 u_2 + t_3 u_3}$. (j)

This relationship can be used to determine which mining layout produces the minimum cost per pound

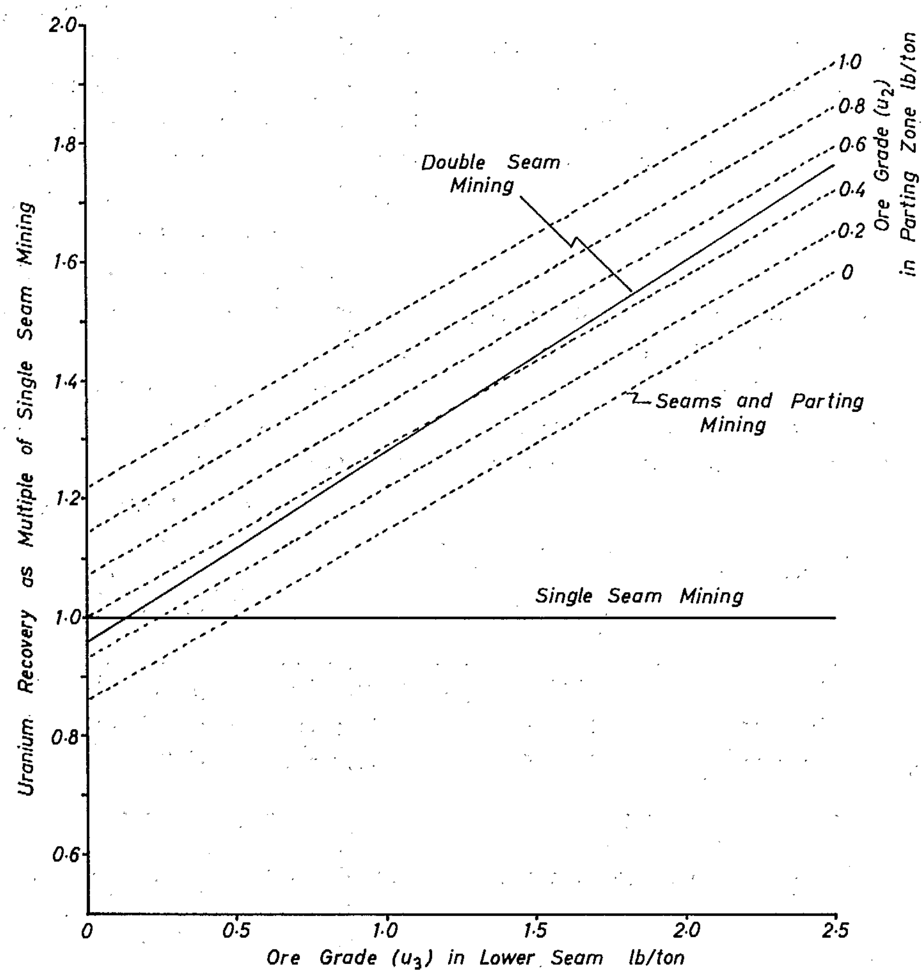


Fig. 8 - Maximum recovery of uranium

of uranium recovered for various ore grades and the difference in mining costs as illustrated in the following example.

Example

As before, $t_1 = 12$ ft, $t_2 = 10$ ft, $t_3 = 8$ ft, and u_1 is 2 lb/ton. The mining and milling costs per ton in benching operations are taken as 85% of those in the one-pass mining system (i.e., B = 0.85). Using these values in the derived relationships, and expressing in terms of single-seam mining gives:

single-seam	double-seam	seams-and-parting	
$\frac{A}{2}$:	$\frac{20A}{24 + 8u_3}$:	$\frac{27.3A}{24 + 10u_2 + 8u_3}$	(k)

$$1.0 \quad : \quad \frac{40}{24 + 8u_3} \quad : \quad \frac{54.6}{24 + 10u_2 + 8u_3} \,. \qquad (1)$$

The cost per pound of uranium for various ore grades, u_2, and u_3, are plotted in Fig. 9, expressed as a multiple of the cost of single-seam mining. The results show that ore grade, u_3, in the lower seam has to be greater than 2.0 lb/ton before costs are minimized by double-seam mining. For the range of ore grades, u_2, assigned to the parting zone, the costs of seams-and-parting mining are never cheaper than either single-seam or double-seam mining.

Maximum Return per Stoping Block

A third basis for deciding on a method is to determine which mining layout produces the maximum

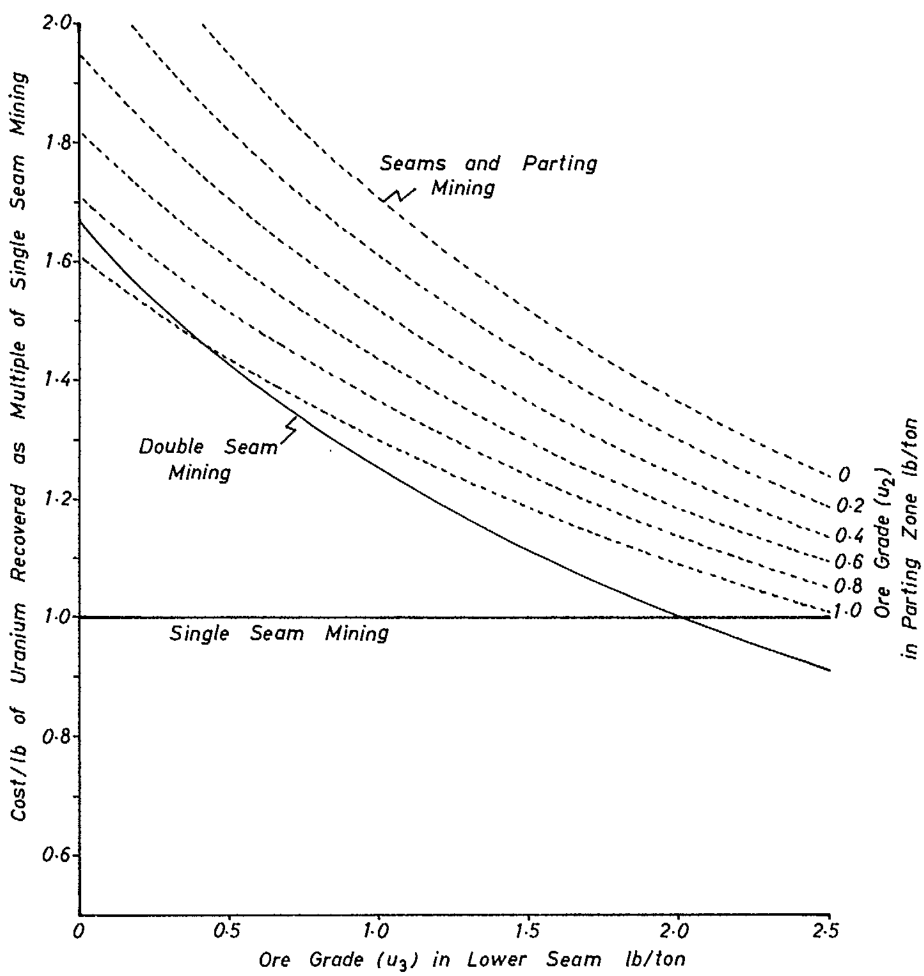


Fig. 9 - Minimum cost per pound of uranium recovery

return per stope. By this is meant that a pound of uranium will have a certain value which will be higher than the cost of producing it. The difference between value and cost multiplied by the pounds of uranium recovered will give the return per stope for each mining layout.

Let C be the value per pound of uranium, and as before, let A be the total cost per ton in a one-pass mining system and B be the total cost in benching operations. In single-seam mining the return is proportional to:

$$\left(C - \frac{A}{u_1}\right) R_1(t_1u_1). \qquad (m)$$

which can be rearranged to:

$$(Cu_1 - A)\, R_1t_1. \qquad (n)$$

In double-seam mining the relationship is:

$$\left[C - \frac{A(t_1 + t_3)}{t_1u_1 + t_3u_3}\right] R_2(t_1u_1 + t_3u_3). \quad (o)$$

which can be rearranged to:

$$[C(t_1u_1 + t_3u_3) - A(t_1 + t_3)]\, R_2. \quad (p)$$

When mining both seams and parting zone the relationship is:

$$\left[C - \frac{At_1 + B(t_2 + t_3)}{t_1u_1 + t_2u_2 + t_3u_3}\right] R_3(t_1u_1 + t_2u_2 + t_3u_3). \qquad (q)$$

which can be rearranged to:

$$\{C[t_1u_1 + t_2u_2 + t_3u_3] - [At_1 + B(t_2 + t_3)]\}R_3. \qquad (r)$$

These derived relationships can now be used to determine which mining layout maximizes the return for various ore grades, ratios of mining costs, and ratios of cost to value per pound of uranium as illustrated in the following example.

Example

As before at a depth of 2000 ft, t_1 = 12 ft, t_2 = 10 ft, and t_3 = 8 ft with u_1 = 2 lb/ ton. Extraction ratios are $R_1 = 0.808$, $R_2 = 0.775$ and $R_3 = 0.696$. Cost of benching is taken as $B = 0.85A$ and the cost of operations on a one pass system compared with the value per pound of uranium is taken as $A = C$.

Substituting these values into the three relationships and expressing in a ratio format gives:

single seam : 9.70A
double seam : $0.775A(4 + 8u_3)$
seams and parting : $0.696A(10u_2 + 8u_3 - 5)$. (s)

Expressing these values in terms of single-seam mining gives:

$$1.0 \;:\; \frac{1 + 2u_3}{3.13} \;:\; \frac{10u_2 + 8u_3 - 5}{13.94}. \qquad (t)$$

The return for various ore grades of u_2 and u_3 are plotted in Fig. 10. The results show that the ore grade, u_3, in the lower seam should be greater than 1.06 lb/ton before the return is higher in double-seam than in single-seam mining.

For the range of ore grades, u_2, assigned to the parting zone, the return from seams-and-parting mining is always less than either single-seam or double-seam mining.

Mining with Backfill and Separate Disposal of Parting Zone

At the Milliken mine it was observed that collapse of the parting zone occurred at a number of locations where the parting thickness was 8 ft and stope span 45 ft. Hence, a conventional double-seam mining layout is not suitable under these circumstances, but there are a number of alternatives:

a) The lower seam is mined first and backfilled with uncemented hydraulic tailings. Then the top seam is mined and if the parting zone fails it would merely rest on top of the backfill. Because the backfill will support the parting zone, rib pillars can be spaced at 75-ft centres rather than 60-ft centres.
b) As before, both seams and the parting zone are mined and all material is sent through the mill.

c) The seam is mined first, then the parting zone, and this material is disposed of in mined-out workings.

The return per stoping block for each of these alternatives can be compared.

As in the previous section, the return when mining both seams and parting zone is proportional to:

$$\{C[t_1u_1 + t_2u_2 + t_3u_3] - [At_1 + B(t_2 + t_3)]\}R_3. \quad (r)$$

In double-seam mining and placing backfill in the lower seam, let J be the total cost per ton of mining.
The return can then be expressed as:

$$[t_1(Cu_1 - A) + t_3(Cu_3 - J)]R_2. \quad (u)$$

For separate disposal of the parting zone, let E be the total cost per ton of mining and disposing of this material. The return can then be expressed as:

$$[t_1(Cu_1 - A) - Et_2 + t_3(Cu_3 - B)]R_3. \quad (v)$$

As illustrated in the following example, these relationships can determine which alternative should be used.

Example

As before, at a depth of 2000 ft, $t_1 = 12$

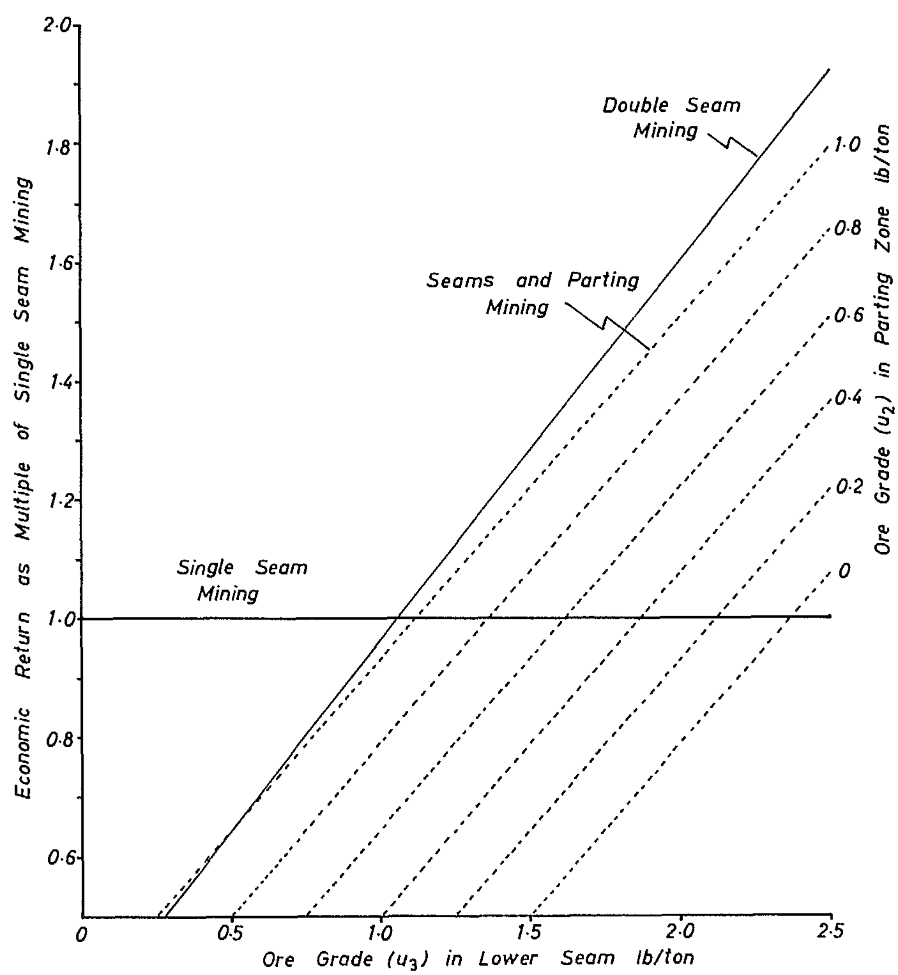


Fig. 10 - Maximum economic return per stoping block

ft, t_3 = 8 ft, and u_1 = 2 lb/ton, but in this case t_2 = 8 ft and for double-seam mining the pillars are spaced at 75-ft centres. From Fig. 5 for double-seam and both-seams-and parting mining respectively the effective pillar heights are 14.7 and 28 ft, pillar widths are 16.0 ft and 22.0 ft, and the extraction ratio $R_2 = 0.787$ and $R_3 = 0.707$. The cost of mining with backfill is taken as $J = 1.1A$ and the cost of separate mining and disposal of the parting zone is taken as $E = 0.6A$ and as before $B = 0.85A$ and $C = A$. Substituting these values in the derived relationships gives:

double-seam with backfill : $0.787A(3.2 + 8u_3)$
seams and parting : $0.707A(8u_2 + 8u_3 - 1.6)$
seams and parting with
separate disposal : $0.707A(8u_3 + 0.4)$. (w)

which simplifies to:

$$3.56 + 8.91u_3 : 8u_2 + 8u_3 - 1.6 : 8u_3 + 0.4. \quad (x)$$

When the return from double-seam mining with backfill and mining both seams and parting are equal:

$$3.56 + 8.91u_3 = 8u_2 + 8u_3 - 1.6. \quad (10)$$

which simplifies to:

$$u_2 = 0.645 + 0.114u_3. \quad (11)$$

Hence, if the ore grade, u_3, in the lower seam is 1 lb/ton then the ore grade, u_2, in the parting zone should be over 0.76 lb/ton before mining the parting zone rather than backfilling the lower stope.

Comparing double-seam mining and backfill with separate disposal of the parting zone for the conditions chosen, indicates that backfilling the lower stope always gives a higher economic return.

If the choice lies between separate disposal of the parting zone or sending this material through the mill then:

$$8u_2 + 8u_3 - 1.6 = 8u_3 + 0.4. \quad (12)$$

which gives u_2 = 0.25 lb/ton.

Consequently, if the ore grade in the parting zone is over 0.25 lb/ton, then a higher return is obtained by milling this material rather than disposing of it separately.

Other alternatives which could be tested include:

a) mining the lower seam first, then blasting down the parting zone to form a type of rock fill, followed by mining the upper seam;
b) when the seam thickness is less than 10 ft, taking enough of the parting zone to allow the use of trackless rather than slusher equipment.

DESIGN GUIDELINES

Procedures for determining parting zone stability, pillar stability, pillar shape and spacing, and the economic rate of return can now be formulated to provide design guidelines. These guidelines can be more readily understood by applying them in an example, indicating decisions and calculations required at each step.

Step 1. Divide the ore deposit into blocks based on continuity of individual reefs and ore grades and obtain information on average depth, seam and parting thicknesses, and ore grades for each block.

Depth 3200 ft; dip 10°; upper seam t_1 = 10 ft thick and u_1 = 2.25 lb/ton; lower seam t_3 = 7 ft and u_3 = 1.75 lb/ton; parting zone t_2 = 10 ft, and u_2 = 0.60 lb/ton.

Step 2. Decide on pillar shape and spacing, and on mining methods and sequence.

For existing equipment, Tables 3a and 3b can be used.

Double-seam mining: rib pillars on dip spaced at 60-ft centres, trackless equipment used in upper seam, slushers in lower seam.

Single-seam (upper) mining: square pillars spaced at 60-ft centres; trackless equipment used.

Seams-and-parting mining: square pillars spaced at 60-ft centres; trackless equipment used.

Step 3. Calculate pillar and stope dimensions and extraction ratio for each mining layout.

Using equipment 7 and 9, or Fig. 4 and 6.

	Single-seam	Double-seam	Seams and parting
Effective pillar height (H)	10 ft	11.8 ft	27 ft
Pillar width W	26.3 ft	16.5 ft	35.4 ft
Stope width L	33.7 ft	43.5 ft	24.6 ft
Extraction	R_1 = 80.8%	R_2 = 72.5%	R_3 = 65.2%

Step 4. Calculate percentage of the contained uranium.

The total uranium contained in a stoping block is proportional to:

$$t_1u_1 + t_2u_2 + t_3u_3. \qquad (y)$$

Substituting the relevant values gives:

$$10 \times 2.25 + 10 \times 0.60 + 7 \times 1.75 = 40.75.$$

Using relationship (a) and the relevant values, the uranium recovery for each mining layout is as follows:

single seam : $R_1t_1u_1$
: 18.18
: 44.6%

double seam : $R_2(t_1u_1 + t_3u_3)$
: 25.2
: 61.8%

seams and parting : $R_3(t_1u_1 + t_2u_2 + t_3u_3)$
: 26.6
: 65.2%

Step 5. Estimate mining costs and selling price of uranium.

The costs can either be total costs including overhead in which case the selling price of uranium is used, or only mining and milling costs in which case the value per pound of uranium at the mine gate is used.

Cost of slusher operations/ton = A
Cost of benching operations/ton, B = 0.85A
Price of uranium,/lb, C = A
Cost of backfilling operations/ton, J = 1.1A
Cost of separate disposal of parting zone/ton, E = 0.6A
Cost of trackless operations/ton, F = 0.9A

The cost could equally well be expressed in terms of trackless operations rather than slusher operations.

Step 6. Calculate economic return for each mining layout.

a) For single-seam mining using relationship (n) the return is:

$$(Cu_1 - F)\ R_1t_1$$

Substituting the relevant values gives:

$$(A \times 2.25 - 0.9A)\ 0.808 \times 10 = 10.91A$$

b) For double-seam mining, breaking down by individual seams the return is:

for upper seam = $(Cu_1 - F)\ R_2t_1$
for lower seam = $(Cu_3 - A)\ R_2t_3$

Substituting the relevant values gives:

$(A \times 2.25 - 0.9A)\ 0.725 \times 10$
$+ (A \times 1.75 - A)\ 0.725 \times 7$
$9.79A + 3.81A = 13.60A$

c) For both-seams-and-parting mining, breaking down by individual seam the return is:

for upper seam = $(Cu_1 - F)\ R_3t_1$
for parting zone = $(Cu_2 - B)\ R_3t_2$
for lower seam = $(Cu_3 - B)\ R_3t_3$

Substituting the relevant values gives:

$(A \times 2.25 - 0.9A)\ 0.652 \times 10$
$+ (A \times 0.60 - 0.85A) 0.652 \times 10$
$+ (A \times 1.75 - 0.85A)\ 0.652 \times 7$

$8.80A - 1.63A + 4.11A = 11.28A$

The economic returns expressed in terms of single-seam mining and the percentage recovery of uranium are as follows:

	single-seam		double-seam		seams and parting
Economic return	1.0	:	1.25	:	1.03
Uranium recovery	44.6%	:	61.8%	:	65.2%

Consequently, if mining were to proceed in the immediate future, a double-seam layout would give the greatest economic return and a uranium recovery only 3.4% less than seams and parting mining.

The effect of a change in the price of uranium on the economic return is illustrated in Fig. 11. Single-seam mining gives the greatest econo-

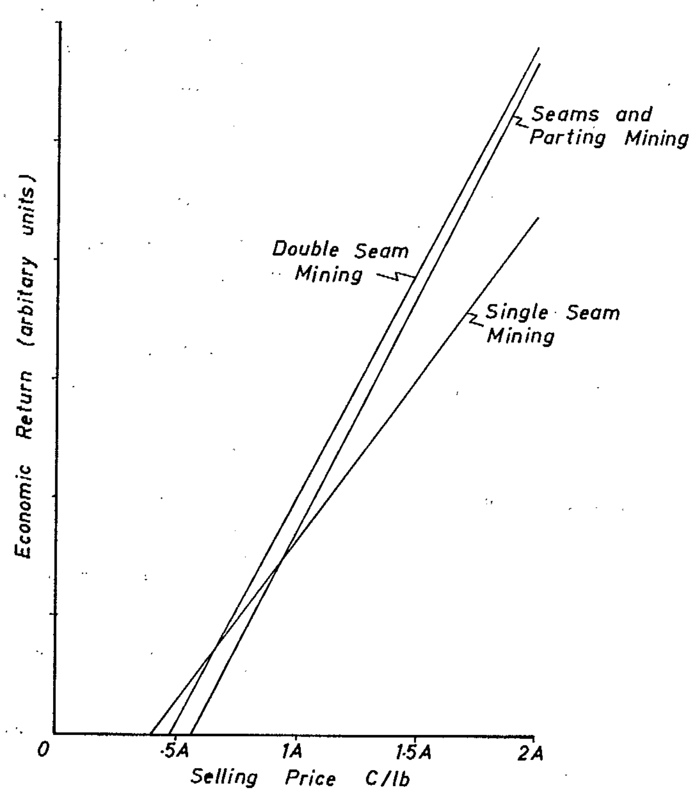


Fig. 11 - Effect of change in selling price on economic return

mic return when the price, C, is below 0.62A, whereas above this value double-seam mining has the greatest economic return. The critical cost favouring either single-seam or seams-and-parting mining occurs when the price, C = 0.95A.

Step 7. Check whether backfilling the lower stope or separate removal of the parting zone is economically more viable.

If backfill is used to support the parting zone, in the event of its failure the rib pillars in double-seam mining can be spaced at 75-ft centres. From Fig. 5, pillar width is 19.1 ft and extraction ratio $R_2 = 0.745$. Breaking down by individual seams, the return is:

$$\text{for upper seam} = (Cu_1 - F)\ R_2 t_1$$
$$\text{for lower seam} = (Cu_3 - J)\ R_2 t_3$$

Substituting the relevant values gives:

$$(A \times 2.25 - 0.9A)0.745 \times 10$$
$$+ (A \times 1.75 - 1.1A)0.745 \times 7$$

$$10.06A + 3.39A = 13.45$$

Consequently, the economic return from double-seam mining with backfill is only marginally less than without backfill, i.e., 13.45A compared with 13.60A.

For separate removal of the parting zone, the economic return for individual seams are:

$$\text{for upper seam} = (Cu_1 - F)\ R_3 t_1$$
$$\text{for parting zone} = -\ ER_3 t_2$$
$$\text{for lower seam} = (Cu_3 - B)\ R_3 t_3$$

Substituting the relevant values gives:

$$(A \times 2.25 - 0.9A)0.652 \times 10$$
$$- 0.6A \times 0.652 \times 10$$
$$+ (A \times 1.75 - 0.85A)0.652 \times 7$$

$$8.80 - 3.91A + 4.11A = 9.00A$$

This economic return is lower than that calculated for any other alternative and hence separate disposal of the parting zone is not viable.

A similar analysis could be done using the pricing formula developed for the contracts with Ontario Hydro. In a somewhat simplified form the selling price is the cost of producing a pound of uranium plus five dollars, giving a base price, plus a fraction of the difference between the world and base price, e.g., 1/2 in the case of one company and 1/3 for another. The pricing formula, using the lower fraction, can be expressed as:

$$\text{Selling price/lb} = \frac{\text{cost/ton}}{u} + 5 + \frac{G - \left(\frac{\text{cost/ton}}{u} + 5\right)}{3} \qquad (13)$$

where, G = world price/lb
u = ore grade lb/ton.

The economic return per lb of uranium is:

$$5 + \frac{G - \left(\frac{\text{cost/ton}}{u} + 5\right)}{3}$$

In single-seam mining the economic return for stoping block will be proportional to:

$$\left\{5 + \left[\frac{G - \left(\frac{F}{u_1} + 5\right)}{3}\right]\right\} R_1 u_1 t_1$$

which, for the working example gives:

$$60.6 + 3.64\ G.$$

Similarly the economic returns for the other mining layout can be calculated to give:

double-seam: $83.96 + 4.53\ G$
seams and parting: $88.56 + 3.76\ G$

The effect of a change in the world price of uranium on the economic return is illustrated in Fig. 12. In this case it is necessary to assign a value to the total cost of mining a ton of ore, A, and a typical value was chosen. For the conditions analyzed, the results indicate that double-seam mining gives the greatest economic return although the difference is small between it and seams-and-parting mining. There is a lower limit beyond which the pricing formula is inoperable, occurring when the base and world prices are equal. For comparison, the economic return using a full world price criteria, same as shown in Fig. 11, is also illustrated. As expected, the lower limit of the Hydro pricing formula occurs when the two sets of lines intersect.

In conclusion, this engineering analysis of multi-seam mining at Elliot Lake provides planning engineers with a set of design procedures and guidelines. Essentially, they provide reasonable estimates of what can be achieved for a given set of conditions. However, they are not "carved in stone" and once mining starts, actual conditions will dictate required modifications.

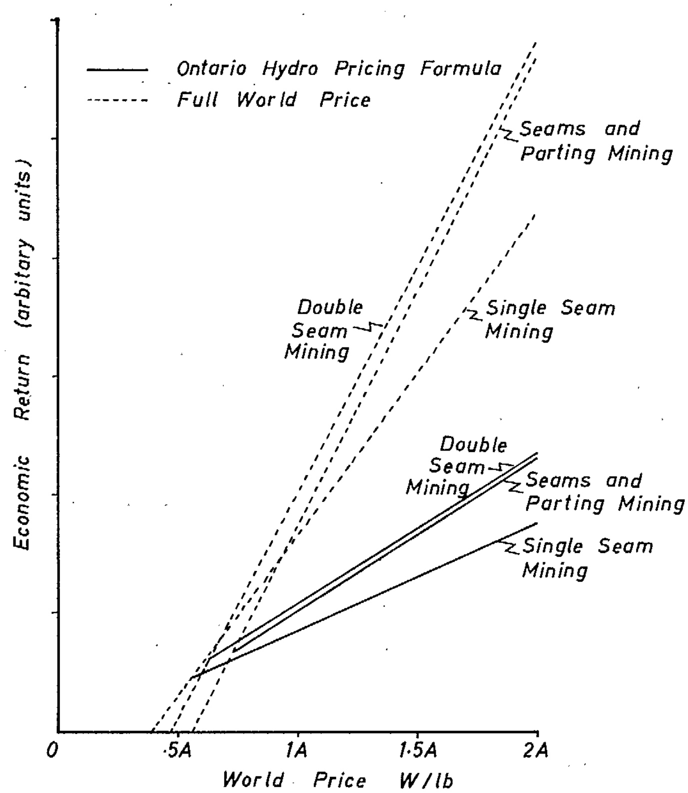


Fig. 12 - Effect of change in world price on economic return

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SYMBOLS

σp = average pillar stress, psi

Qu = average pillar strength, psi

S.F.= safety factor

W = pillar width, ft

H = effective pillar height, ft

L = stope span, ft

D = depth below surface, ft

α = dip of orebody, degrees

R = extraction ratio

t_1 = thickness of major seam (either upper or lower), ft

t_2 = thickness of parting zone, ft

t_3 = thickness of minor seam (either upper or lower), ft

u_1 = ore grade of major seam, lb/ton

u_2 = ore grade of parting zone, lb/ton

u_3 = ore grade of minor seam, lb/ton

A = total cost* of slusher mining, $/ton

B = total cost of benching operations, $/ton

C = selling price of uranium, $/lb

J = total cost of mining and backfill operations, $/ton

E = total cost of mining and separate disposal of parting zone, $/ton

F = total cost of trackless mining operations, $/ton

G = world price of uranium, $/lb

* all cost figures include mining, milling and overhead

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