



June 26, 1990
Project Number 60619

Curragh Resources Ltd.
117 Industrial Road
Whitehorse, Yukon
Y1A 2T8

Attention: Mr. Leo Hwozdyk

Dear Leo:

RE: Room and Pillar Review

The following is a brief report of the visit by Chris Page and Jim Mathis to the room-and-pillar operations at Faro during the week of 19 July.

The following points are briefly discussed under separate headings:

- underground stability
- rock support
- blasting
- pillar design procedure
- initial room-and-pillar layouts and procedures

The overall impression is of reasonable stability within a generally competent rock mass. In the southern section, through, the excavations usually have a sulphide roof. Where hanging wall material is exposed, patterned roof support is necessary. There will be a considerable difference between the present conditions and final extraction when the room size will have doubled and when most of the roof will be in hanging wall material. The present standards of excavation control and support installation are not adequate. Serious considerations should be given to leaving ore in the roof to reduce support requirements.

Mining cost is a key element in the operation. Operating costs will be sensitive to the room size but as the room size increases the need for more intensive support will quickly override the savings in production costs. The operating costs will be extremely sensitive to room stability. It may be



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necessary to limit percentage extraction in favour of overall cost reduction. This factor is addressed in the section on layouts.

The mining operations are proceeding in a fashion typical of a tunnelling contract. Room-and-pillar operations require much more careful control. As you are aware, the operations in Elliot Lake depend on control of continuous roof beams, careful cutting of pillars, efficient scaling of the roof and a tight pattern of well installed bolts. This level of care is not evident.

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Finally, the use of numerical analysis technique to improve the calculation of pillar stresses, rather than using "tributary area theory", could increase the percentage extraction from approximately 70% to 87%. The analysis could be carried out in two phases; the first to demonstrate the potential reduction in pillar stresses and the second to evaluate actual pillar stresses. The approximate cost of the first phase would be three thousand dollars.

Yours truly,

STEFFEN, ROBERTSON AND KIRSTEN (B.C.) INC.

C.H. Page, P. Eng.

CHP/007cs

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1. Underground Stability

The underground excavations are generally very stable and safe. The ore material is strong and competent and most of the development is in ore. There are very few exposures of the 'true' waste hanging wall. Where these exposures occur the roof is often incompetent. Mineralised graphitic quartzite is very similar to the ore, especially where it has been silicified.

A detailed review of the southern area was undertaken. Figure 1 shows a plan of the area with rock type and a measure of the roof stability. This indicates that 55% of the 'true' hanging wall requires fairly intensive conditioning. Of approximately 2500 feet of development in the southern area only 15% has a non-sulphide roof.

In the SRK report 60610, December 1988 it was estimated that some 35 to 40% of the roof would require intensive support (bolting, mesh, straps and/or shotcrete. This value is supported by the conditions seen in the southern area with reference to the limited amounts of hanging wall that have been exposed.

The decision has to be made, as discussed in the section on roof support, whether to leave a 'skin' of sulphide ore against the hanging wall. This skin would have to be at least 3 feet thick with 30 foot rooms. There is not a clean, physical break between high grade massive sulphide and the graphitic quartzite but when very thin wedges of sulphide are left in the roof they tend to fall. But there are many instances of a 'mixed' ore and waste roof, Figure 2. The concern with thin bands of ore is that they may support, temporarily, very weak material. There is an instance of extremely weak, highly graphitic quartzite which has caused a large roof fall, Figure 2. The slabs of ore left in the roof must be stable enough to span the room and hold patches of weak strata in place. Whether this will be possible given the frequency of faulting and folding remains to be seen.

There is no equipment on site which would enable inspection and repair of excavation roofs if the full orebody thickness is extracted. Therefore inaccessible roofs must be "oversupported" and should include mesh or shotcrete to catch the smaller fragments. Where mesh is used it should be pulled tight to the back by installing bolts in hollows. In this way the mesh will add to the support effort.

Pillar stability is very good, as expected in the competent sulphides. But the maximum exposure is about 15 feet and very little extraction has taken place. Pillar heights up to 50 feet may occur but there is no indication of continuous structures with weak infilling that could intersect pillars at unfavourable angles. It is evident that room-and-pillar layouts will be dictated by room stability.

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There is an important difference between the conditioning of excavation roofs and pillar walls. A roof should be well scaled before support. Pillar walls should not. Instead, the loose material should be well supported. There is a risk of scaling too much material from a pillar wall and significantly reducing its effective size. Where pillar walls become inaccessible, due to orebody thickness, they must be supported.

Further
classification
required.

2. Rock Support

The predominant support method has been split-sets. Straps have been added in the poorer areas. The overall impression is that the holes for the split-sets are too large and there is insufficient closure on the split. It is reported, though, that the operators were having difficulty in installing the split-sets because the holes were too small. There are a considerable number of split-sets hanging out of holes Figure 3. Many of these must have shaken loose during blasting. Some attempt at pull testing of randomly selected bolts should be carried out to assess the longer term stability of the installations.

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Recently there has been a conversion to mechanically anchored bolts. Discussion have revealed that there may be problems in security anchoring these bolts in the weaker strata. Our view is that the split-sets, if properly installed, are superior to the mechanical bolts. Both types are inferior to cement grouted bolts and these should be considered in the more critical areas. Especially in areas where a thick sulphide skin or sulphide rich waste is left in the back. When straps are installed it is most important that the straps should be pulled in tight to the rock surface by installing the bolts in hollows. The installation of the strap should proceed consecutively, one hole to the next. There are many instances of straps bridging large hollows in the rock surface.

The majority of the ground conditions are good but, to date, there have been few exposures of the hanging wall, the spans are relatively small and pillar stresses are very low. Much of the development is within the orebody and there is very competent sulphide material in the roof.

In the few areas in the south zone where the hanging wall has been exposed most of these areas have required straps and some areas are extremely incompetent. In the future when the final excavation exposes the hanging wall and much larger spans are mined then the requirement for correct, well installed support will become critical. The present level of scaling and quality of support will not be adequate for the planned exposures. It may be necessary to leave a skin of sulphides in the roof to reduce the support requirements.

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Most of the major problems with roof control have been with foliated material and the creation of unstable slabs. Part of the problem is that the development roof is not mined to a single plane. Overbreak and poor alignment cause an uneven roof, Figure 3. There were a number of instances

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where the roof has become loose after support installation. It is not known whether this is because of poor scaling or incompetent material. If it is a function of incompetent material then more intensive support methods, reference SRK report, 60610, December 1988, must be used.

The alternative is to maintain a minimum thickness of ore in the roof and this would appear to be the most economic option considering the blasting and support practices which are in use.

3. Blasting

It is evident that there is not auto-parallelism on the drill jumbos, or it is not in use, and that there are many instances of incorrect hole alignment. This problem is especially critical in the corner holes of the roof. These are consistently overbreaking due to large look-out angles. The roof often has a convex profile due to poor spotting of holes and differences in hole angle. These points are highlighted in the accompanying sketch, Figure 4. It does not appear that the hole pattern is formally marked-up at the face. The angle on the holes varies throughout the ground. The result of poor layout is an uneven roof and walls, extra confinement for some of the holes and overbreak. The resulting surfaces will, therefore, be less stable.

A review of the standard drill layout indicates two problem areas. These are identified in Figure 5 and concern incorrect timing and overcharging of holes close to the roof.

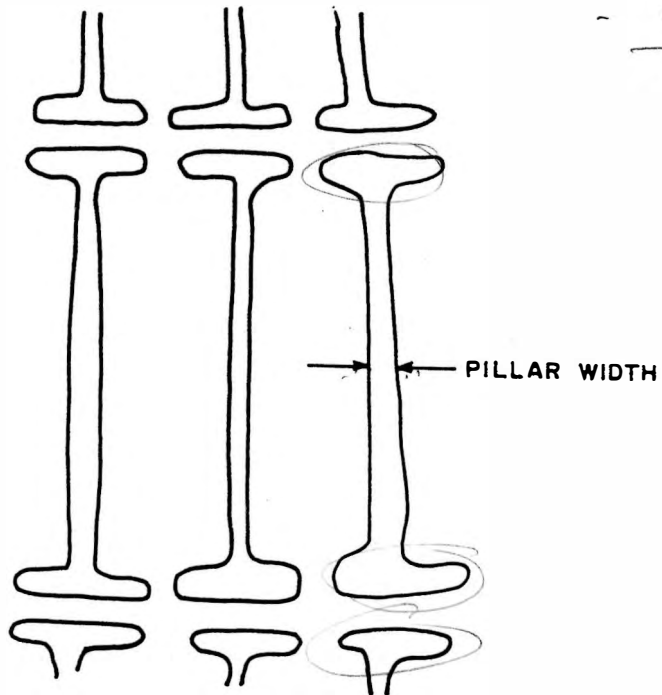
The layout from CMD, "typical stope mining" has been reviewed. The layout appears effective but far fewer holes could be used. Most of the holes should have a good free face if the blast is broken into stages. A powder factor of 1 kg/m³ should be adequate which is an average burden and spacing of 4.5 ft.

4. Pillar Design Procedure

The empirical, pillar design formula which has been used for this investigation has been developed from work carried out by Hedley in Canada and Salamon in South Africa. Attached is a description of the relationship.

It is important to note that Hedley did most of his work on pillars that had the following shape:

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Plan view of room and pillar layout at Elliot Lake

The pillar design formula takes the following form:

$$\text{Pillar strength} = \text{rock strength} \times \frac{(\text{pillar width})^{0.5}}{(\text{pillar height})^{0.75}}$$

But in Hedley's relationship the pillar width only relates to the width of the high, thin rib pillars. The short, fat pillars around the access level are not included. Long rib pillars are more stable than small square pillars given the same extraction ratio. Therefore some adjustments must be made if a square pillar layout is envisaged. This can be taken into account by using an "effective" pillar width, (described in the attachment):

$$W_{\text{eff}} = \frac{A_p}{R} \quad \text{where } \begin{matrix} A_p & = & \text{pillar plan area} \\ R & = & \text{pillar perimeter} \end{matrix}$$

If the "effective" pillar width is taken into account then the rock mass strength of 173 MPa, in Hedley's formula, becomes 122 MPa.

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A design relationship is only as good as the data and experience that are used in the development of that relationship. The data base for Faro is small and the experience, at present, is zero. But experience will only be available when large areas have been extracted to the final percentage extraction. There is not time to accumulate this experience.

In the interim it is suggested that the following design relationship should be used by site personnel:

• Safety factor =
$$\frac{8250 W_{\text{eff}}^{0.5}}{H^{0.7}}$$
 } pillar strength

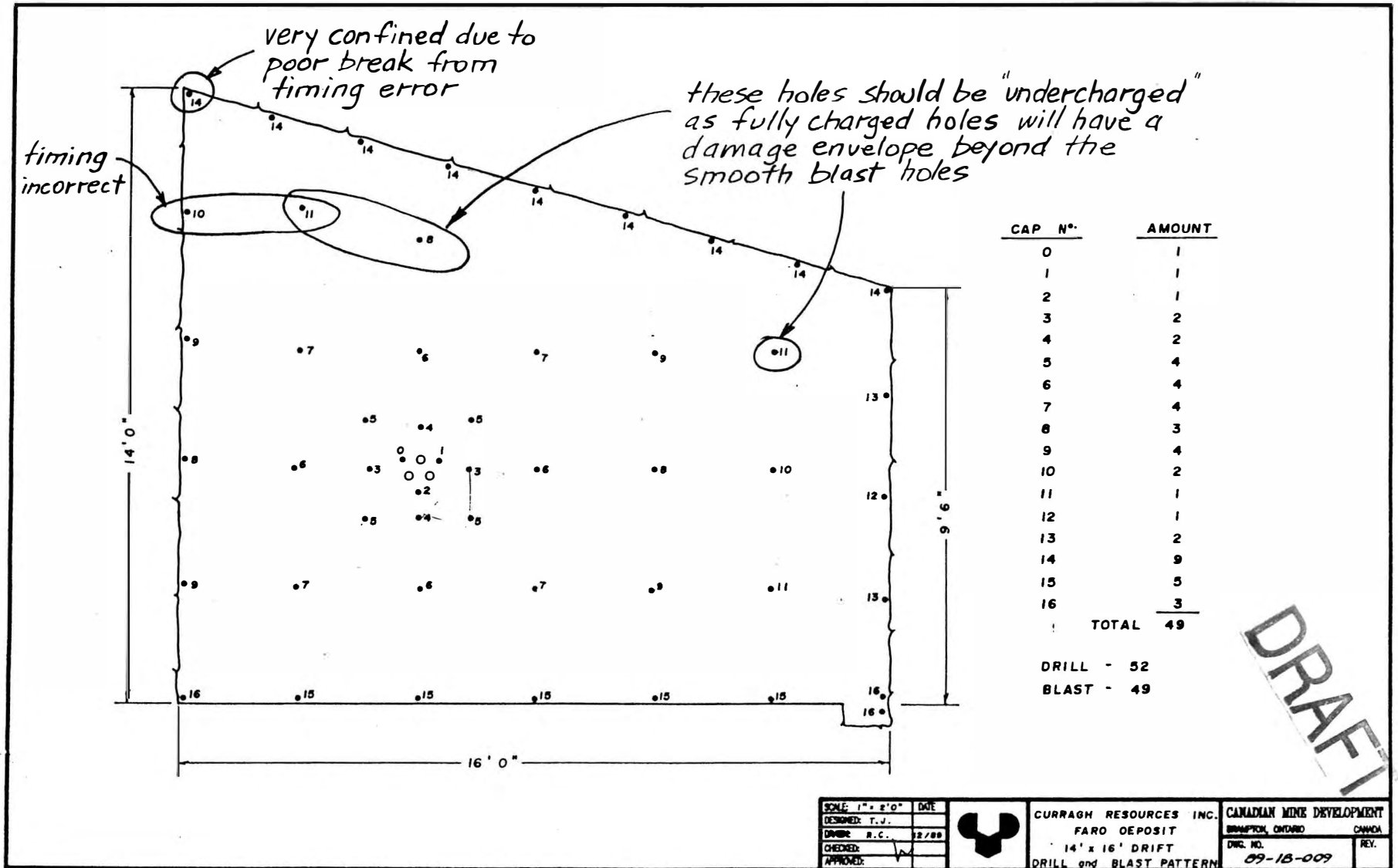
$$\frac{1.1 D \cos^2 \alpha + (1.1 D \times 2) s \sin^2 \alpha}{1-e}$$
 } pillar stress

Where W_{eff} = effective pillar width = $\frac{4A_p}{R}$ (ft)

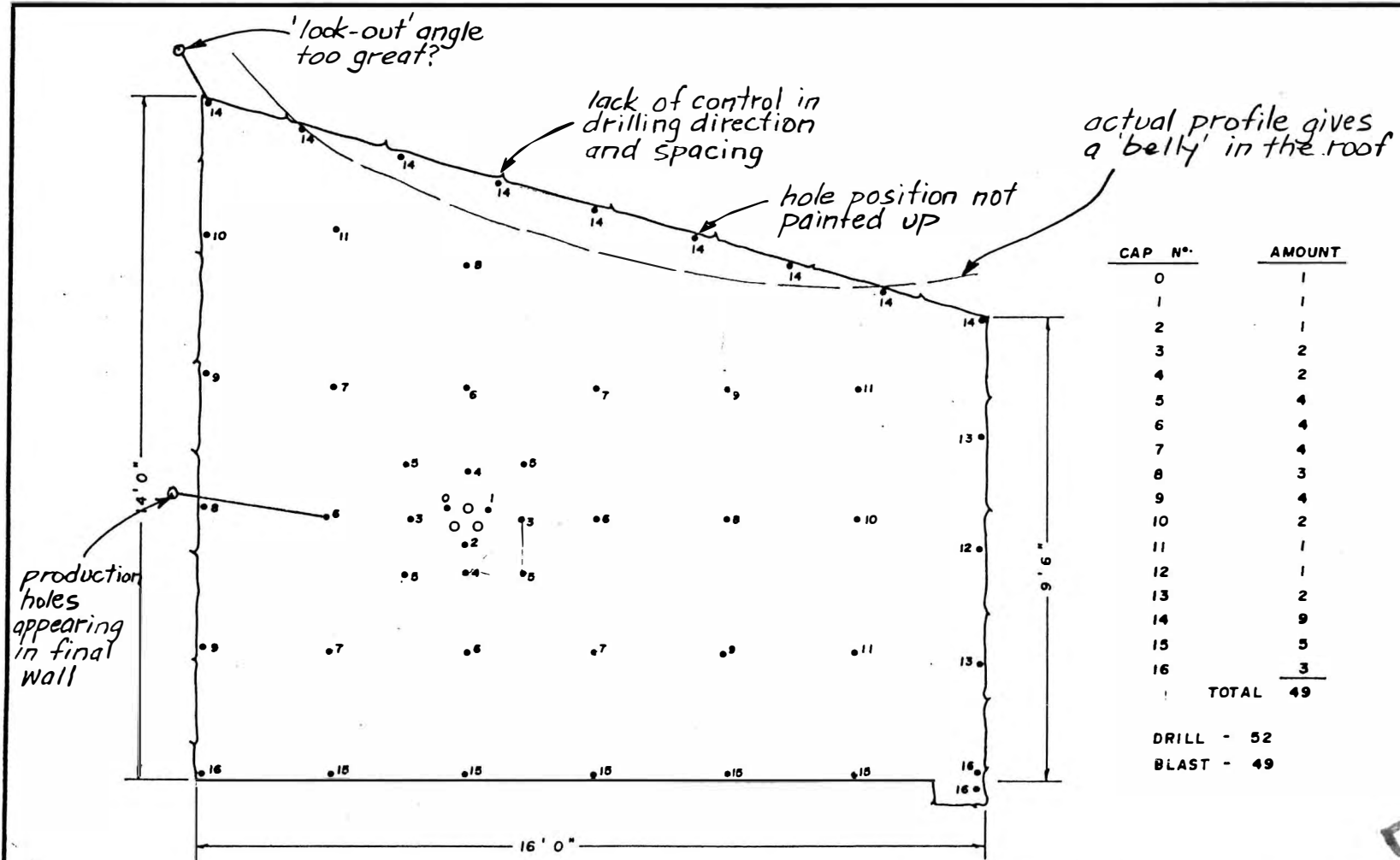
- H = pillar height, ft.
- D = depth below surface, ft.
- α = dip angle
- e = percentage extraction/100

The above equation assumes "tributary area theory", pillars carrying total overburden load. In practice, with a relatively small mining strike, the abutments will carry appreciable load and numerical analysis of the proposed mining system is recommended.

A factor of safety of 1.5 is suggested, to comply with recommendations from Hedley.



FIGURE



CAP N°	AMOUNT
0	1
1	1
2	1
3	2
4	2
5	4
6	4
7	4
8	3
9	4
10	2
11	1
12	1
13	2
14	9
15	5
16	3
TOTAL	49

DRILL - 52
BLAST - 49

SCALE: 1" = 8'0"	DATE
DESIGNED: T.J.	
DRAWN: R.C.	12/89
CHECKED: [initials]	
APPROVED: [initials]	



CURRAGH RESOURCES INC.
FARO DEPOSIT
14' x 16' DRIFT
DRILL and BLAST PATTERN

CANADIAN MINE DEVELOPMENT	
BRAMPTON, ONTARIO	CANADA
DWG. NO. 09-16-009	REV.

FIGURE