Royal Oak Mines Inc.

Proposal
to
Develop # 15 Arsenic Trioxide Storage Chamber
Giant Mine - Yellowknife NT
July 24th, 1998

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Prepared for:

WCB - Mines Inspection Services

Prepared by:

Royal Oak Mines Inc.
Giant Mine
Yellowknife NT

July 1998

Royal Oak Mines Inc.
Proposal to Develop #15 Arsenic Trioxide Storage Chamber

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1.0 Introduction:

General:

This report is submitted in support of a request for approval to construct the #15 Arsenic Trioxide Storage Chamber.

Since the first production of arsenic trioxide dust, resulting from the refining of refractory ores, Giant Mine has constantly re-evaluated and updated its arsenic trioxide disposal practices to ensure adherence to existing regulations and to maintain the practice of environmentally acceptable disposal methods.

Arsenic trioxide management plans since the early 1950's have included placement of arsenic trioxide dust underground in specially designed storage areas.

2.0 Site Description:

Physiography and Infrastructure:

The Giant Mine Site is situated approximately five kilometers north of the City of Yellowknife, adjacent to Great Slave Lake along the western shore of Yellowknife Bay. The mine is situated within a zone of discontinuous permafrost and the local topography is characterized by a series of exposed bedrock highs with minor overburden deposits in low lying areas.

Geological Setting:

The Giant Mine is located within the structurally complex Yellowknife Greenstone Belt of Archean Age. The belt extends from Great Slave Lake for a distance of more than 50 kilometers, and is comprised of a homoclinal steep easterly to vertically dipping sequence of metabasalts and metagabbros intruded by sheeted dykes and overlain by sedimentary units. The package of rocks was subsequently intruded by granitic intrusions.

Gold mineralization is present within the metabasalt units, associated with arsenopyrite mineralization. The rocks have undergone middle greenschist to middle amphibolite facies metamorphism. Arsenopyrite is a naturally occurring arsenic bearing mineral. The gold mineralization is refractory meaning that the arsenopyrite mineralization must be broken down and oxidized to allow the recovery of the gold.

The Yellowknife Greenstone Belt is a structurally complex sequence of rocks. Three prominent fault trends exist within the Giant Mine; 000 to 025° , 060° , and 160°, with the main structural features known as the Townsite Fault, the 3-12 Fault, and the West Bay Fault. The 160° faults are prominent faults with variable easterly dips and are characterized by clay fault gouge and breccia. The sense of movement on these faults is sinistral. The 060° faults are generally characterized by having little or no clay gouge and may appear as thin hairline fractures. The sense of motion on these faults is dextral and they dip to the west. Faults with the 060° trend may occur as major faults or appear as lesser faults. Water seepage into the mine generally occurs along the major fault zones.

Hydrogeological Setting:

Hydrogeologically, the groundwater regime present at the Giant Mine is controlled by the regional topography and complex structural geology. Groundwater movement will occur predominantly as fracture flow, as is evidenced from observations noted within the mine workings.

Little evidence or reports of original groundwater movement are available. The mine has been in production for 50 years and therefore the regional groundwater table has been severely depressed in the mine area.

Baker Creek is the main water course through the mine property, and discharges to Great Slave Lake at Yellowknife Bay, at the extreme south end of the property. The stream channel is a potential groundwater discharge zone. The other main sources of water entering the mine include modern precipitation recharge, seepage from the Northwest Pond, and leakage of mine water supply lines.

A preliminary hydrogeologic study has been completed recently by Fracflow Consultants Inc. for DIAND - Water Resources Division, with the full support of Royal Oak Mines. The report will be available in mid-August for review.

One of the conclusions of the study, based on isotopic investigations was that, *"Shallow groundwater in the UBC area, including areas currently utilized for storage of arsenic trioxide dust, is evidently derived mainly from modern precipitation recharge"*

Permafrost:

Documentation indicates that discontinuous zones of permafrost were encountered throughout various areas of the mine during development (1950's) and led to the initial plan to store the arsenic trioxide dust in secure chambers in a complete permafrost envelope.

No long term monitoring of bedrock temperatures and conditions is available from the mines data base.

Six diamond drill holes were drilled into the bedrock from surface to investigate the status of permafrost in the bedrock surrounding the storage chambers, in June of 1994. Five of the six holes are located in the vicinity of the active and inactive arsenic storage areas. The sixth hole was drilled in an area of bedrock not influenced by active mining conditions. In each of the holes a string of thermistors was installed to monitor rock temperatures at various depths. The strings read temperatures at surface, at a 20' depth below surface, and then at 55 foot intervals down until the bottom of each hole. An approximate depth of 350 feet was used as the bottom of each hole as no storage chamber is located below this elevation.

These thermistors have been read on a regular basis since installation in 1994 and have been reported regularly to the NWT Water Board.

Several conclusions are evident from the data:

1) Underground and surface mining activities at the Giant Mine have disturbed the existence of permafrost in the rock, such that rock temperatures range from +0.5 deg C to +3.5 deg. C in the approximate area of all storage chambers.

2) The influence of ambient temperature extends down to a depth of 15 feet below surface although there is a noticeable lag time between the surface temperature and the rock temperature.

3) At a depth of 70 feet and below, there is no measurable impact from ambient temperatures on the rock temperature.

3.0 Mine History:

General:

The first major discoveries of gold in the Yellowknife area were made in 1934. The original claims on the Giant property were staked in 1935 by C.J. Baker and H. Muir. Giant Yellowknife Mines was subsequently incorporated in 1937.

The mine has been in operation since 1948 producing more than seven million ounces of gold since the initial discovery.

The Giant Mine is operated primarily as an underground mine, at an approximate production rate of 1,100 tonnes/day. Several inactive open pits, mined in the 1980's, are also present on the site. The ore body has a strike length of more than 4500 meters and is currently accessed through a main production shaft and two ramp systems. Several inactive shafts, adits, ramps and raises are evident throughout the property. Mining is principally by mechanized cut and fill methods.

Arsenic Trioxide Management:

Arsenic trioxide dust is currently produced at the Giant Mine at an approximate rate of 9 to 12 tonnes per day as a byproduct of the gold milling operation.

The primary gold bearing mineral at the Giant Mine is arsenopyrite. The gold contained in this arsenopyrite is not recoverable until the arsenopyrite crystal lattice has been physically broken apart and the contained arsenic and sulphur mineralization is removed. The conversion process employed at the Giant Mine consists of high temperature roasting of an arsenopyrite concentrate from the flotation circuit. The arsenic is oxidized to form volatile arsenic trioxide which is condensed from the roaster gas stream and recovered in a conventional baghouse dust collector. The baghouse dust is then pneumatically conveyed into underground storage chambers.

The underground storage of arsenic trioxide dust was initiated in 1951, with the arsenic placed in specially designed chambers excavated in waste rock (massive volcanic units). The first series of chambers were located within several hundred feet and to the north of the baghouse, and accessed by raise to surface. From 1962 through 1976 the dust was placed in abandoned production stopes (B208, B212, B213, B214, and C212). Since 1976 the dust has been placed in specially designed chambers excavated in waste rock, in various locations.

Currently, there are a total of fifteen storage chambers in which arsenic trioxide bearing dust has been stored. A new storage chamber, identified as #15 (or B15), is under development.

4.0 Storage and Handling of Arsenic Trioxide:

Production and Delivery Systems:

The existing roaster installed at the Giant mine consists of a two stage Dorrco Fluo-solids roaster. This roaster currently operates with two Cottrell hot precipitators, one active, the other on standby, which collect dust from the roaster exhaust gas. After passing through the hot Cottrells, the roaster gases are air cooled to 224° F for arsenic fume condensation before entering the baghouse. Filtered gases from the baghouse continue on through a booster fan to a 45.7 meter (150 foot) brick discharge stack. The dust from the hot Cottrells is processed for gold recovery and the arsenic collected in the baghouse is pneumatically conveyed to underground storage.

Transportation of the arsenic dust produced by the roaster operations, is by pneumatic stowage-conveyance of the material passing through a standard 100 millimetre (4 in.) diameter steel pipe from the baghouse building to the particular arsenic chamber being filled. The pipes are run underground in drifts used for this purpose only. These distribution drifts are remote from the active production mine workings.

In the more recent designs, several delivery pipes enter the chamber through an engineered concrete bulkhead, in the upper access to the chamber. The delivery lines are of varying lengths inside the chamber. The longest line into a chamber is utilized first, filling from the back of the chamber toward the front. As the chamber fills, the lines are switched to each shorter line.

The air used to transport the dust into the chamber is returned by a parallel 150 millimetre (6 in.) diameter pipe and is vented back into the baghouse inlet flue. The system is therefore a closed system. Dust loss does not occur during transportation, as the only place for the dust to settle out of the transportation air bed is inside the storage chamber being filled. Normal operating pressure, during filling of the chambers is .14 to .28 bar (2 to 4 psi). Maximum pump pressure is 1.1 bar (15 psi).

Chamber Locations and Description:

Currently there are 15 underground storage chambers, including the active B14 storage chamber (see Figure 1 and Figure 2). The storage areas contain approximately 235,000 tonnes (260,000 tons) of dust containing approximately 182,000 tonnes (200,000 tons) of arsenic trioxide (to Dec. 1997). The underground storage of arsenic was initiated in October 1951 and continues to the present day.

The more recent arsenic storage chambers are specially designed chambers, rectangular in shape, however five of the 15 chambers are mined out production stopes, of irregular shape.

Each of the storage areas is isolated from the mine workings by a reinforced concrete bulkhead. All bulkheads have been designed to withstand hydrostatic pressures.

Design Criteria:

Data indicates that the concept of storing arsenic in underground chambers was proposed for the first time in 1950 by Senior Mine Personnel. The strategy was considered viable at that time based on evidence of extensive occurrence of permafrost in the bedrock, during mining, and confirmation by exploratory drilling and temperature measurements. The first storage chamber (designated B230) was constructed in 1951, and was filled in about one year.

There is evidence that cold air was circulated into empty chambers and also the upper part of filled chambers to reinstate permafrost that may have receded during mining or filling operations.

By 1960, the Mine concluded that any underground opening would be suitable for arsenic trioxide storage, "*provided not so much that the area is in permafrost, but that the area is free from water flow or seepage*". In 1962, for the first time, a former production stope was converted to use for arsenic storage. This practice continued until 1976, when C212 was filled. C212 was the last of the candidate 'production stopes' that fulfilled the design criteria, including being above the perceived lower elevation of permafrost.

In 1976 the mine began again, the practice of constructing special chambers for arsenic trioxide storage.

The design of the storage chambers has considered the following general criteria since the practice began:

- the chambers were to be located in areas where permafrost was thought to exist or could exist on mine closure.

- the area was free from water flows and seepage.

- the openings, including exploration drill holes, were to be sealed to prevent the escape of arsenic trioxide dust.

- the storage areas were to be excavated in competent ground and the storage area was to be dry before arsenic storage commenced

5.0 Ground Stability and Grouting:

Ground Stability:

Golder Associates were retained to provide an assessment of the long term stability of the Chamber and to provide recommendations regarding ground support and grouting. The final report, titled "Geotechnical Assessment of the # 15 Chamber" is attached in Appendix I.

The assessment predicted that the Chamber would be stable, and provided recommended ground support based on the long tern requirements for the stope. The ground support included 2.4m long grouted rebar and 5m long cable bolts in the upper sill. Cable bolts and rebar are currently being installed as per Golder's recommendations. This work is being performed by Procon Mining and Tunneling, under the direction of Mr. Denis Gratton P.Eng., Chief Engineer for Giant Mine.

Grouting:

There is potential for groundwater flow to come into contact with stored Arsenic Trioxide dust, based on the evidence of small seeps in the lower sill, and hematitic staining on some joint surfaces. Due to this potential it was decided that an extensive grouting program should be undertaken.

Golder Associates was retained to make recommendations regarding the grouting program. These recommendations are included in the report in Appendix I. The grouting is expected to begin the week of July 27th or earlier, depending on the completion of the ground support program, which precedes any other work for safety reasons. Procon Mining and Tunneling will undertake this work. Grout hole drilling layouts have been provided by Ms. Sharon Cunningham, Senior Geologist, working in conjunction with Mr. Gratton. Grouting will be performed in areas of both the upper and lower sill, where joints and faults are noticeable, and where hematitic staining is present. The lower sill will be grouted where water is seeping from the floor and walls. All grouting operations will be continuous.

There are no exploration diamond drill holes in the vicinity of the Chamber, drilled from surface or underground.

In anticipation of seepage from the area surrounding the bulkheads, grouting of these areas will be performed immediately following the blasting of the bulkhead hitches. Grouting of these areas follows the recommendation of Golder Associates and will be performed by Procon Mining and Tunneling.

The initial grouting layouts are included in Appendix IV for information. Additional holes may be added later depending on the success of the first attempts.

6.0 Excavation Design:

The Chamber is designed with a rectangular shape, oriented to provide maximum available volume given the topography and existing mine development. Figure 5 illustrates the location and orientation of the Chamber with respect to the existing Chambers 11, 12, and 14. The #15 Chamber is nominally 200' long x 45' wide x 110' high. If the full capacity is utilized, the Chamber will provide storage for more than six years of dust production at an average of 11 tons per day.

Initially the chambers are silled on the lower and upper elevations to full size. A drop raise is blasted, at the extreme end of the chamber, to connect the two sills. Blasthole drilling, of 2" holes, is carried out from the top sill down, unless accuracy requires both up and down drilling. Blasthole muck is removed by remote LHD, to trucks that haul to surface. The bulkheads are installed after all blasthole muck is removed.

In #15 Chamber it is considered possible to drill accurate 90' downholes. This option has been tested through the successful drilling of the drop raise and 2 typical blasthole rings. A report from ICI Explosives Canada is included in Appendix III. At the time of writing, the accuracy of the 90' downholes had not been tested. The mine will continue to drill the 90' downholes as long as accuracy remains high. The recommended spacings are being adhered to.

7.0 Bulkhead Design:

The bulkheads for the upper and lower sills follow a similar design as those for the #14 Chamber due to the similar height and width, and elevation from surface (hydrostatic head). Minor changes have been made to accommodate the actual opening size.

The design and load calculations have been prepared by Ferguson, Simek, and Clark Consulting Engineers. These notes, and the bulkhead drawing are included in Appendix II.

As noted previously, the bulkhead areas will be grouted following the excavation of the hitches. The grouting pattern was recommended by Golder Associates, and is attached in Appendix IV.

8.0 Contingency Planning:

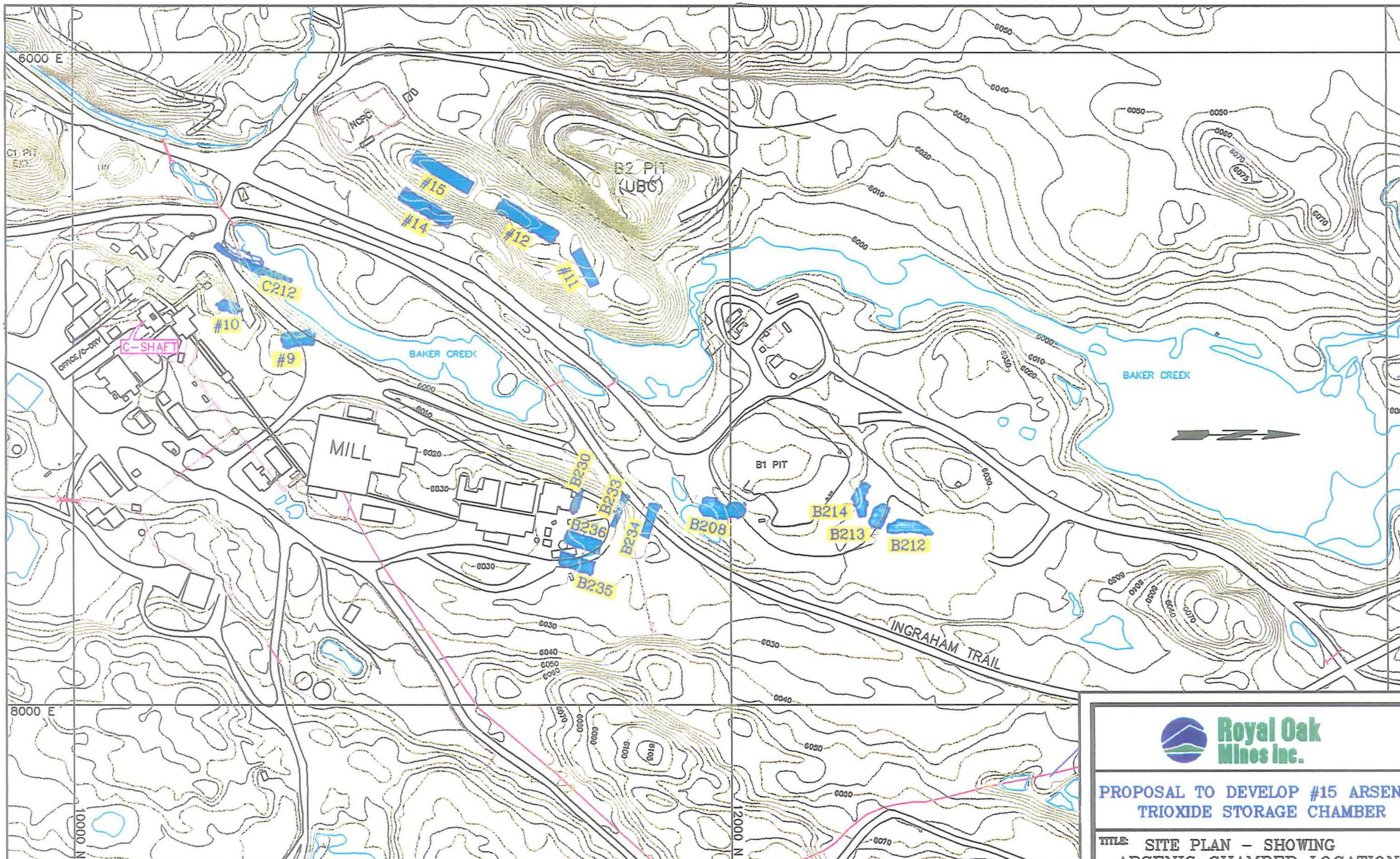
The Chamber will be pressure tested with mine air prior to commissioning in order to determine if there are any obvious leaks in the system. The Chamber will be pressurized to 25 psi, and monitored for leakage over a period of about 24 hours.

An extensive grouting program will be carried out in order to reduce or eliminate the potential for groundwater to contact the stored arsenic trioxide.

In the event that there is a release or seep of contaminated water from the chamber through the lower bulkhead area, the water will be collected in a sump located at the low point nearby the lower bulkheads of the #14 and #15 Chambers. This water will be pumped to a sump near the UBC portal where it is further pumped to the B-Shaft. At that point the contaminated water is diluted as it mixes with other mine drainage. The water is then pumped to surface, to the mill water treatment system, where it is treated, then reused or pumped to the active tailings containment area. The two sumps near the storage chambers will be cleaned of settled material prior to the commissioning of the #15 Chamber, to maximize their effectiveness.

The fill and return lines will be pressure tested with mine air, to ensure integrity, prior to use. Once in use the system is monitored through pressure checks at the Mill Baghouse. If there is a sudden pressure loss or high pressure buildup, the system is shut down and inspected. Any spilled material is recovered manually, and transferred to drums, or is recovered using the vacuum trucks, and transferred to an active or inactive chamber as capacity permits.

Delivery lines are inspected weekly by senior Mine Supervisors. Baghouse system checks are completed each shift by Mill Baghouse operators.



Royal Oak Mines Inc.

PROPOSAL TO DEVELOP #15 ARSENIC TRIOXIDE STORAGE CHAMBER

TITLE: SITE PLAN – SHOWING ARSENIC CHAMBER LOCATIONS

LEVEL: SURFACE		SCALE: 1"= NTS
DRAWN: MSL	CHECK:	DATE: JULY 1998

Figure 1

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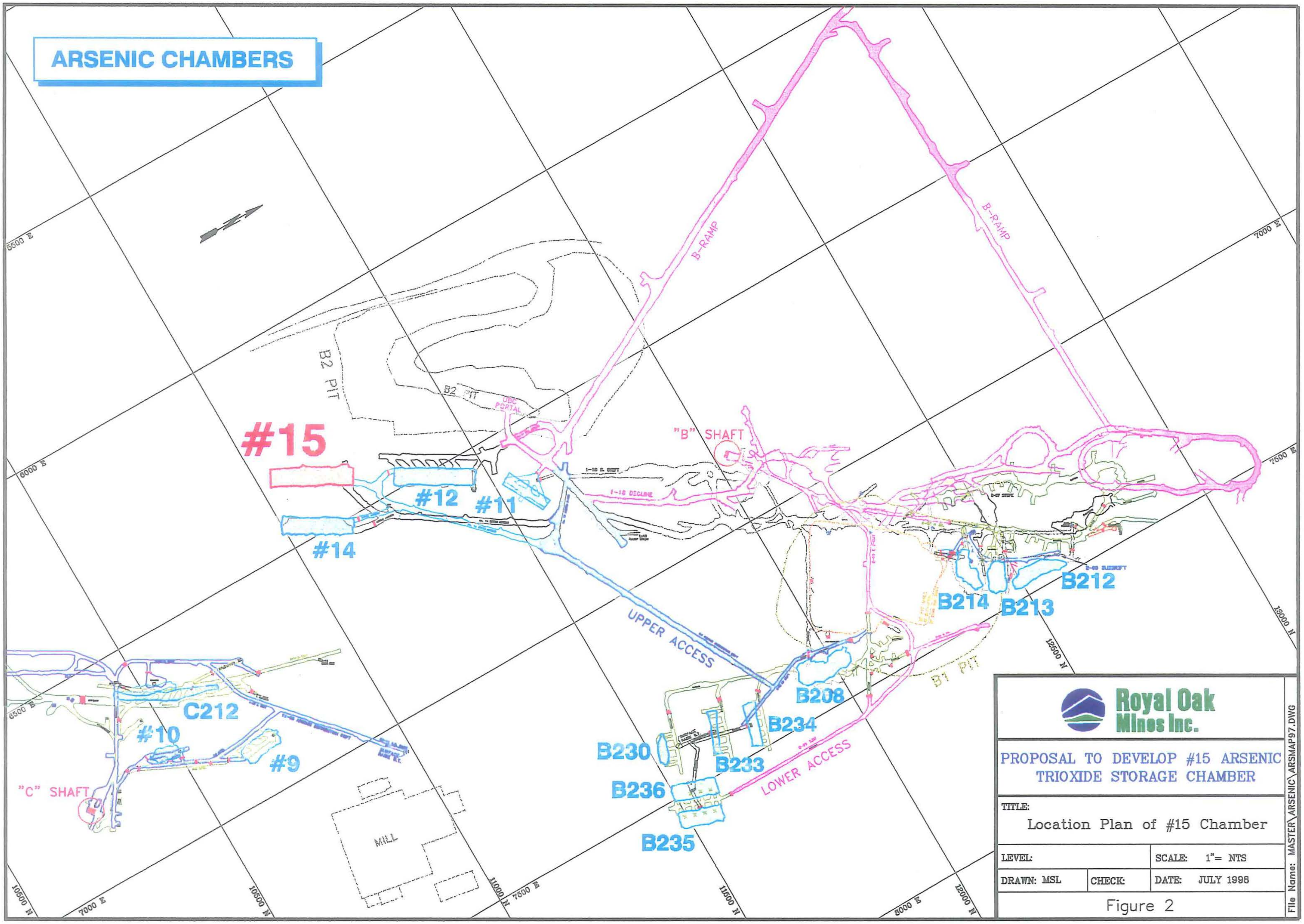
ARSENIC CHAMBERS
#15
#12
#11
#14
#10
#9
C212
B2 PIT
B2 PIT
UBC PORTAL
"B" SHAFT
"C" SHAFT
B-RAMP
B-RAMP
UPPER ACCESS
LOWER ACCESS
B1 PIT
MILL
B212
B214
B213
B208
B234
B230
B233
B236
B235
6500 E
6000 E
7000 E
7500 E
8000 E
7600 E
6500 E
7000 E
10500 N
10500 N
11000 N
11500 N
12000 N
12500 N
13000 N
Royal Oak Mines Inc.
PROPOSAL TO DEVELOP #15 ARSENIC TRIOXIDE STORAGE CHAMBER
TITLE:
Location Plan of #15 Chamber
LEVEL:
SCALE: 1"= NTS
DRAWN: MSL
CHECK:
DATE: JULY 1998
Figure 2
File Name: MASTER\ARSENIC\ARSMAP97.DWG

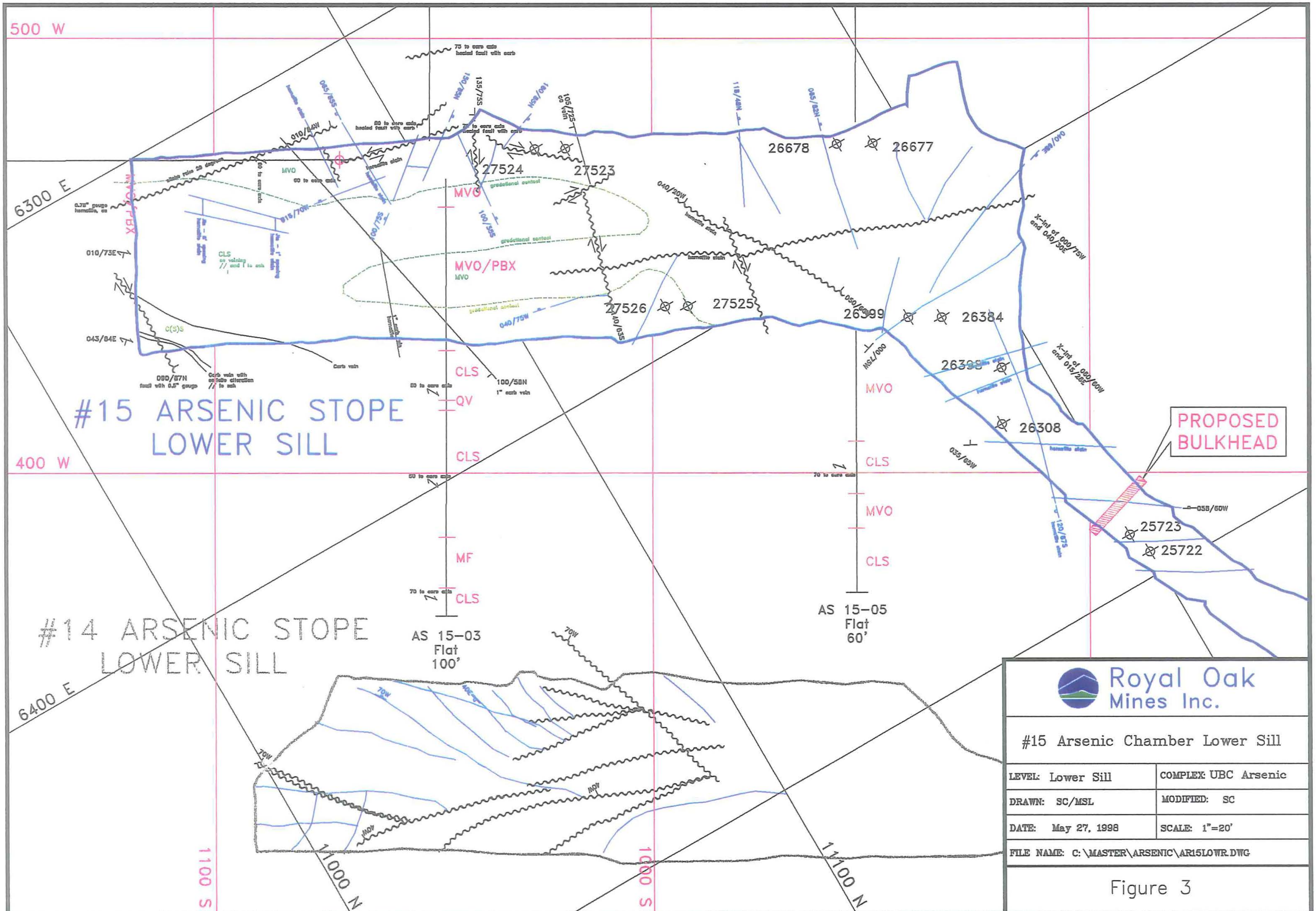
500 W
400 W
6300 E
6400 E
1100 S
1000 S
1000 N
1100 N
#15 ARSENIC STOPE
LOWER SILL
#14 ARSENIC STOPE
LOWER SILL
MVO/PBX
MVO
CLS
QV
MF
AS 15–03
Flat
100'
AS 15–05
Flat
60'
PROPOSED
BULKHEAD
26678
26677
27524
27523
27526
27525
26399
26384
26398
26308
25723
25722
Royal Oak
Mines Inc.
#15 Arsenic Chamber Lower Sill
LEVEL: Lower Sill
COMPLEX: UBC Arsenic
DRAWN: SC/MSL
MODIFIED: SC
DATE: May 27, 1998
SCALE: 1"=20'
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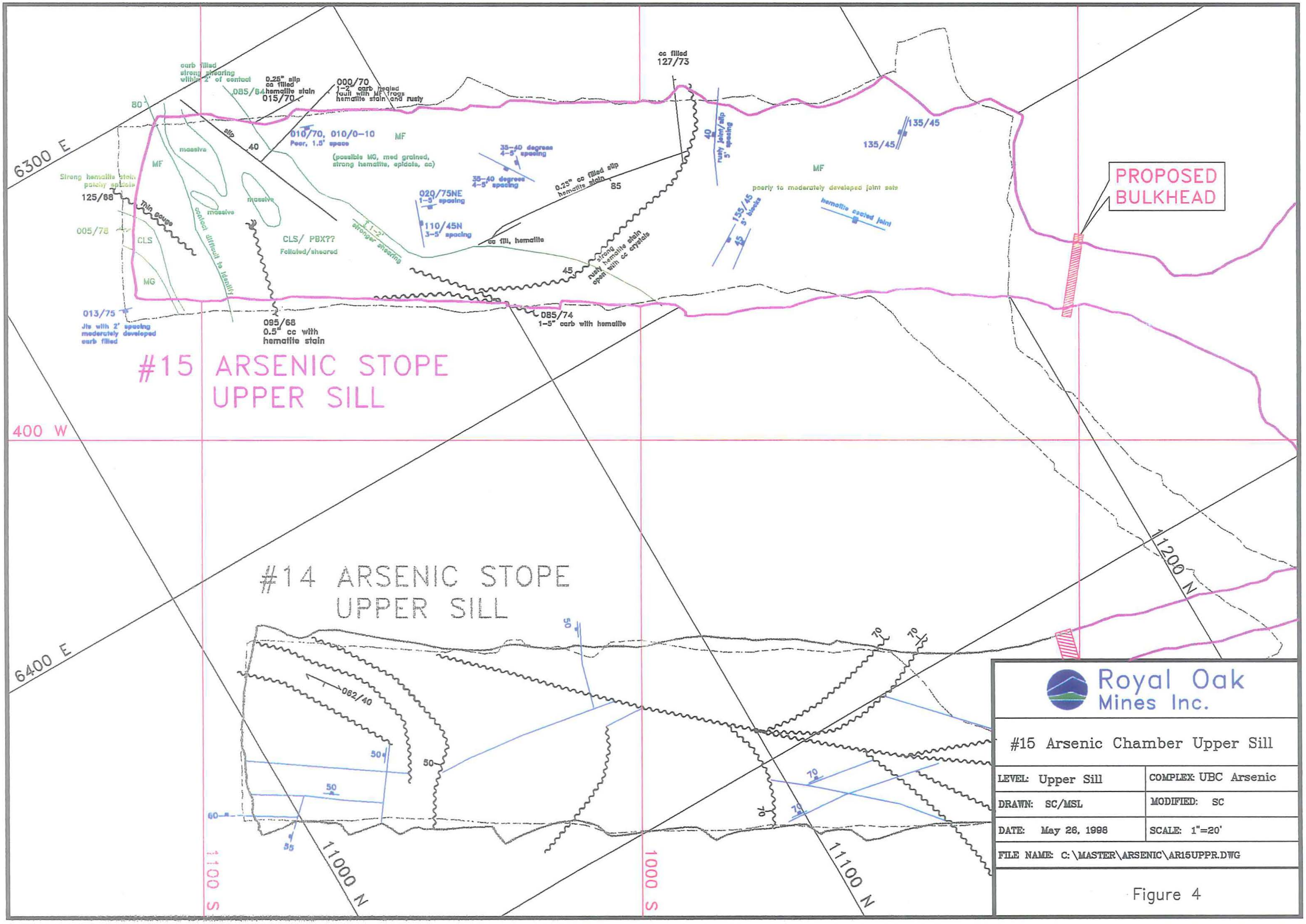
Figure 3



#15 ARSENIC STOPE
UPPER SILL
#14 ARSENIC STOPE
UPPER SILL
PROPOSED
BULKHEAD
6300 E
6400 E
400 W
1100 S
1000 S
11000 N
11100 N
11200 N
carb filled
strong shearing
within 2' of contact
085/84
0.25" slip
cc filled
hematite stain
015/70
000/70
1-2" carb healed
fault with MF frags
hematite stain and rusty
cc filled
127/73
010/70, 010/0-10
Poor, 1.5' space
(possible MG, med grained,
strong hematite, epidote, cc)
MF
massive
Strong hematite stain
patchy epidote
125/68
Thin gouge
contact difficult to identify
005/78
CLS
MG
CLS/ PBX??
Foliated/sheared
1-2" stronger shearing
020/75NE
1-5' spacing
110/45N
3-5' spacing
35-40 degrees
4-5' spacing
cc fill, hematite
0.25" cc filled slip
hematite stain
strong rusty hematite stain
open with cc crystals
rusty joint/slip
5' spacing
135/45
poorly to moderately developed joint sets
hematite coated joint
155/45
5' blocks
013/75
Jts with 2' spacing
moderately developed
carb filled
085/68
0.5" cc with
hematite stain
085/74
1-5" carb with hematite
062/40
Royal Oak
Mines Inc.
#15 Arsenic Chamber Upper Sill
LEVEL: Upper Sill
COMPLEX: UBC Arsenic
DRAWN: SC/MSL
MODIFIED: SC
DATE: May 26, 1998
SCALE: 1"=20'
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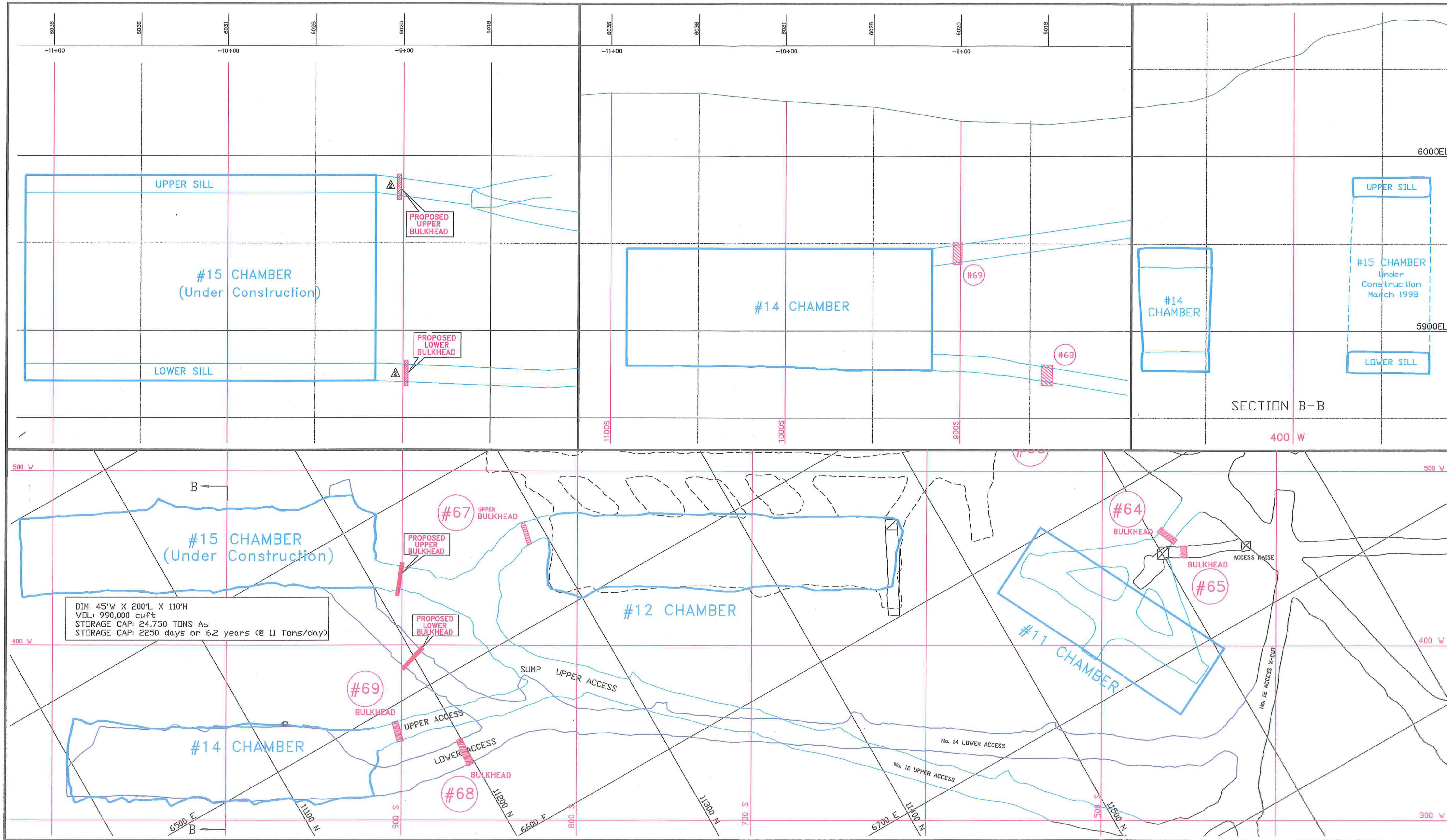
Figure 4



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REFERENCE DRAWINGS	1	INTERIM REPORT TO NWT WATER BOARD	MSL	MAR'98	R.A.	MAR'98
	2	PRE-CONSTRUCTION UPDATE	MSL	JULY'98	D.G.	JULY'98
* #15 CHAMBER UNDER CONSTRUCTION (MARCH 1998)	3					
	4					
	5					

SCALE: 1" = 50'	
DRAWN: B.M.	DATE: 23/05/96
CHECK:	DATE:
VENTILATION:	
ENGINEERING:	MINE CAPT.:
GEOLOGY:	MINE SUPT.:

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Figure 5

TITLE: ARSENIC STORAGE CHAMBERS #14 and #15

LEVEL: | COMPLEX:

DWG NO : 7150-012

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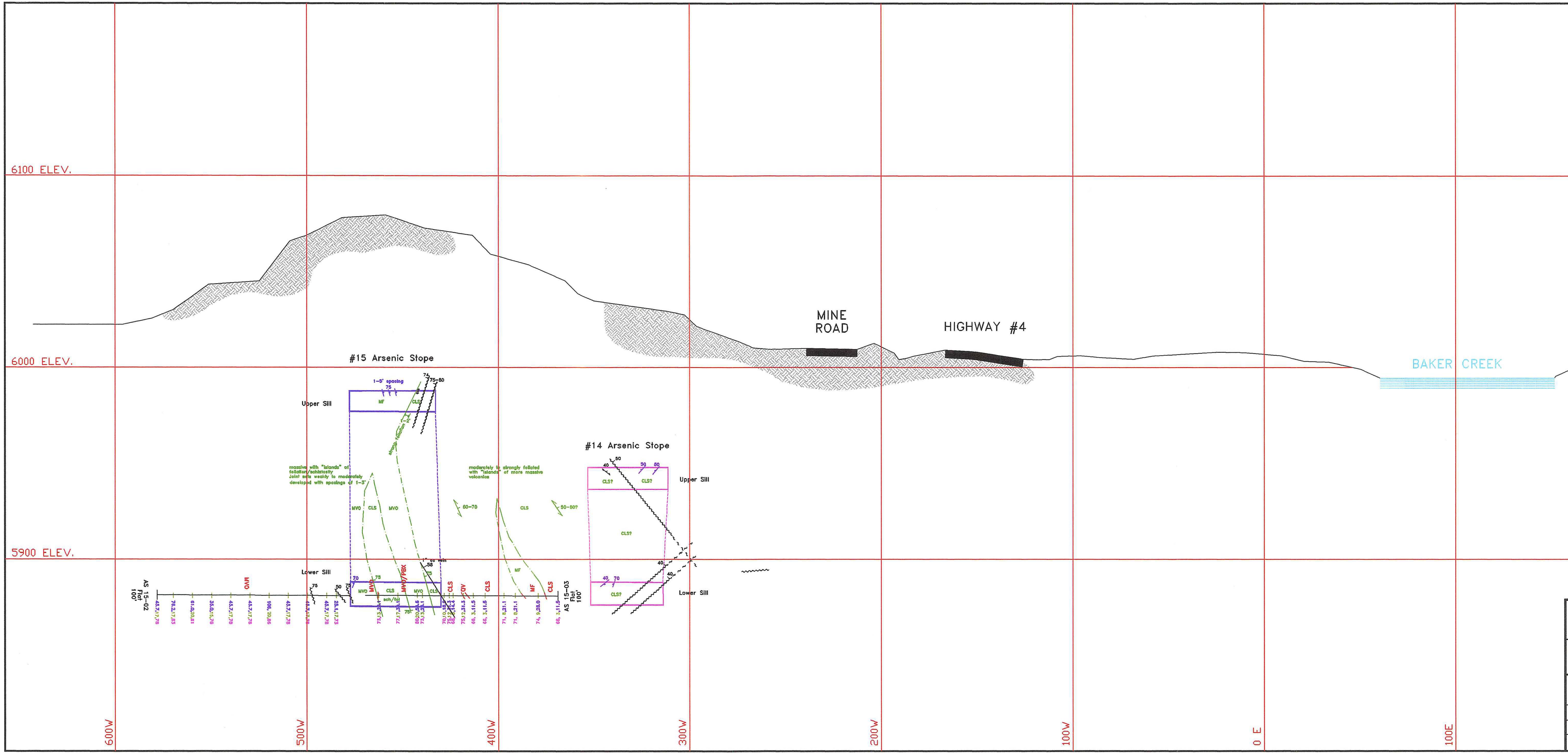
COPY 1



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PROPOSAL TO DEVELOP #15 ARSENIC TRIOXIDE STORAGE CHAMBER

TITLE: #15 Arsenic Chamber Geology Cross Section 1050 South – Looking North	
LEVEL:	SCALE: 1"= 40'
DRAWN: SC/MSL CHECK:	DATE: JULY 1998

File Name: MASTER\ARSENIC\ARS15X.DWG

Figure 6

Golder Associates Ltd.

500 - 4260 Still Creek Drive
Burnaby, British Columbia, Canada V5C 6C6
Telephone (604) 298-6623
Fax (604) 298-5253

Golder Associates

REPORT ON

GEOTECHNICAL ASSESSMENT OF THE #15 CHAMBER GIANT MINE

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DISTRIBUTION:

3 Copies - Royal Oak Mines Inc., Yellowknife, NWT
2 Copies - Golder Associates Ltd., Burnaby, BC

July 1998

982-1418

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Golder Associates

July 20, 1998

982-1418

Royal Oak Mines Inc.
NWT Division
P.O. Box 3000
Yellowknife, NWT
X1A 2M2

Attention: Mr. D. Gratton
Chief Engineer

RE: GEOTECHNICAL ASSESSMENT - #15 ARSENIC CHAMBER

Dear Mr. Gratton:

Golder Associates is pleased to provide this report on our geotechnical assessment of the #15 Arsenic Chamber at the Giant Mine. If you have any questions regarding this report, please contact us.

Yours very truly,

GOLDER ASSOCIATES LTD.

W. Forsyth

W.W. Forsyth, P. Eng.
Associate, Mining Group

for A.E. Moss, Eur. Ing.
Principal, Mining Group

WWF/AEM/lc
982-1418

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1.0 INTRODUCTION

This report provides a geotechnical review of the proposed #15 Arsenic Chamber at Royal Oak Mines' Giant Mine. The work was commissioned by Mr. R. Allan, P.Eng., Manager of Mining Projects.

2.0 SCOPE OF WORK

The tasks outlined for completion as part of this project work were as follows:

- Perform a stability analysis of the #15 Chamber.
- Provide ground support recommendations for the upper sill of the #15 Chamber.
- Provide recommendation for a grouting program for the #15 Chamber.

Preliminary recommendations were provided to Royal Oak in March of 1998 in the form of two technical memorandum.

3.0 INFORMATION PROVIDED

The following information was provided by Royal Oak:

- Internal report entitled "Structural Features of the #15 Arsenic Stope – Giant Mine", prepared by Jody Todd, April 24th, 1996.
- Arsenic Stope – Structural Mapping, plans of upper and lower sills, traverse mapping, May 6th, 1998.
- Photographs of the upper and lower sills, May 6th, 1998.

General knowledge of mine and previous arsenic stopes was available from historical Golder Associates' site visit reports to the Giant Mine.

4.0 ROCK MASS DESCRIPTION

The rock mass in and around the #15 Chamber was characterized by assessing the following:

- Lithological setting
- Structural features
- Rock quality

4.1 Geology

The #15 Chamber is located in the Townsite Formation of the Archean Kam Group. The formation consists of rhyodacite breccias, tuffaceous felsic porphyrys and gabbro sills. Mapping by the mine staff identified the primary rock type as a massive chloritized greenstone, varying from dacitic to basaltic in appearance with minor breccia.

The observed alteration was primarily chloritization with minor sericitic-chloritic horizons. Local areas of hematitic staining was observed on faults surfaces.

4.2 Structural Geology

Structural features in the upper and lower sill of the #15 Chamber were mapped by the mine staff. Subsequent stereographic analysis (Figure 1) identified the primary fault orientations as the following (dip/dip direction):

F1 – 69/162
F2 – 58/310
F3 – 69/046

The orientations of these faults were used to assess the potential for wedge formation in the back and walls of the #15 Chamber and to plan grouting operations.

4.3 Rock Quality

Design of underground excavations benefit from a number of well-established empirical and semi-empirical rules. These enable estimates to be made of the expected mining conditions and support requirements on the basis of a detailed description of the rock mass. The complete design procedure involves two steps. First, the quality of the rock mass is rated on the basis of a pre-defined classification system and second, the expected performance of the underground opening is predicted using empirically derived correlations with the rock quality factor.

The assessment of rock quality provides a method of summarizing a number of important geotechnical parameters into a single number. This number is a measure of how good, or bad, the rock is in an engineering context.

Rock mass quality was assessed using Barton's 'Q' System. This system is described in detail in Barton (1974). The Q system has the form:

$$Q = \left(\frac{RQD}{J_n}\right) * \left(\frac{J_r}{J_a}\right) * \left(\frac{J_w}{SRF}\right)$$

where:

RQD	=	the Rock Quality Designation
Jn	=	the joint set number
Jr	=	the joint roughness number
Ja	=	the joint alteration number
Jw	=	the joint water reduction factor
SRF	=	the stress reduction factor

The physical significance of the system is that the term RQD/Jn, represents, in principle, the average block size of the mass, Jr/Ja, the average total shearing strength of the joints, Jw, both the relative potential of the water eroding the joint infilling and the relative potential head on the support and SRF, both the structurally controlled kinematic possibility of dislodgment of blocks of ground and the concentrated stress related confinement, fracturing, squeezing and swelling of the excavation walls.

The parameters RQD, Jn, Jr and Ja are usually measured during geotechnical core logging. Jw is estimated from previous experience and drilling reports of water levels. SRF is determined by empirical methods relating the estimated in-situ stress and rock strength.

4.3.1 #15 Chamber Rock Quality

Rock quality measurements were made on core recovered from five diamond drill holes drilled in the walls of the proposed chamber. Detailed logs for these holes are on file at the mine site. The quality of the immediate wall and back rock masses ranges from Q=5.9 (fair rock quality) to Q= 25 (good rock quality).

5.0 STABILITY ANALYSIS

The stability of the #15 Chamber was assessed using the Mathews' Method empirical design approach, a wedge analysis and a two dimensional stress analysis.

5.1 Mathews' Method

The Mathews' Method, discussed in Golder (1981) and Stewart (1995), has been used to assess the stability of the walls and back formed during primary and secondary mining. This empirical method was specifically developed for open stoping applications. It uses a relationship between a modified Q value and the stability of a two-way excavation span. In this approach, a modified Q value, known as the stability number (N), accounts for the rock mass quality, the state of stress and the orientation of the exposed surfaces. An excavation factor, equivalent to the hydraulic radius (area/perimeter), is used to represent the exposed surface. The specific combination of stability number and hydraulic radius is compared to a large database of information to assess its likely stability.

The stability number (N) is calculated using the following formula:

$$N = Q' * A * B * C$$

where:

Q' = modified Q rating

A = stress factor

This factor is a measure of the ratio of intact rock strength to induced stresses. As the maximum compressive stress acting on a free stope surface approaches the uniaxial compressive strength of the rock, factor A degrades to reflect the related instability due to rock yield. The range of values for this factor are between 0.1 and 1.

B = rock defect orientation factor

This factor is a measure of the relative orientation of dominant jointing with respect to the excavation surface. Joints which form a shallow, oblique angle

(10°-30°) with the free face are the most likely to become unstable (slip or separate). Joints which are perpendicular to the face are assumed to have the least influence on stability. The range of values for this factor are between 0.2 and 1.

C = design surface orientation factor

This factor is a measure of the influence of gravity on the stability of the stope surface being considered. Overhanging stope faces (backs) or structural weaknesses which are oriented unfavourably with respect to gravity sliding have a maximum detrimental influence on stability. The range of values for this factor is between 1 and 8.

5.1.1 #15 Chamber Stability Analysis

The proposed chamber has approximate dimensions of 13.7m wide, 61m long and 34m high. Figure 2 is a graphical presentation of the Stability Number, N, and the hydraulic radius of the chamber surfaces.

Table 1 Summary of Mathews' Method Assessment

Chamber Surface	Q'	A	B	C	N
Back	5.9 – 25.1	1	0.8	1	4.7 – 20.1
North Wall	5.9 – 25.1	1	0.4	8	18.9 – 80.3
South Wall	5.9 – 25.1	1	0.9	8	42.5 – 180.7
East Wall	5.9 – 25.1	1	0.8	8	37.8 – 160.6
West Wall	5.9 – 25.1	1	0.8	8	37.8 – 160.6

According to the Mathews' Method analysis:

- The vertical walls are predicted to be stable without the addition of artificial support.
- The back of the chamber will require artificial support to ensure long term stability.

5.2 Wedge Analysis

The back of the chamber was evaluated using a limit equilibrium analysis to assess the potential formation and stability of three dimensional wedges. The program Unwedge (Hoek, 1995) was used to carry out the analysis. Structural information presented earlier in this report provided input to the computer program.

Output from the wedge analysis program is given in Figure 3. The program indicates the following:

- Potentially unstable wedges could be formed in the back and walls of the chamber by the intersection of the recorded fault planes.
- The wedge analysis confirms the requirement for cable support of the upper sill.
- Unstable wedges may be formed in the walls of the chamber after excavation. Failure of these wedges, if any are present, would occur shortly after excavation and should not pose any long term stability problems.

It is important to note that the program determines the potential for wedge formation on average orientation and persistence of input structures, not on the actual structures. The detailed mapping undertaken by the mine staff does not show any potential wedges in the back of the upper sill.

5.3 Stress Analysis

The distribution of stress around the #15 Chamber was evaluated using a two dimensional boundary element analysis package, Examine 2-D (Hoek,1995). A section through the #15 and #14 Chambers was input into the program. Input parameters for the analysis were as follows:

$\gamma = 0.024$ MN/m	E = 20 GPa
$\sigma_h = 3 \times \sigma_v$	$\mu = 0.25$
Hoek Brown Failure Criterion m = 4.87 and s = 0.021	UCS = 80 MPa

Figure 4 is the predicted level of stress after the excavation of the chamber. The results of the stress analysis indicate the following:

- Horizontal stress in the back of the chamber increases to a level of 6 to 8 MPa from an in-situ level of approximately 2.5 MPa.
- The level of stress relative to the strength of the of intact rock is low and should not present any stability problems.
- The stress analysis does highlight the relatively low stress around the chamber. Structurally controlled, gravity driven failures tend to be more prevalent in low stress regimes. The potential wedges identified in the previous sections are examples of this type of failure.

5.4 Stability Summary

The stability of the chamber was assessed using two empirical design methods and a two dimensional stress analysis.

- Our initial assessment of the stability indicates that the vertical walls will be stable in the proposed configuration without any artificial support. Local wedge failures in the walls should be expected during excavation.
- The back will require systematic support to maintain long term stability.

6.0 GROUND SUPPORT

Support design can be divided into five steps. These are:

- characterization of the rock mass;
- prediction of induced stress over the life of the opening;
- prediction of likely ground behaviour;
- assessment of support requirements; and
- support selection.

The rock mass characterization, predicted stress levels and likely ground behaviours have been discussed in previous sections.

6.1 Support Requirements

Barton et al (1974) analyzed extensive data relating 'Q' to the behaviour and support requirements of an isolated underground excavation. The relationship that was developed makes use of a quantity called the equivalent dimension (D_e) of the excavation. This dimension is obtained by dividing the projected span by the excavation support ratio (ESR).

The excavation support ratio is related to the use for which the excavation is intended and the extent to which some degree of instability is acceptable. Based on recommendations of Barton et al (1974) and experience, an ESR of 1.6 was selected to reflect the long term stability requirements of the chamber.

The formation of stress arches above underground excavations occurs in most mines. The arches are the result of the stress redistribution about the opening. The location of the stable arch beyond the excavation is dependent upon the rock mass properties and excavation span. In order to maintain the stability of the natural arch, the de-stressed rock between the excavation boundary and natural arch boundary must be stabilized. Stabilization is best achieved by installing bolts that anchor above the stress arch boundary. The length of rock bolts is thus designed according to the span of the excavation. Normally, the bolts should be a minimum of one third of the span of the excavation. For example, in a drift which is less than 5.5m wide, 1.8m long bolts are required; drifts up to 7m wide require 2.3m, etc.

6.1.1 Cable Bolt Design

An empirical approach to design cable bolt support of underground openings is presented in Hutchinson (1996). This methodology relates the hydraulic radius of the exposed surface, the Stability Number and cable bolt performance at existing mining operations. A series of four graphs provide the following:

- Applicability of cable bolts to the proposed back/wall exposure;
- Recommended cable spacing for single strand cables;
- Recommended cable spacing for double strand cables; and

- Recommended minimum cable bolt lengths.

In addition to this approach, the wedge stability analysis provided the size of potentially unstable wedges in the chamber back.

6.2 Support Selection

The basic rock bolting pattern was determined for a 13.7m wide back. Figure 5 is a chart showing the support recommendations. Cable bolting requirements are shown in Figures 6, 7, and 8 which give the spacing for single cables, spacing for double cables and cable length respectively.

The proposed ground support for the upper sill of the chambers is as follows:

- 2.4m long, 19mm diameter cement or resin grouted rebar on a 1.5m diamond pattern
- 5m long, 15.8mm diameter cable bolts on a 2.5m diamond pattern (cables should be plated and tensioned to 2 tonnes). If double cables are used, the spacing could be expanded to 3m.

7.0 GROUTING

The proposed grouting is designed to address potential water inflow during operation of the mine. If the mine floods upon closure, water would enter the chamber.

7.1 Assessment Basis

We have based our assessment of the grouting requirements on the following information:

- Water inflow which is occurring in the lower sill at low, but observable rates (no estimate of flow rates has been provided). No water has been noted in the upper sill drift.
- Water chemistry measurements which indicate that the water is fresh, surficial water and not related to the adjacent arsenic stope (#14).
- The location of hematite stained surfaces on joints and faults mapped by Royal Oak staff.

7.2 Grouting Based on Current Inflow

Given that the water entering the stope is not contaminated, it must be originating from surface. The most likely pathway is along the 50° to 75° east dipping structures that connect to steeper west dipping structures that intersect the sill of the stope.

The east dipping structures should be intersected by holes drilled in the west wall of the lower sill. A fan of three 9.1m (30 foot) long holes with dips between 15° and 40° downward to the east should be drilled. The fans should be on nominal 4.6m (15 foot) centers for primary grouting. Holes should be drilled between the fans to check grout take.

The west dipping structures should be intersected by holes drilled in the sill of the lower level. Two rows of 30 foot long, 45° east dipping holes should be used to grout these features. The fans should be drilled on 4.6m (15 foot) centers. Hole should be drilled between these rows to check grout take.

7.3 Grouting Based on Hematite Staining

Hematite staining on fault and joint surfaces may indicate the historical passage of water along the discontinuity. Grout holes should be planned to intersect these features at roughly right angles, approximately 6.1m (20 feet) from the opening (this implies approximately 9.1m (30 foot) long grout holes). These holes should be drilled at 1.8m (6 foot) centers along the structures.

7.4 Grouting Around the Lower Bulkhead

Grouting around the lower bulkhead could be accomplished using three fans of holes. These fans should each have six holes drilled at increments of 60° and to a depth of 4.6m (15 feet). The fans should have a spacing of 1.8m (6 feet).

7.5 Other Grouting Items

Grouting should take place after the recommended ground support has been installed.

Exploration diamond drill holes in the vicinity of the stope should be marked on the plans and sections and grouted where possible.

In addition to the existing geologic structures, blast induced fracturing along the walls of the stope could provide pathways for water flow. In order to minimize the level of blast induced fracturing, excavation of the chamber should include wall control blasting techniques.

8.0 SUMMARY

A geotechnical assessment of the #15 Arsenic Chamber was completed. This assessment provided the following results:

1. The proposed excavation is predicted to be stable. However, given the long term requirements for the chamber, systematic support of the back of the upper sill is recommended.

2. Ground support consisting of 2.4m long grouted rebar and 5m long cable bolts are recommended for the back of the upper sill.

3. Grouting will be required to limit groundwater flow into the chamber.

4. The proposed grouting will be effective during the operation of the mine (i.e., when the mine is not flooded).

5. Care and attention to controlled blasting during the excavation of the chamber and quality control during support installation and grouting are required.

9.0 REFERENCES

Barton, N., Lien, R. and Lunde, J., 1974. "Engineering Classification of Rock Masses for the Design of Tunnel Support." Rock Mechanics, V. 6, No. 4, pp. 189-236.

Golder Associates Ltd., 1981. "Prediction of Stable Excavation Spans for Mining at Depths Below 1,000 m in Hard Rock", Canmet Report 055 05Q80-00081.

Stewart, S.V.B. and Forsyth, W.W., 1995. "The Mathew's method for open stope design", CIM Bulletin Vol.88, No.992.

Hoek, E., Kaiser, P.K. and Bawden, W.F., 1995. "Support of Underground Excavations in Hard Rock".

Hutchinson, D.J. and Diederichs, M.S., 1996. "Cablebolting in Underground Mines".

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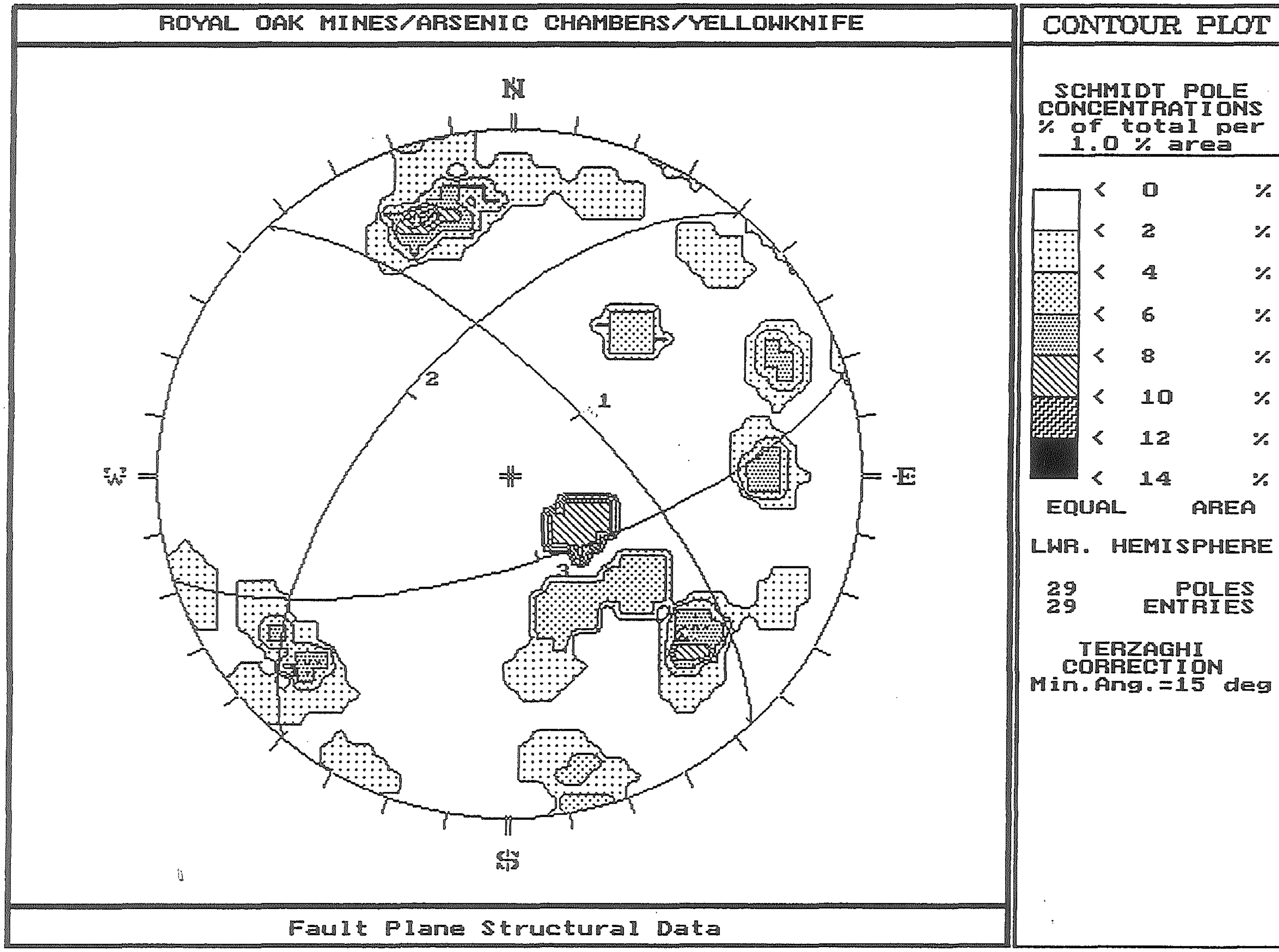
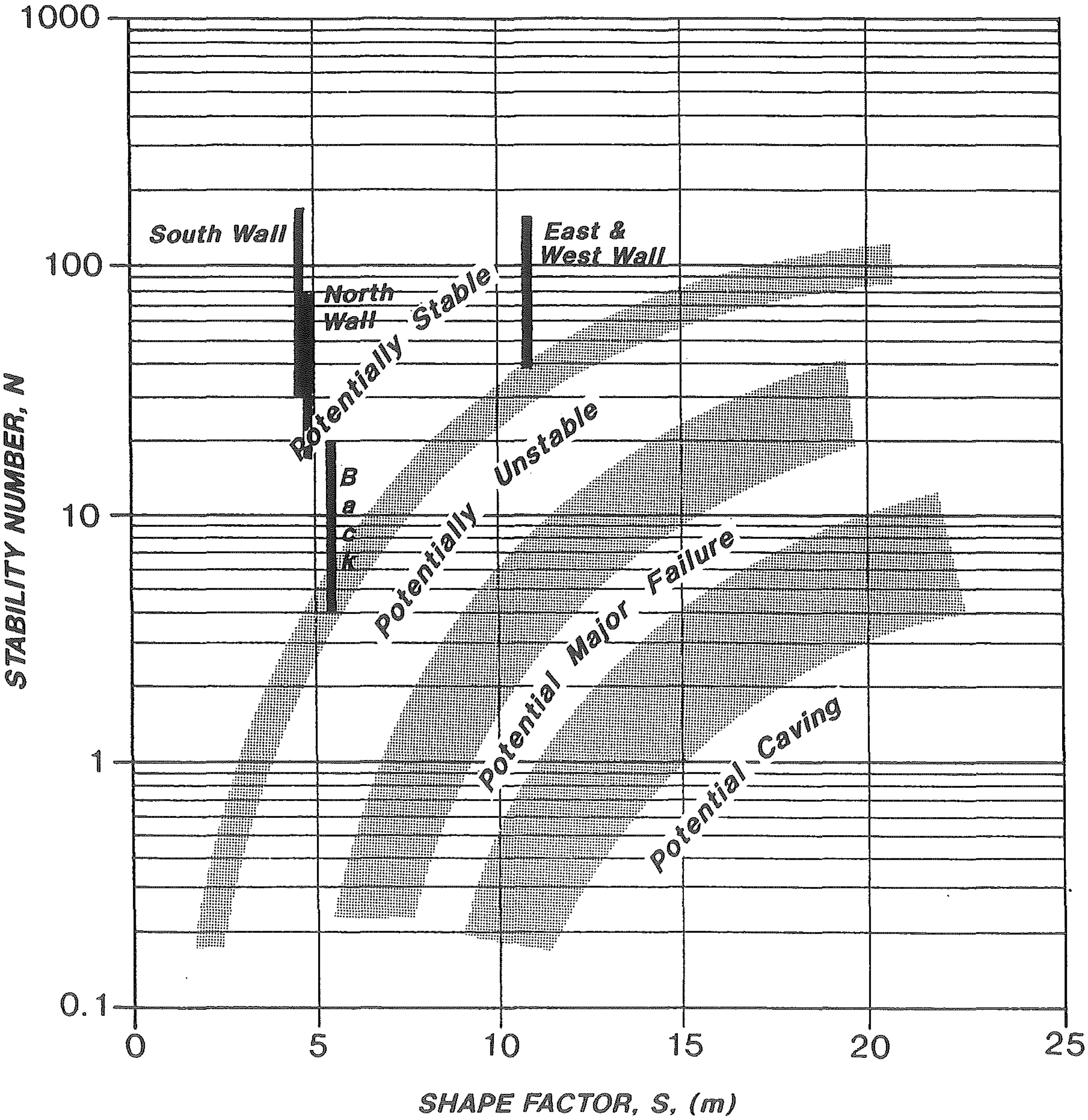


Project No. 982-1418 Drawn w.t. Reviewed W.W.F. Date July '98	Golder Associates	**STEREOGRAPHIC ANALYSIS OF FAULT ORIENTATIONS**	Figure 1



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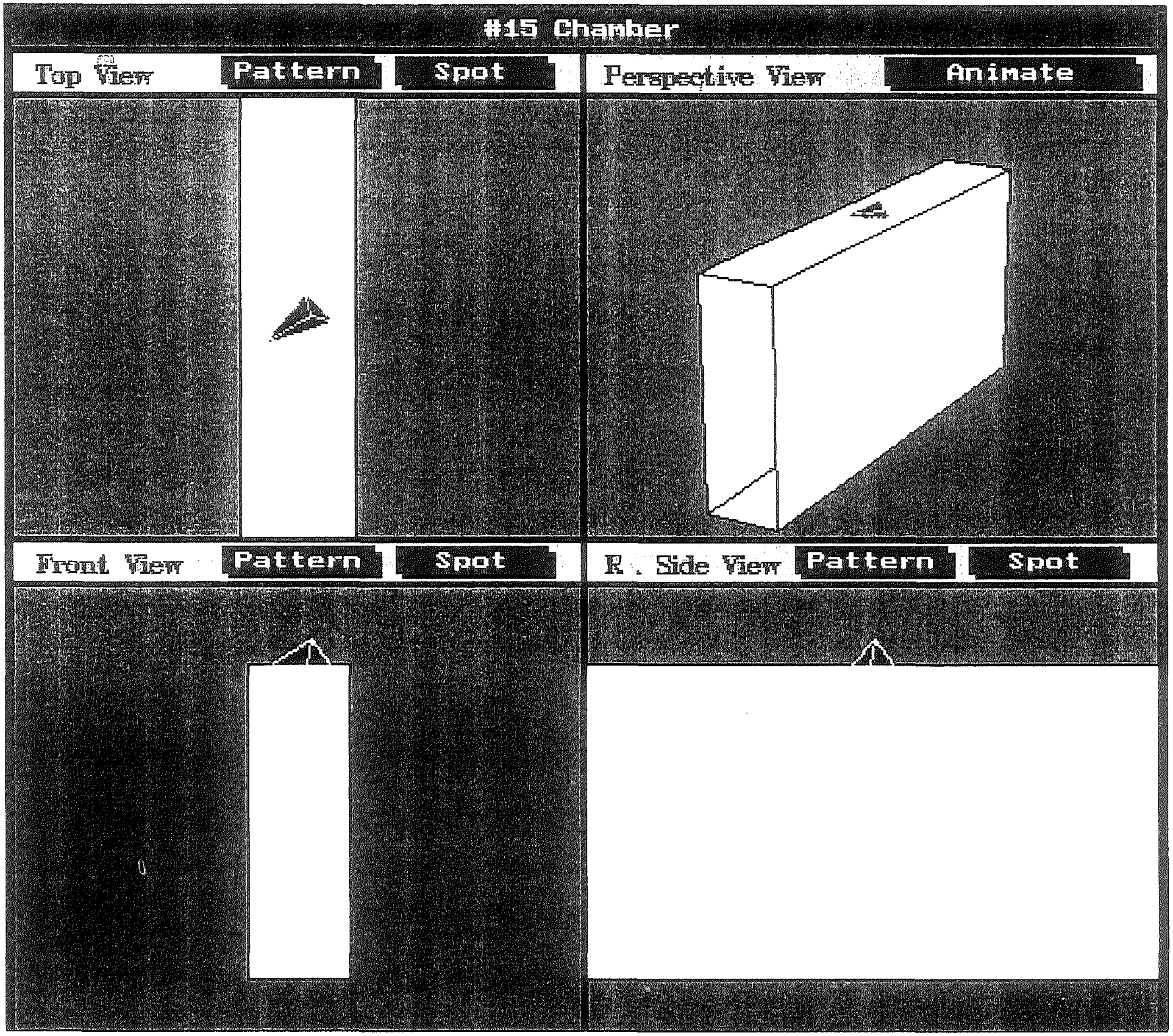
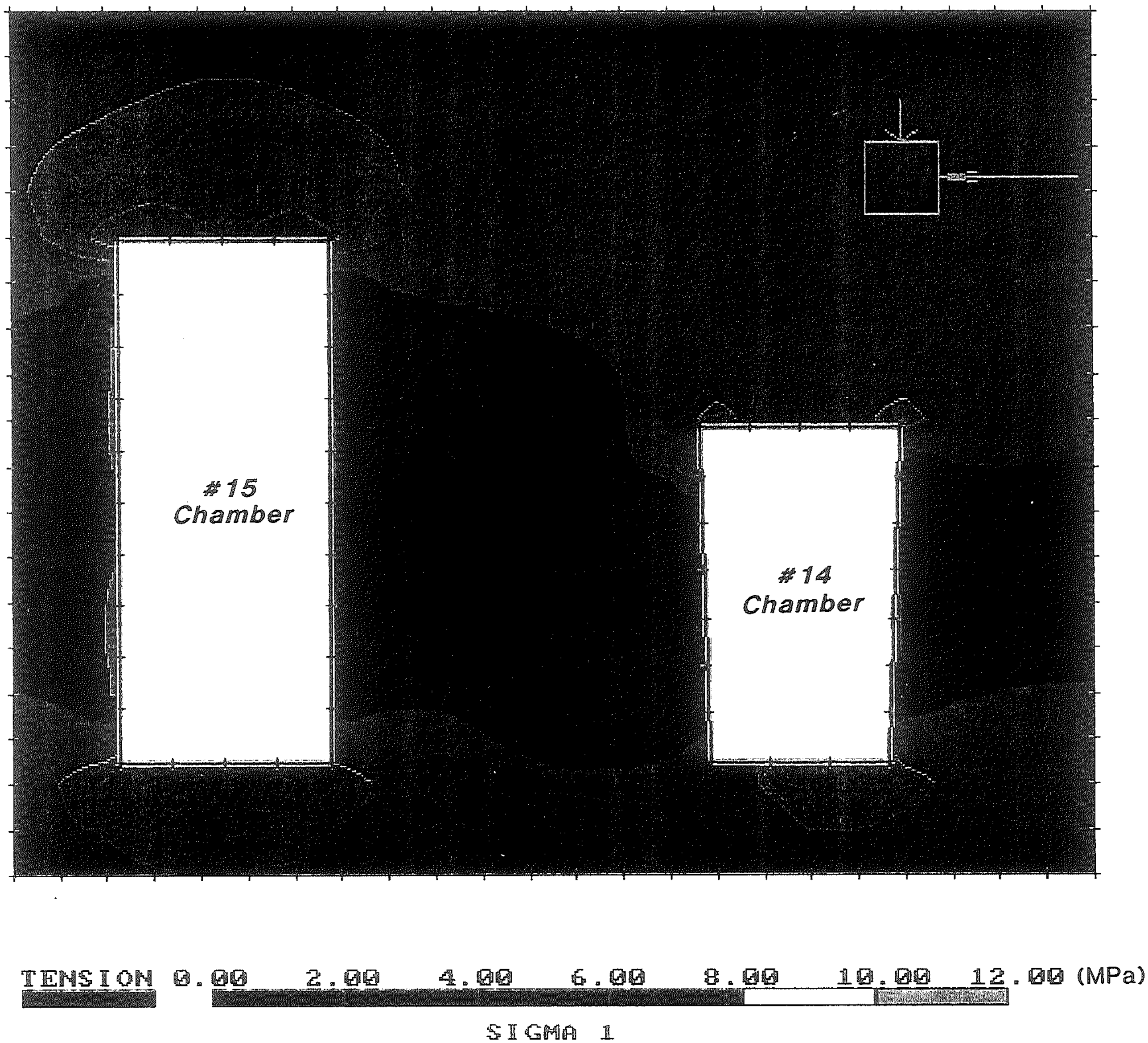
Golder Associates

STABILITY ANALYSIS
#15 ARSENIC CHAMBER

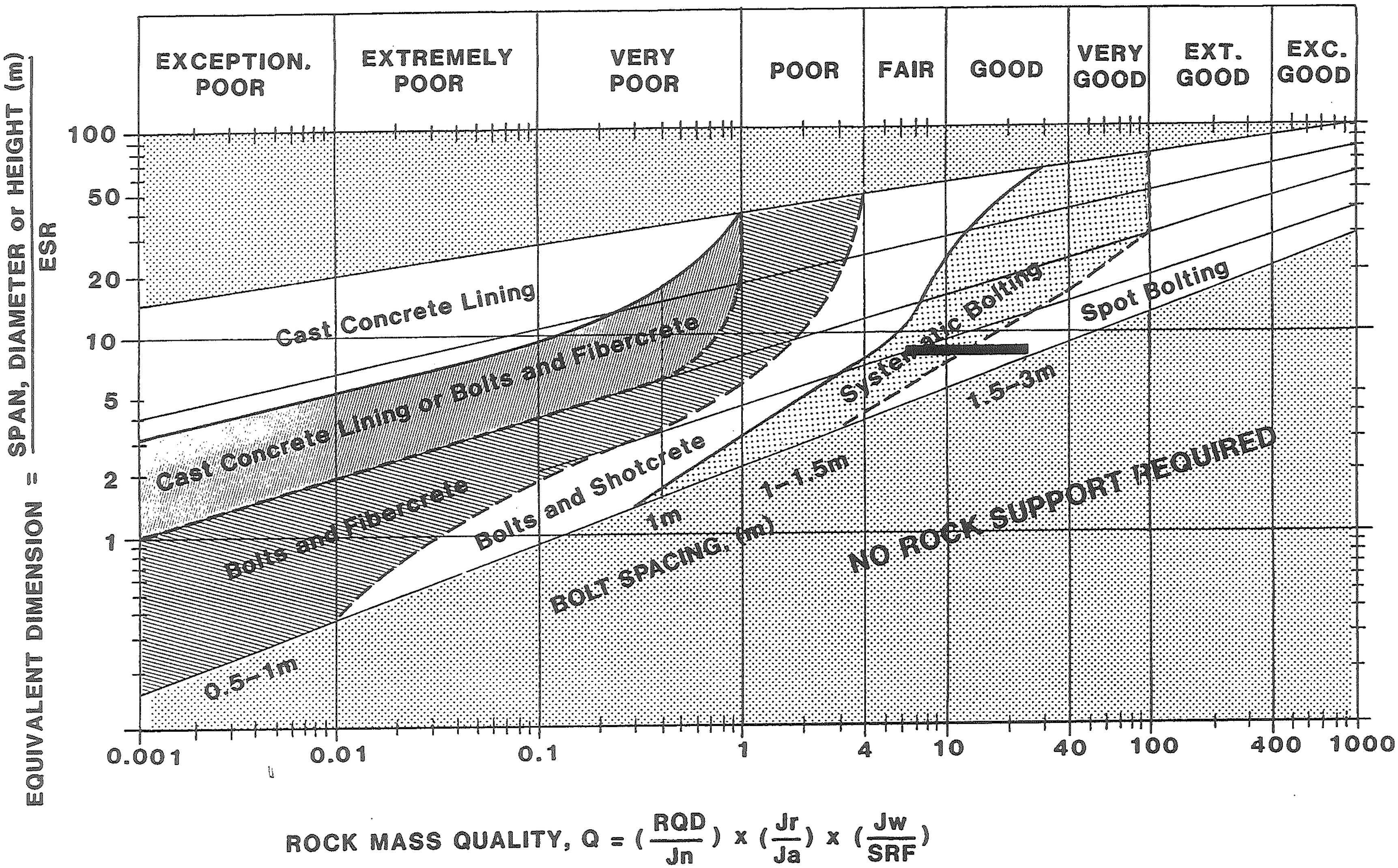
Figure 2



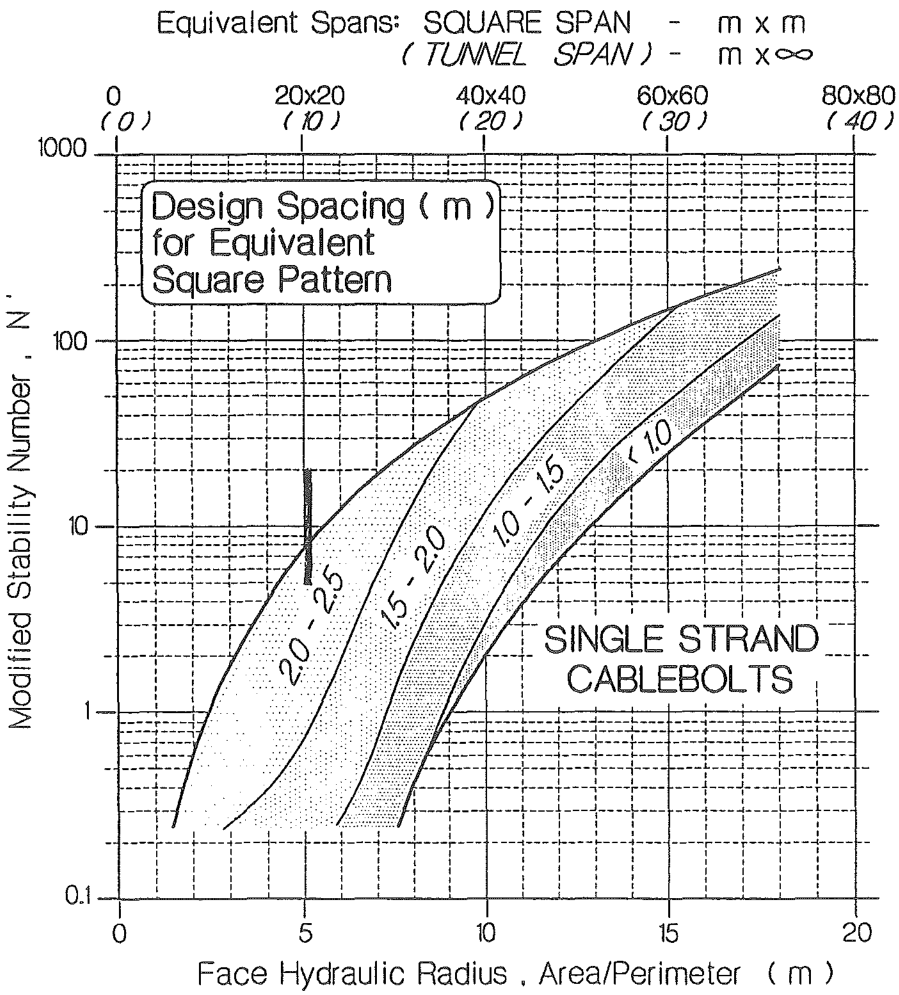
Project No. 982-1418 Drawn w.t. Reviewed W.W.F. Date July '98	Golder Associates	WEDGE ANALYSIS – #15 CHAMBER	Figure 3



Project No. 982-1418 Drawn w.t. Reviewed W.W.F. Date July '98	Golder Associates	**PREDICTED LEVEL OF STRESS AFTER EXCAVATION**	Figure **4**



Project No. 982-1418 Drawn w.t. Reviewed W.W.F. Date July '98	Golder Associates	**ESTIMATED BOLTING REQUIREMENTS**	Figure 5



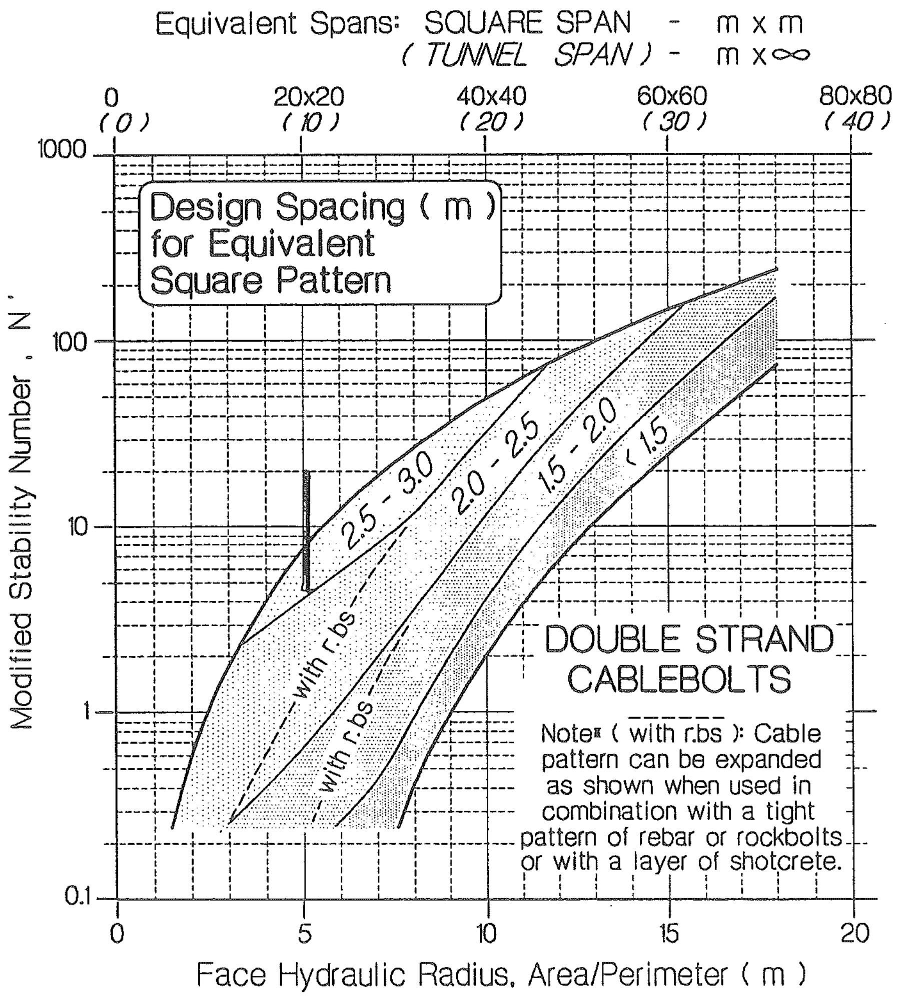
REFERENCE: Hutchinson, (1996)

Project No. ....418 .... Drawn .... Reviewed .... Date July ...

Golder Associates

RECOMMENDED SPACING FOR SINGLE STRAND CABLES (regular pattern)

Figure 6



REFERENCE: Hutchinson, (1996)

Project No. 982-1418 Drawn w.t. Reviewed W.W.F. Date July '98

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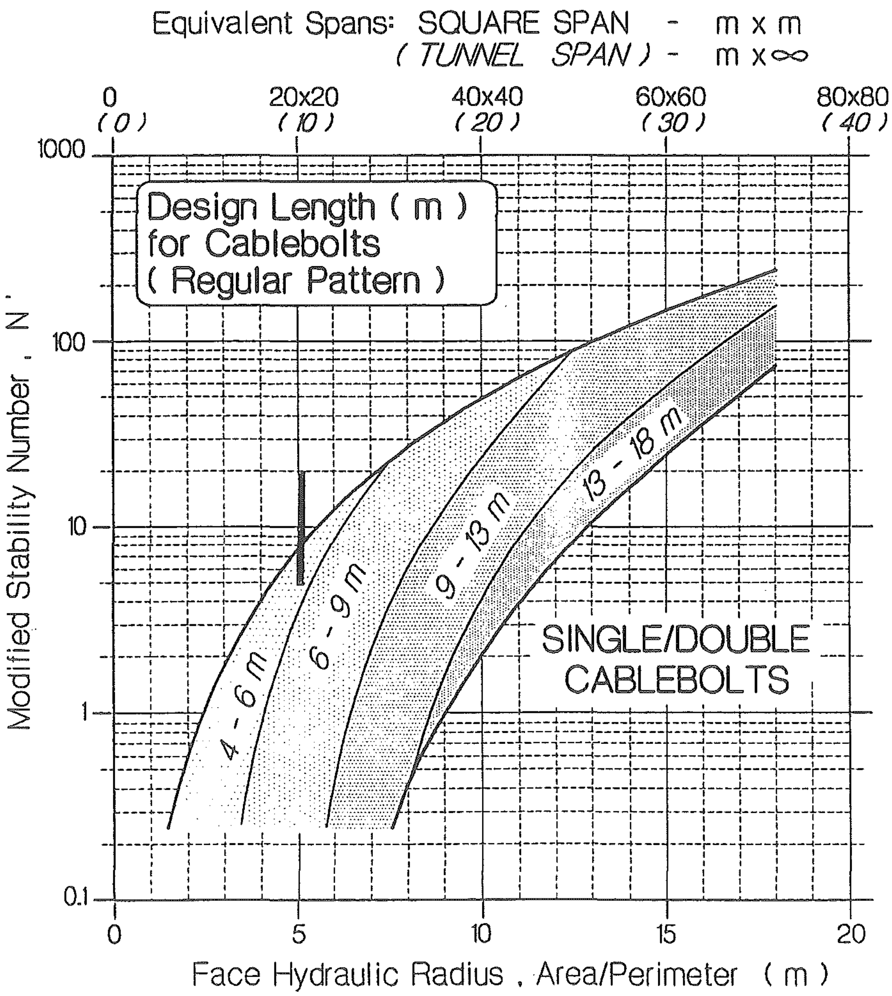
RECOMMENDED SPACINGS FOR DOUBLE STRAND CABLEBOLTS

Figure **7**



REFERENCE: Hutchinson, (1996)

Project No. 982-1418 Drawn W.T. Reviewed W.W.F. Date July 98

Golder Associates

RECOMMENDED MINIMUM LENGTHS FOR GROUTED CABLEBOLTS

Figure 8

FERGUSON SIMEK CLARK
ENGINEERS AND ARCHITECTS

July 24, 1998

Our File: 96-1480-11

Mr. Denis Gratton, P. Eng.
Royal Oak Mines Inc.
Yellowknife Division
PO Bag 3000
Yellowknife, NT
X1A 2M2

Dear Mr. Gratton,

Re: Arsenic Storage Chamber #15 (ASC-15)

Please find attached the plan for As#15, Upper and Lower Bulkheads. We have also included a copy of our calculations for your records.

Should you have any questions, please call me at your convenience.

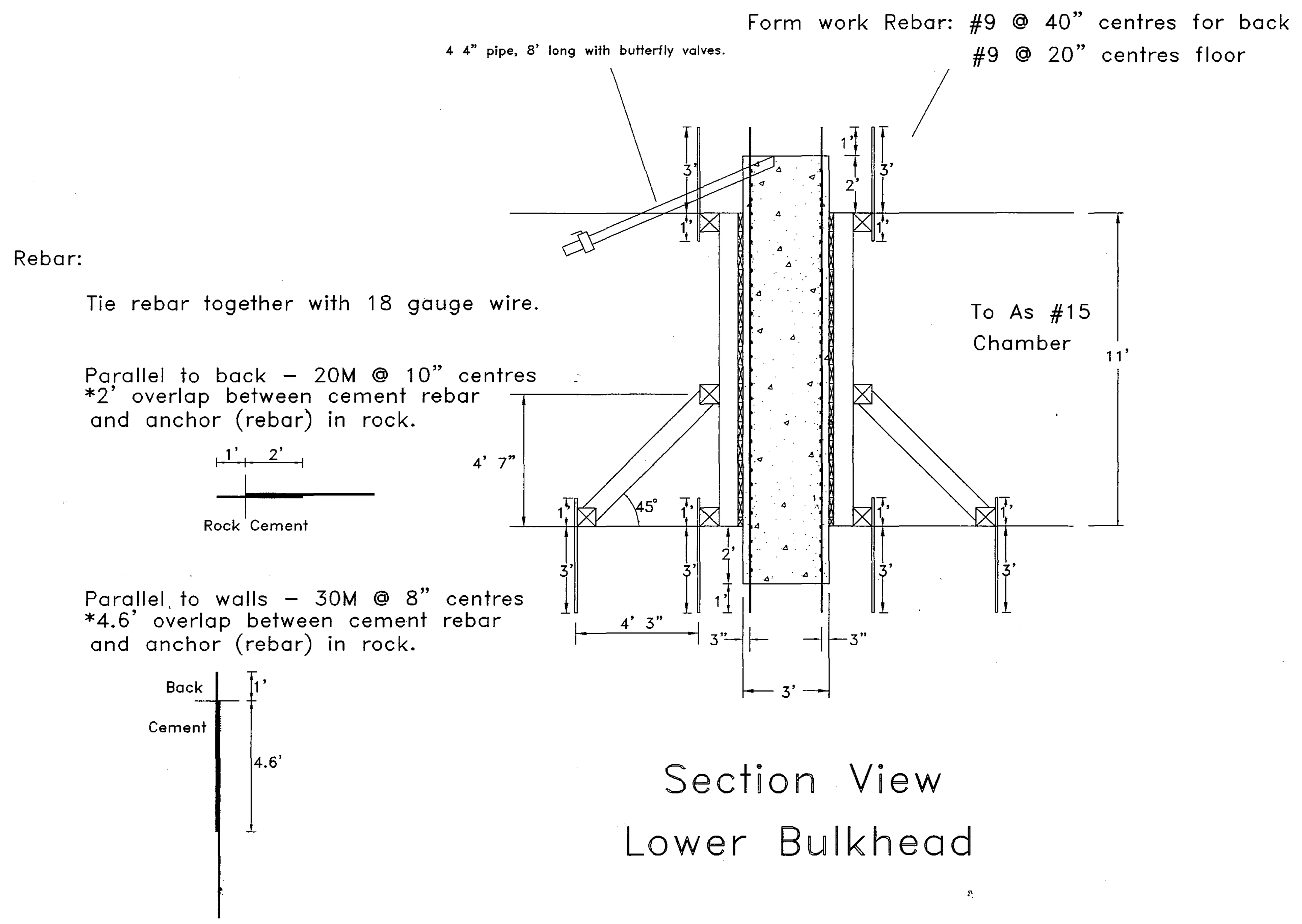
Sincerely yours,

PROFESSIONAL ENGINEER REGISTERED P.D. NOLAN N.W.T.

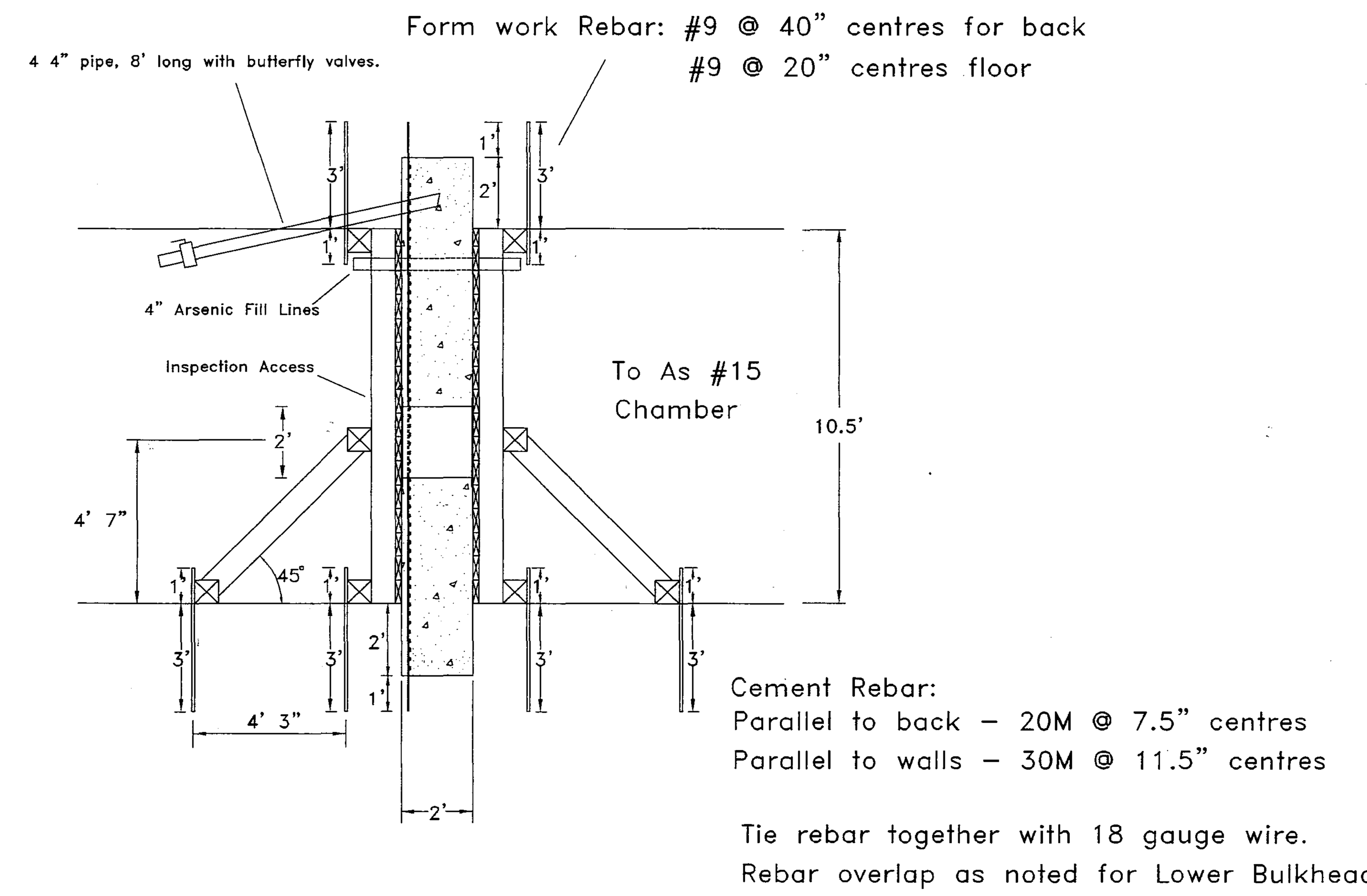
Philip D. Nolan, P.Eng.
Structural Engineer

THE ASSOCIATION OF PROFESSIONAL ENGINEERS, GEOLOGISTS & GEOPHYSICISTS OF THE NORTHWEST TERRITORIES
PERMIT NUMBER
P 003
FERGUSON SIMEK CLARK

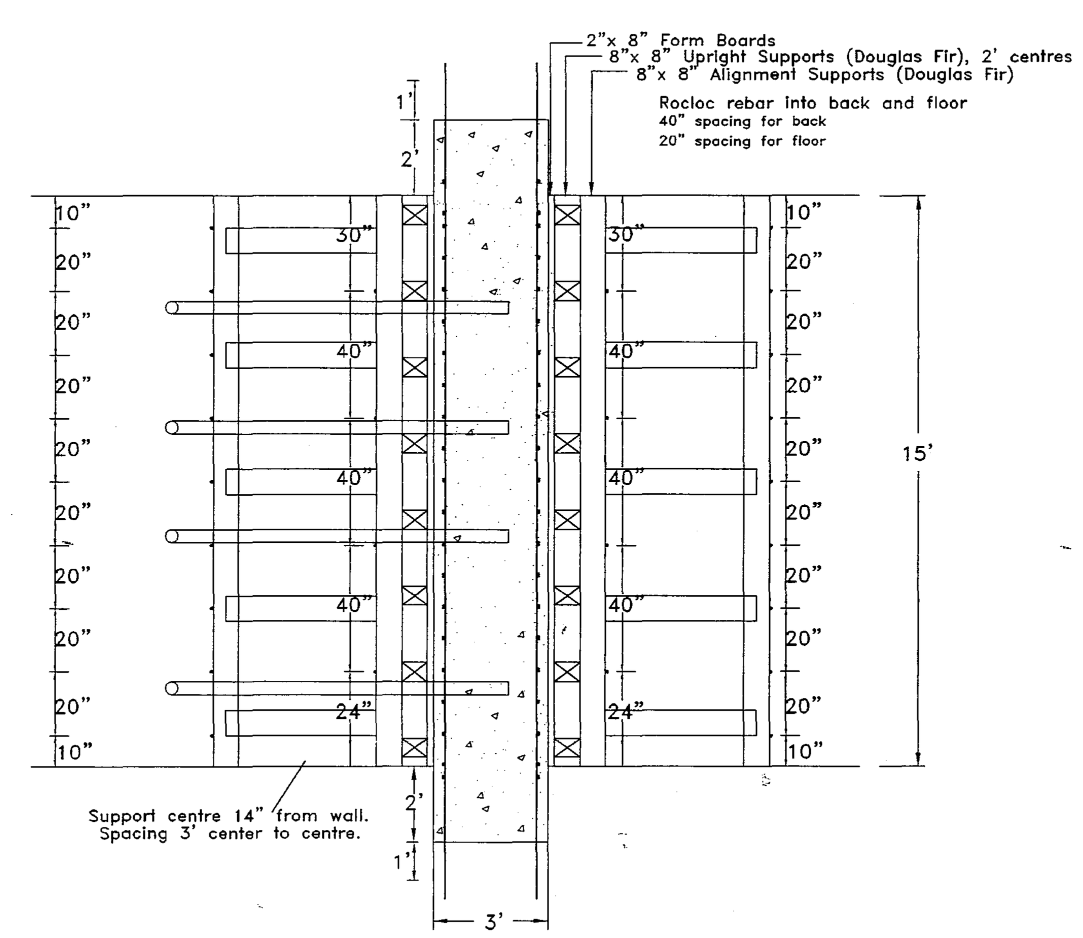
- YELLOWKNIFE 4910 53 Street, P.O.Box 1777, Yellowknife, NT Canada X1A 2P4 Tel: (867) 920-2882, Fax: (867) 920-4319, email: fscyk@fsc.ca
- IQALUIT building 1052, P.O.Box 1779, Iqaluit, NT Canada X0A 0H0 Tel: (867) 979-0555, Fax: (867) 979-5711, email: fsc@nunanet.com
- INUVIK 1 - 3 Council Crescent, P.O.Box 2385, Inuvik, NT Canada X0E 0T0 Tel: (867) 777-2427, Fax: (867) 777-3025, email: fscinuvk@permafrost.com
- WHITEHORSE 101 Copper Road, Whitehorse, YT. Y1A 2Z7 Tel: (867) 633-2400 Fax (867) 633-2481



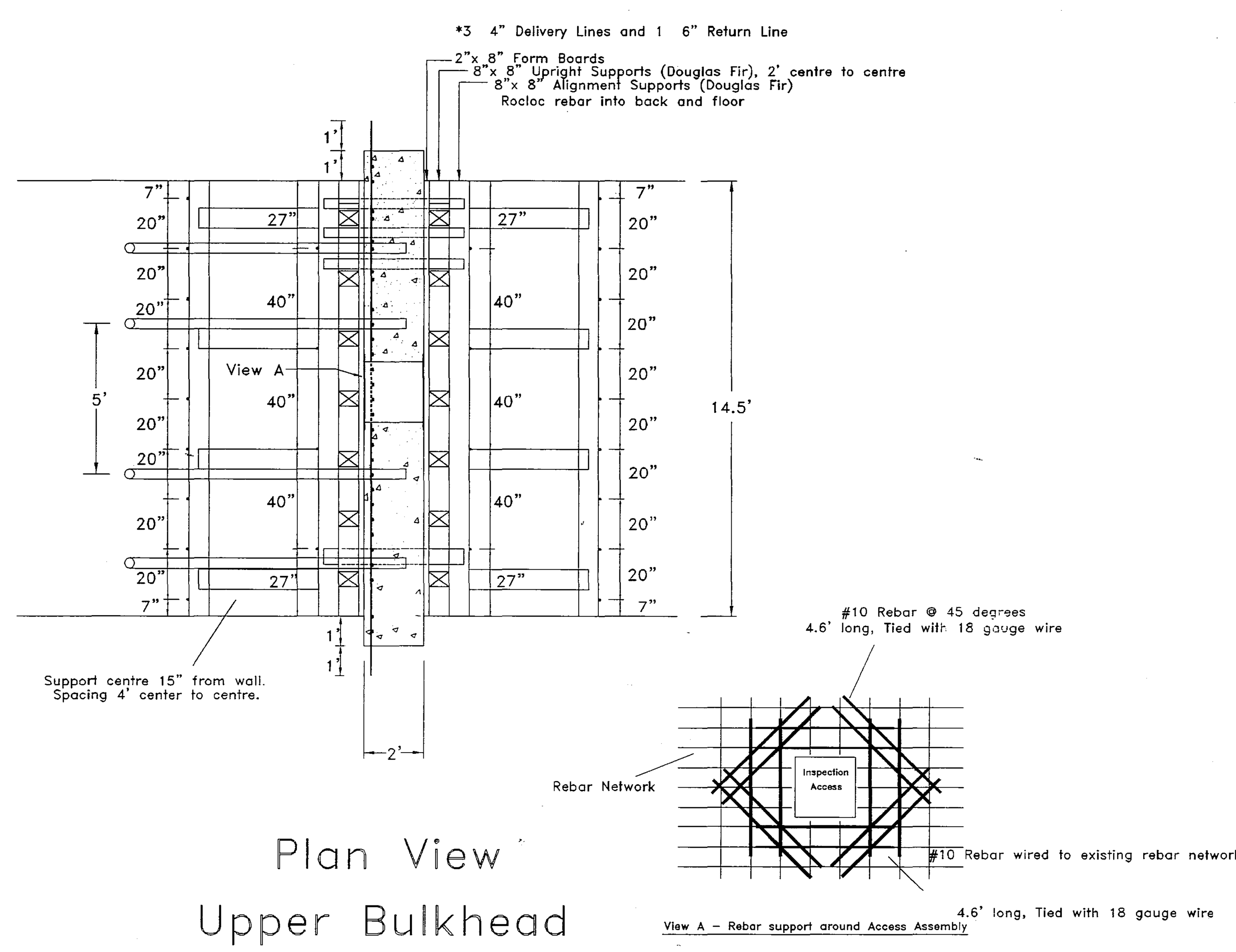
Section View
Lower Bulkhead



Section View
Upper Bulkhead



Plan View
Lower Bulkhead



Plan View
Upper Bulkhead

ROCK ANCHOR NOTES

1. ANCHORS SHALL BE DEFORMED REINFORCING BARS CONFORMING TO CSA STANDARD G30.12-M77, HAVING A MINIMUM YIELD STRENGTH OF 400 Mpa.
2. GROUT SHALL BE CELTITE ANCHORTITE OR APPROVED EQUAL, USED STRICTLY IN ACCORDANCE WITH THE MANUFACTURERS INSTRUCTIONS. GROUT SHALL BE MIXED IN QUANTITIES IMMEDIATELY REQUIRED. (MIN. COMP. STRENGTH 45 Mpa AT 28 DAYS)
3. DRILL 37mm DIA. HOLES TO THE DEPTHS SHOWN ON THE DRAWINGS. PRIOR TO GROUTING, EACH HOLE SHALL BE CLEANED AND FREE OF WATER. IF THE TEMPERATURE OF THE ROCK IS LESS THAN 5°C THE HOLES SHALL BE HEATED WITH STEAM. MAXIMUM TOLERANCE ON HOLE LOCATION SHALL BE (±) 50mm ON PLAN, AND (±) 150mm IN DEPTH.
4. AFTER PLACING THE GROUT THE ANCHORS MUST BE WEDGED IN THE CENTRE OF THE HOLE AND BE UNDISTURBED FOR AT LEAST 48 HOURS.
5. UNLESS SPECIFIED OTHERWISE, TESTING OF ANCHORS SHALL BE AS FOLLOWS: ONE ANCHOR IN EACH OF THE FIRST THREE FOOTINGS, AND 10% OF ALL SUBSEQUENT ANCHORS SHALL BE TESTED. (OR AS REQUIRED BY THE OWNER'S REPRESENTATIVE.) THE TEST LOAD SHALL BE THE EQUIVALENT OF THE DESIGN LOAD SHOWN ON THE DRAWINGS. THE ANCHORS SHALL BE TESTED 3 DAYS AFTER INSTALLATION. TEST PROCEDURES MUST BE APPROVED OWNERS REPRESENTATIVE. IN THE EVENT THE ANCHOR FAILS, FSC AND THE OWNER'S REPRESENTATIVE SHALL BE INFORMED. FURTHER INSTRUCTIONS WILL BE GIVEN.

CONCRETE REINFORCEMENT

1. DO CONCRETE REINFORCEMENT TO CAN/CSA-A23.1-M94.
2. DEFORMED BAR REINFORCEMENT SHALL BE BILLET STEEL TO CSA G30.12 M77. GRADE 400.
3. WELDED WIRE MESH TO CSA G30.5-M1983.
4. CONCRETE COVER TO REINFORCEMENT (mm):
   - .1 SURFACES POURED IN CONTACT WITH GROUND .... 75 (3 in.)
   - .2 FORMED SURFACES EXPOSED TO GROUND OR WEATHER:
     - BEAMS, PEIRS AND COLUMNS
     - PRINCIPAL REINFORCEMENT .... 50 (2 in.)
     - TIES AND STIRRUPS .... 40 (1.6 in.)
     - SLABS AND WALLS
     - 20M AND SMALLER .... 40 (1.6 in.)
   - .3 FORMED SURFACES NOT EXPOSED TO GROUND OR WEATHER
     - PIERS
     - TIES .... 40 (1.6 in.)
     - WALLS
     - 20M AND SMALLER .... 20 (0.8 in.)
     - 25M AND LARGER .... 1.5 X BAR DIAM
     - MINIMUM COVER .... AGG. SIZE
7. WALL REINFORCEMENT:
   ALL WALL REINFORCEMENT SHALL BE CONTINUOUS.
   LAP HORIZONTAL SPLICES A MINIMUM OF 40 BAR DIAMETERS OR 400 MM. (15.75 Inches)
8. SEE DRAWINGS FOR SLEEVES, NAILERS, INSERTS, ETC. TO BE ENCASED IN CONCRETE.
9. OBTAIN ENGINEERS REVIEW OF IN PLACE REINFORCEMENT PRIOR TO PLACING CONCRETE.

CONCRETE NOTES

1. DO CONCRETE WORK TO CAN/CSA A23.1-M94.
2. TYPE OF CEMENT: .... TYPE 50
3. CONCRETE STRENGTH AT 28 DAYS (MPa): .... 25 (3650 psi)
4. MAX. AGGREGATE SIZE (mm): .... 20 (0.8 in.)
5. SLUMP (mm) .... 50 - 75 (2 - 3 in.)
6. AIR ENTRAINMENT:
   1. WALLS .... 4% - 6%
7. CONCRETE AND MATERIALS WILL BE TESTED BY A TESTING AGENCY APPOINTED BY THE CONSULTANT.
8. DO NOT USE CALCIUM CHLORIDE ADMIXTURES.
9. SEE DRAWINGS FOR SLEEVES, NAILERS, INSERTS, ETC. TO BE ENCASED IN CONCRETE. REPORT TO THE ENGINEER ANY SLEEVE OR OPENING REQUIRED THROUGH WALLS THAT ARE NOT SHOWN ON STRUCTURAL DRAWINGS.
10. WHEN AIR TEMPERATURE IS AT OR BELOW 5 C OR WHEN THERE IS A PROBABILITY OF ITS FALLING TO THAT LIMIT DURING THE PLACING PERIOD, CONCRETE MIXING, PLACING AND CURING SHALL COMPLY WITH THE LATEST EDITION OF ACI-STANDARD 306, RECOMMENDED PRACTICE FOR COLD WEATHER CONCRETING.
11. REINFORCING STEEL SHALL BE DEFORMED BARS CONFORMING TO CSA STANDARD G30.12-M77 HAVING A MINIMUM YIELD STRENGTH AS FOLLOWS:

10M	-	300Mpa
Bar #3	-	43525 psi
15M TO 35M	-	400Mpa
Bar # 5 TO #11	-	58025 psi

P.O. NOLAN

Bar Scale

Royal Oak Mines Inc.

TITLE: As #15 Upper & Lower Bulkheads			
LEVEL: 100		COMPLEX: UBC	
DRAWN: M.D./MSL	CHECK:	SCALE: 1/4" = 1' 0"	
DATE: JULY 20, 1998			
REV:		REV:	

MSL C:\MASTER\ARSENIC\BLKHD15S.DWG

From The Desk Of ______________________

PLANT AND HEAD OFFICE
1709 - 8th STREET
NISKU, ALBERTA T9E 7S8
TEL: (403) 955-3390
FAX: (403) 955-2074

B.C. SALES OFFICE
12720 - 82nd AVENUE
SURREY, BC V3W 3G1
TEL: (604) 596-1747
FAX: (604) 596-1138

CALGARY SALES OFFICE
206, 2723 - 37th AVENUE N.E.
CALGARY, ALBERTA T1Y 5R8
TEL: (403) 250-7871
FAX: (403) 250-7178

WINNIPEG SALES OFFICE
108 - 167 ST. MARY'S ROAD
WINNIPEG, MANITOBA R2H 1J1
TEL: (204) 237-3528
FAX: (204) 237-1099

OMEGA
OMEGA JOISTS INC.
open web steel joists

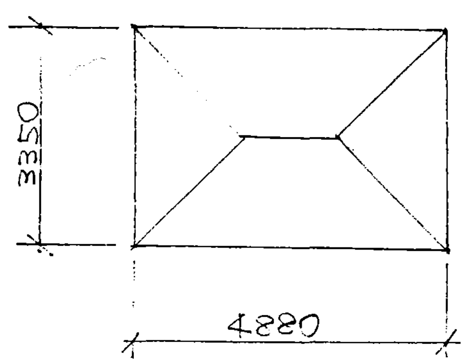
#15 ARSENIC LOWER BULKHEAD DESIGN

DESIGN ASSUMPTIONS

- USE LIMIT STATE DESIGN
- USE CAN3-A23.4-M90
- $f'_c = 25$ MPa
  $f_y = 400$ MPa
- SG = 3.4 ARSENIC TRIOXIDE
  SG = 1.0 WATER

LOAD CALCULATION

- ASSUME COMBINED DENSITY OF WATER AND ARSENIC TRIOXIDE TO BE 1.7 kN/m^3
- MAXIMUM HEAD = 46 m.
- $1.7 \times 46 \times 9.81 = 767 \; kN/m^2$



$$m = \frac{A}{B} = \frac{3350}{4880} = 0.69$$

BASED UPON TABLE E-2, $C_A = 0.068$
$C_B = 0.016$

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
98-7-24
N.W.T.

From The Desk Of ______________________

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FAX: (403) 955-2074

☐ B.C. SALES OFFICE
12720 - 82nd AVENUE
SURREY, BC V3W 3G1
TEL: (604) 596-1747
FAX: (604) 596-1138

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206, 2723 - 37th AVENUE N.E.
CALGARY, ALBERTA T1Y 5R8
TEL: (403) 250-7871
FAX: (403) 250-7178

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108 - 167 ST. MARY'S ROAD
WINNIPEG, MANITOBA R2H 1J1
TEL: (204) 237-3528
FAX: (204) 237-1099

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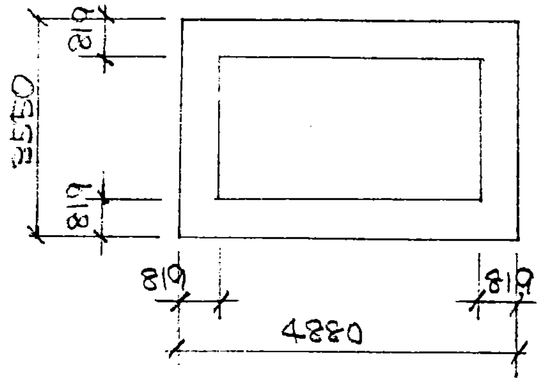
OMEGA JOISTS INC.
open web steel joists

ASSUME SHEAR IS DISTRIBUTED OVER:



TOTAL LOAD ACTING ON WALL

$= (767\ kN/m^2)\,1.5\,(1.712)^{*}\,3.242 =$

$= 6386\ kN$

ASSUME APPROXIMATE DISTRIBUTION OF LOAD WILL BE 60% TO B AND 40% TO A.

∴ CRITICAL SECTION BECOMES

$$0.60\,(6386) = \frac{3832}{2\,(3.242)} = 590\ kN/m$$

$$0.40\,(6386) = \frac{2554}{2\,(1.712)} = 746\ kN/m$$

AVERAGE STRESS = 643 kN/m ⟵ USE THIS

BASED UPON $V_c = (0.4 \times \phi_c \sqrt{f'_c}\, bd) = 0.4(1)\,0.60\sqrt{25}\,(9.980)\,819$

$= 9780\ kN$

$$v_u = \frac{V_u}{b_w d} = \frac{643\,(10^3)}{1000\,(819)} = 0.78\ MPa$$

$$v_c = 1.2\ MPa$$

AS A CHECK, USE $W = 746\ kN/m$

$$v_f = \frac{746\,(10^3)}{1000\,(819)} = 0.91\ MPa \quad \therefore OK$$

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
98-7-24
N.W.T.

2

RE: DATE: 24JULY98   SCALE: 1/8"=

From The Desk Of

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MOMENT CAPACITY - DIRECTION "A"

$M_p = 0.068 \times 767\ kN/m^2 \times 3.35^2 \times 1.5$
$= 378\ kNm$

$K_r = \frac{M_r 10^6}{bd^2} = \frac{378 \times 10^6}{1000(819)^2} = 1.13$

- LOOK UP ρ IN TABLE 2.2 AND CHECK IT FALLS IN THE RANGE OF ρ_{min} TO ρ_{max}

$\rho = 0.40\%$

$\rho_{min} = \frac{1.4}{f_y} = \frac{1.4}{400} = 0.035\%$

$\rho_{max} = 0.36 \beta_1 \frac{f_c'}{f_y} = 1.91\%$

ρ_{min} (SLAB) $= 0.0020$

∴ REQ'D AREA OF STEEL

$A_s = \rho bd$
$= 0.0040 \times 1000 \times 819 = 3276\ mm^2/m$

∴ PROVIDE No. 30 BAR @ $\frac{3276}{700} = 4.68$ SPACES ; USE 5

∴ WE HAVE No. 30 BAR @ 200 mm O.C.

$M_r = 0.0043(0.85)400\left[1 - \frac{0.0043(0.85)400}{1.7(0.60)25}\right]1000(819)^2 = 924.4\ kNm$

MOMENT CAPACITY - DIRECTION "B"

$M_A = 0.016 \times 767 \times 4.88^2 \times 1.5$
$= 438\ kNm$

$K_r = \frac{M_r 10^6}{bd} = \frac{438 \times 10^6}{1000(819)^2} = 0.65$ ∴ ρ_{min} CONTROLS

$\rho = 0.0020$

REGISTERED PROFESSIONAL ENGINEER
P. D. NOLAN
98-7-24
N.W.T.

3

RE:

DATE: 24 JULY 98

SCALE: 1/8"=

From The Desk Of

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REQ'D AREA OF STEEL:

$$A_s = \rho b d$$

$$= 0.0020(1000)819 = 1638 \text{ mm}^2$$

PROVIDE NO. 20 @ 250 mm O.C.

$$A_{s\,PROVIDED} = 4(500) = 2000 \text{ mm}^2$$

$$M_r = \rho \phi_s f_y \left[1 - \frac{\rho \phi_s f_y}{1.7 \phi_c f'_c}\right] b d^2$$

$$= 0.00244(0.85)400\left[1 - \frac{0.00244(0.85)400}{1.7 \times 0.60 \times 25}\right]1000(819)^2$$

$$M_r = 529.5 \text{ kNm}$$

ANCHORAGE INTO ROCK

SHEAR MAXIMUM ALONG LONG WALL,

$$\therefore \frac{767 \text{ kN/m}^2 (3.35)\, 0.7}{2} = 900 \text{ kN/m} \quad \text{(ESTIMATED DISTRIBUTION)}$$

1) BASED UPON ALLOWABLE BEARING CAPACITY OF ROCK @ 3830 kPa

$$\frac{900}{(1)(t)} = 3830 \text{ kPa} \qquad t = 235 \text{ mm}$$

2) BASED UPON PREVIOUS EXPERIENCE $t = \frac{h}{2}$

$$= \frac{914}{2} = 457 \text{ mm.}$$

REGISTERED PROFESSIONAL ENGINEER
P. D. NOLAN
98-7-24
N.W.T.

4

DATE: 24 JULY 98 SCALE: 1/8"=

From The Desk Of

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WINNIPEG SALES OFFICE
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WINNIPEG, MANITOBA R2H 1J1
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SUMMARY

WALL THICKNESS	h = 914 mm	
	d = 819 mm	
REINFORCEMENT	NO. 20 @ 250 mm O.C HORIZONTAL	
	NO. 30 @ 200 mm O.C. VERTICAL.	
SHEAR CAPACITY	1.2 MPa	
MOMENT CAPACITY	924.4 kNm VERTICAL	
	529.5 kNm HORIZONTALLY.	
HITCH DIMENSIONS	DESIGN REQUIREMENT	295 mm
	PROVIDED	457 mm (MINIMUM)

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
98-7-24
N.W.T.

RE:

DATE: 24 JULY 98

SCALE: 1/8"=

From The Desk Of

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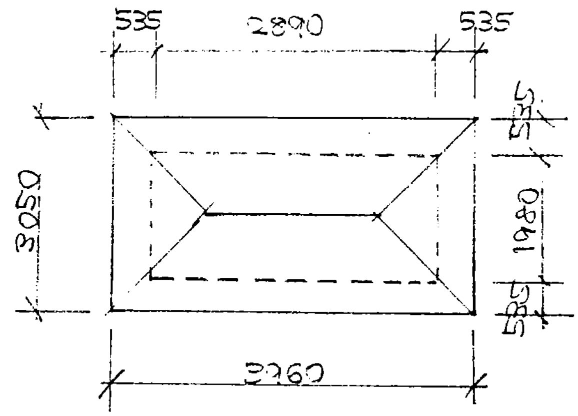
CISC

#15 ARSENIC UPPER BULKHEAD DESIGN



MAXIMUM HEAD = 24.4 m

DENSITY = 1.7 kN/m^3

MAXIMUM SPECIFIED LOAD

$= 9.81 \times 1.7 \times 24.4 = 407\ kN/m^2$

$$m = \frac{A}{B} = \frac{3050}{3960} = 0.77$$

BASED UPON TABLE 3-2, $C_A = 0.058$

$C_B = 0.021$

ASSUME SHEAR IS DISTRIBUTED OVER PERIMETER AT DISTANCE "d" FROM SUPPORT. WE ARE GIVEN $h = 610$

$d = 535$

∴ TOTAL LOAD CONTRIBUTING TO SHEAR

$407(1.5)\,2890\,(1980) = 3493.4\ kN$

ASSUME DISTRIBUTION:

"A" $\quad 0.6(3493.4) = \dfrac{2096\ kN}{2(2890)} = 362.6\ kN/m$

"B" $\quad 0.4(3493.4) = \dfrac{1397.4}{2(1980)} = 352.9\ kN/m$

BASED UPON $V_c = 0.4 \lambda \phi_c \sqrt{f'_c}$

$= 1.2$ MPa

$v_c = \dfrac{[illegible] \times 10^3}{1000(535)} = 0.62$ MPa

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
98-7-24
N.W.T.

6

RE:

DATE: 24 JULY 98

SCALE: 1/8"=

From The Desk Of ____________

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MOMENT CAPACITY ON "A"

$M_f = 0.058 \times 407\ kN/m^2 \times 3.05^2 \times 1.5$

$= 329.4\ kNm$

$K_r = \dfrac{M_r 10^6}{bd^2} = \dfrac{329.4 \times 10^6}{1000(335)^2} = 1.5$

$\rho_{min} = 0.0020$

$\rho_{max} = 0.0191$

ACTUAL $\rho = 0.0035$

REQ'D AREA OF STEEL $A_s = \rho bd$

$= 0.0035(1000)335$

$= 1173\ mm^2$

PROVIDE NO.30 BAR @ 300 mm O.C.

$A_{s\ PROVIDED} = 2333\ mm^2$

$M_r = 0.0044(0.85)400\left[1 - \dfrac{0.0044(0.85)400}{1.7 \times 0.60 \times 25}\right]1000(335)^2$

$= 403\ kNm$

REGISTERED PROFESSIONAL ENGINEER
P. D. NOLAN
98-2-24
N.W.T.

MOMENT CAPACITY ON "B"

$M_f = 0.021 \times 407 \times 3.96^2 \times 1.5$

$= 201\ kNm$

$K_r = \dfrac{M_r 10^6}{bd^2} = \dfrac{201 \times 10^6}{1000(335)^2} = 0.7 \therefore$ USE $\rho_{min} = 0.0020$

$A_s = \rho bd = 0.0020(1000)335 = 1177\ mm^2$

7

RE:

DATE: 24/02/98

SCALE: 1/8"–

From The Desk Of

PLANT AND HEAD OFFICE
1709 - 8th STREET
NISKU, ALBERTA T9E 7S8
TEL: (403) 955-3390
FAX: (403) 955-2074

B.C. SALES OFFICE
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PROVIDE NO. 20 @ 200 mm O.C.

$A_{s\,PROVIDED} = 1500\ mm^2$

$$M_r = 0.0028(0.85)400\left[1 - \frac{0.0028 \times 0.85 \times 400}{1.7 \times 0.60 \times 25}\right]1000(335)^2$$

$$= 262.3\ kNm$$

ANCHORAGE INTO ROCK

MAXIMUM SHEAR ALONG "B"

$$\frac{407(3.05)0.7}{2} = 434.5\ kN/m$$

BASED UPON BEARING CAPACITY OF ROCK @ 3830 kPa

$$\frac{434.5}{1t} = 3830\ kPa \qquad t = 0.113\ m \text{ OR } 113\ mm$$

BASED UPON PREVIOUS EXPERIENCE $t = \frac{h}{2} = \frac{610}{2} = 305\ mm$

OWNER HAS SUPPLIED 457 mm THK ∴ OK.

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
99-7-24
N.W.T.

RE:

DATE: 24 JULY 98

SCALE: 1/8"=

From The Desk Of ______________________

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SUMMARY

WALL THICKNESS	h = 610 mm d = 535 mm
REINFORCEMENT	NO. 20 @ 200 mm O.C. HORIZONTAL NO. 30 @ 300 O.C. VERTICAL
SHEAR CAPACITY	1.2 MPa
MOMENT CAPACITY	403 kNm VERTICAL 201 kNm HORIZONTAL
HITCH	DESIGN REQUIREMENTS 113 mm PROVIDED 457 mm

REGISTERED PROFESSIONAL ENGINEER
P.D. NOLAN
N.W.T.

RE:

DATE: 24 JULY 92

SCALE: 1/8"=

Explosives

11/24/97

ICI Explosives Canada

Technical Services Group
49, Diamond Ave., Box 4360,
Spruce Grove, T7X 3B5
Telephone: 403-962-3770
Facsimile: 403-962-1980

Richard Allan
Manager - Mining Projects
Royal Oak Mines Inc.
5501 Lakeview Dr.
Kirkland, WA 98033
USA

Subject: Wall Control for #15 Arsenic Storage Chamber

Dear Rick:

This letter will give you recommendations for wall control and minimize wall damage for you arsenic storage area blasts.

1.0 Background

The chamber is designed to store dust from the roaster that contains arsenic trioxide. It is very important that wall damage be limited to ensure proper confinement of the waste material.

The storage area is designed to be 90 ft high, about 45 ft wide and 200 ft long. The blast holes will be 2 in. in diameter.

The stope can be accessed from the top and bottom cuts and all holes will be vertical, with no dip or dump.

The initial plan was to drill 90 ft downholes from the top access, with most holes breaking through the lower access. This plan was latter changed to 60 ft. downholes and 35 ft. upholes.

2.0 Wall Control Parameters

There are several factors that will influence wall control results. The most important are:

- geology;
- drilling accuracy;
- blast design;
- timing.

2.1 Geology

The location of the chamber being decided, the geology cannot be modified. However, since the location was chosen because of the rock quality, one can assume that there are not many open joints and faults that can adversely affect wall control operations.

2.2 Drilling accuracy

One point of great concern when dealing will small diameter holes is drilling accuracy. Holes of 2 in. diameter will certainly deviate over a long distance.

We have recently conducted a hole deviation survey in larger upholes (2-1/8 in.) and found that there is no significant deviation in the first 15 to 25 ft. However, a hole can easily deviate 1 degree every 3 ft. after that. Therefore, it is quite possible that a 60 ft. hole would deviate by 4 ft. Ideally, a larger hole would be required.

2.2 Blast design

Wall control blast design is based on a series of parameters. These are:

Hole Spacing:	10 to 18 times the hole diameter
Stand-off distance to production holes:	50 to 75% of normal burden
Shear Factor:	0.1 to 0.14 lb/ft^2
Decoupling factor:	0.1 to 0.3

The shear factor is the amount of explosive used per square foot of wall, while the decoupling factor is the ratio between the charge diameter and the hole diameter.

The choice of explosive will be dictated by the application. XACTEX is a rigid cartridged NG based explosive developed for wall control. This product comes with plastic couplers that will help in charging upholes. Cartridge size is 19mm x 600mm.

The plastic couplers cannot support the weight of the cartridges in downholes, but this can be easily solved. The cartridges can be tapped to a length of CORDTEX 18 as they are lowered into the holes. This will also ensure that all cartridges detonate if any gap occurs between XACTEX cartridges.

For both upholes and downholes, a suitable primer is needed for XACTEX. For downholes, a cartridge of MAGNAFRAC 3000, 25mm x 300mm, primed with a detonator can be lowered to the toe of the hole. For upholes, two cartridges of MAGNAFRAC 3000 can be pushed up to the toe of the hole. The first cartridge will have the detonator, and the second will be tamped to keep the cartridges in place. In both cases, it is very important that the XACTEX cartridges come in direct contact with these primer charges.

Also, when loading XACTEX, plugs or "birdies" must be inserted every 3 or 4 cartridges. This will stop any channel effect from desensitizing the cartridges.

Given these product choices, the blast parameters for wall control are:

	Upholes and Downholes
Wall hole spacing	2 ft. (see figure 1)
Stand-off	3 ft. (see figure 1)
Explosive	XACTEX
Shear Factor	0.13 lb/ft^2
Decoupling Factor	0.37

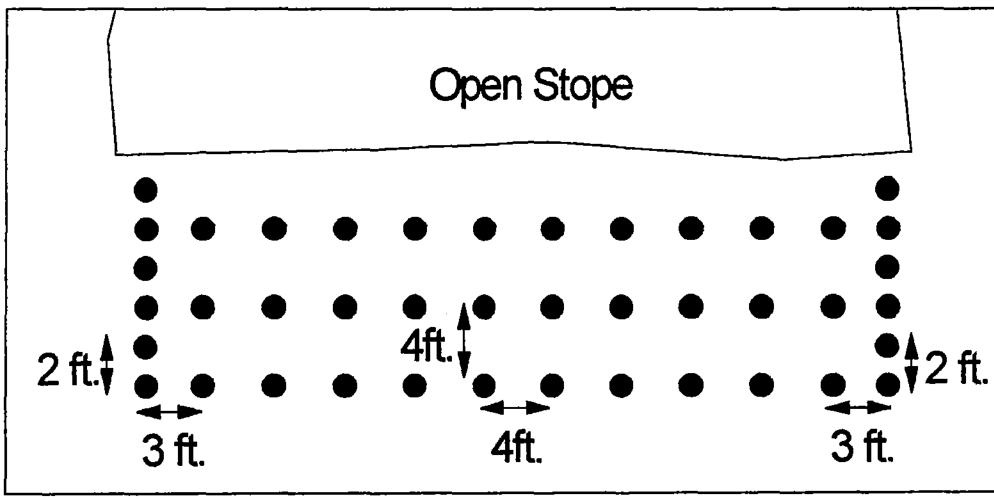


Figure 1: Spacing and stand-off.

There is another alternative: PRIMAFLEX could also be used. This is a high core load detonating cord designed for wall control. Normal design parameter do not apply to PRIMAFLEX as the PETN explosive behaves very differently than traditional explosives. Instead, this product can be used as a one-to-one substitute for XACTEX in the appropriate applications. Loading of this product in long 2 in. downholes could be difficult, so I would recommend using XACTEX.

2.3 Timing

To reduce vibrations levels, the number of production holes fired per delay should not exceed two. Figure 2 describes the proposed timing for these blasts. Two rings can be fired per blast with the available delays. If three rings are fired per blast, than three or four holes will be fired per delay, resulting in an increase in vibrations and potential wall damage.

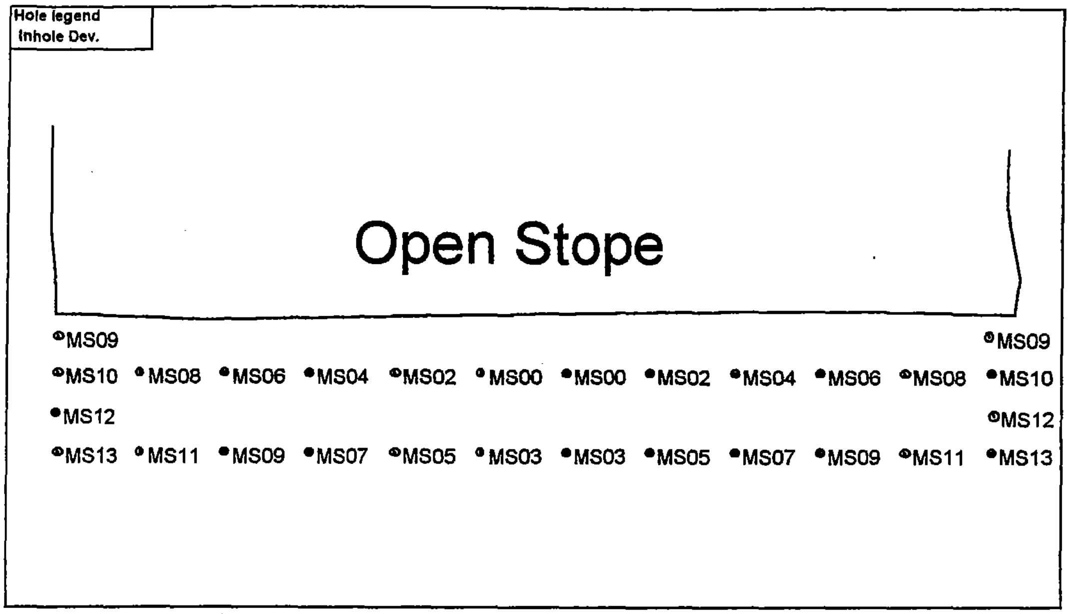


Figure 2: Proposed timing sequence.

The MS# represent the EXEL MS period.

This timing sequence should be used for both upholes and downholes.

3.0 Conclusion

Given the hole diameter and proposed hole lengths, careful attention should be given to drilling accuracy and hole deviation. A deviation of more than 12 in. for the wall holes would render all wall control attempts ineffective.

This being said, the use of XACTEX in the wall holes on a 2 ft. spacing and a 3 ft. stand-off to regular holes (see figure 1) should limit the amount of wall damage, if deviation is controlled.

Upholes would be primed with two cartridges of MAGNAFRAC 3000; downholes with one.

Upholes would be loaded by pushing up the cartridges one after the other; downholes would be loaded by taping the XACTEX cartridges to CORDTEX 18 and lowering them. Plugs or "birdies" must be inserted every 3 or 4 cartridges in both up and downholes.

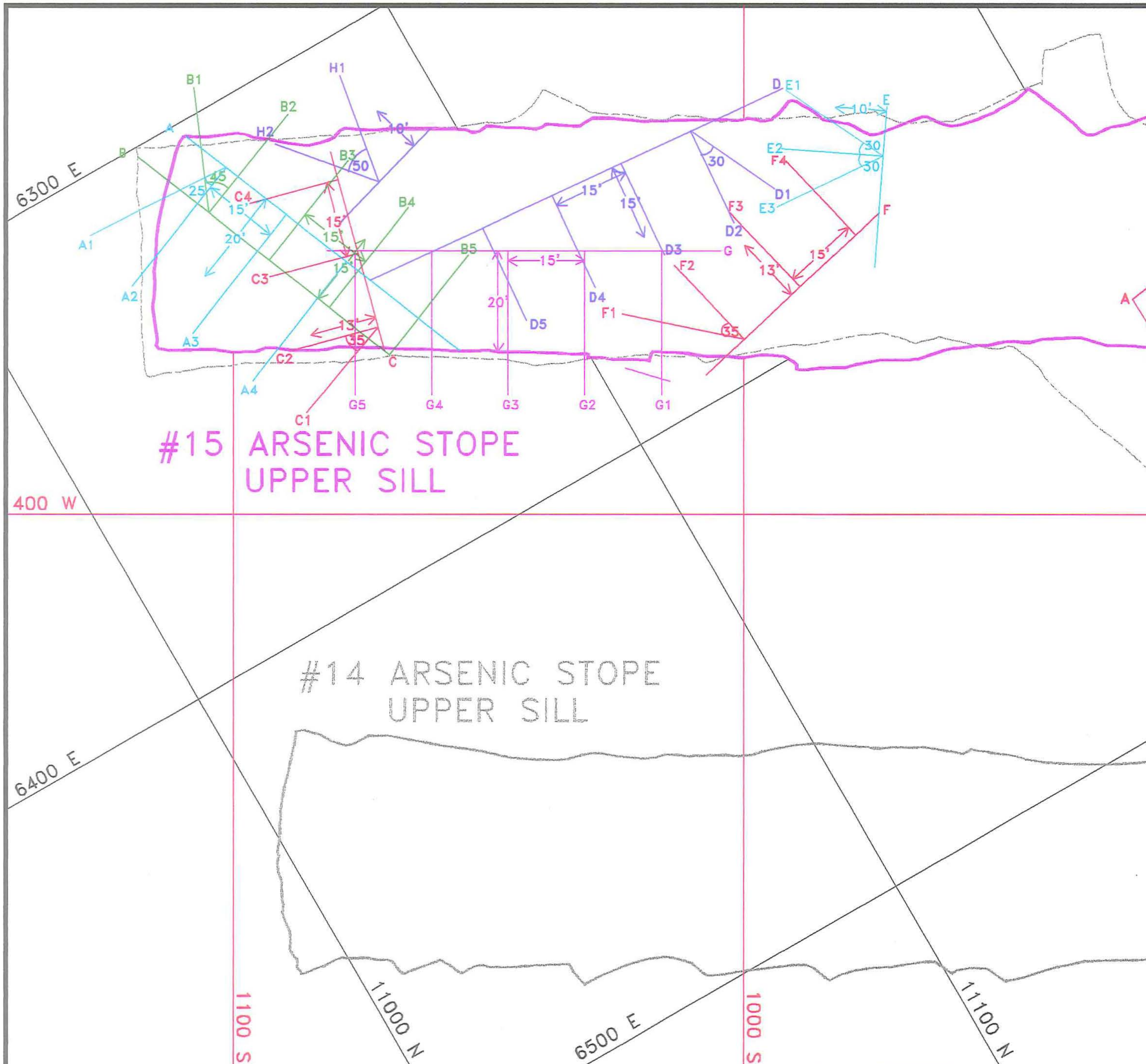
The number of hole per delay should not exceed two, as shown in figure 2.

If you have any question, please do not hesitate to contact me,

Best regards,

Remi Aguila
Technical Service Representative, Technical Services Group

c.c.	J. Stard	
	D. Gratton	
	R. Moore	
	T. Matts	ICI Vancouver
	P. Lightfoot	ICI North York
	M. Stern	ICI Edmonton
	T.Tobin	XL Calgary
	B. Pankhurst	XL Calgary
	G. Marston	XL Edmonton

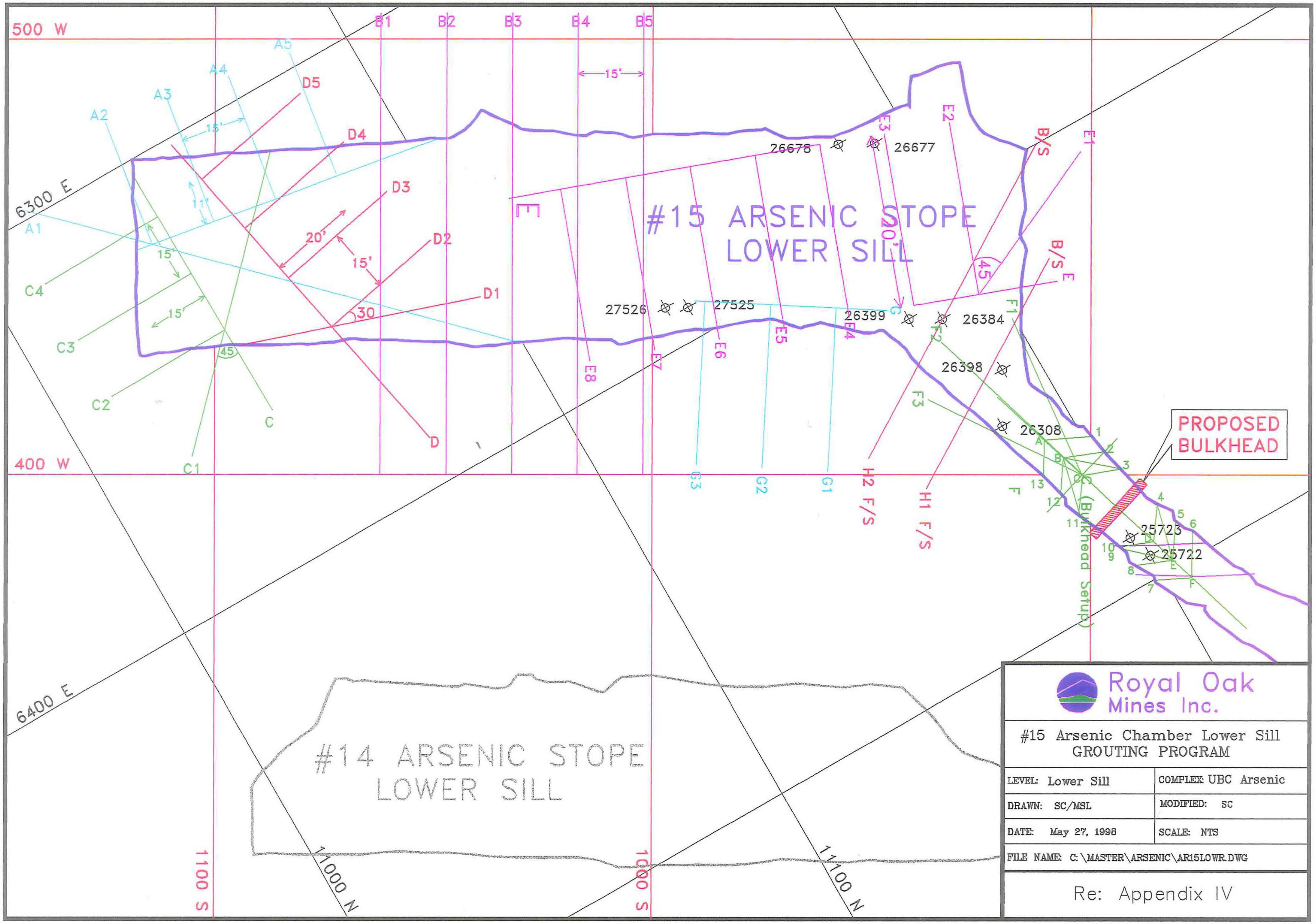


Royal Oak Mines Inc.

#15 Arsenic Chamber Upper Sill
GROUTING PROGRAM

LEVEL: Upper Sill	COMPLEX: UBC Arsenic
DRAWN: SC/MSL	MODIFIED: SC
DATE: May 26, 1998	SCALE: NTS
FILE NAME: C:\MASTER\ARSENIC\AR15UGRO.DWG	

Re: Appendix IV

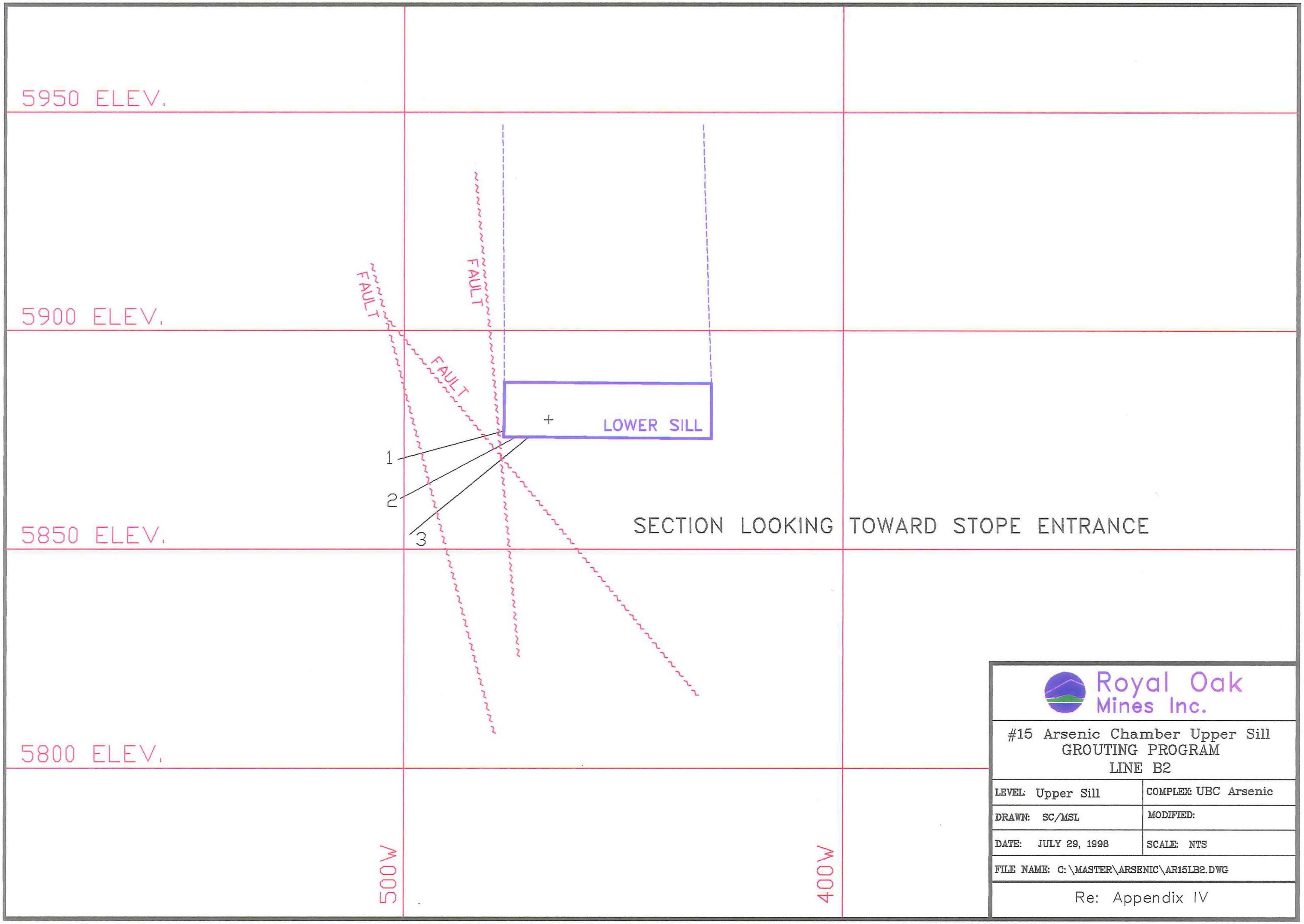


Royal Oak Mines Inc.

#15 Arsenic Chamber Lower Sill
GROUTING PROGRAM

LEVEL: Lower Sill	COMPLEX: UBC Arsenic
DRAWN: SC/MSL	MODIFIED: SC
DATE: May 27, 1998	SCALE: NTS
FILE NAME: C:\MASTER\ARSENIC\AR15LOWR.DWG	

Re: Appendix IV



5950 ELEV.
5900 ELEV.
5850 ELEV.
5800 ELEV.
500W
400W
FAULT
FAULT
FAULT
LOWER SILL
1
2
3
SECTION LOOKING TOWARD STOPE ENTRANCE
Royal Oak Mines Inc.
#15 Arsenic Chamber Upper Sill
GROUTING PROGRAM
LINE B2
LEVEL: Upper Sill
COMPLEX: UBC Arsenic
DRAWN: SC/MSL
MODIFIED:
DATE: JULY 29, 1998
SCALE: NTS
FILE NAME: C:\MASTER\ARSENIC\AR15LB2.DWG
Re: Appendix IV