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ARROW DEPOSIT CONCEPTUAL LEVEL MINE OPERATING COST ESTIMATE

1.0 Introduction

The purpose of this report is to present a conceptual level viable mine plan for the Arrow deposit currently being drill defined by NexGen Energy. This plan is then used to estimate possible mine operating costs for the mining methods selected.

2.0 Basic Mine Design

The proposed mine plan has numerous similarities to the Millennium mine plan proposed by the author in a previous pre-feasibility study. Figure 1 illustrates the Millennium Project pre-feasibility level mine design.

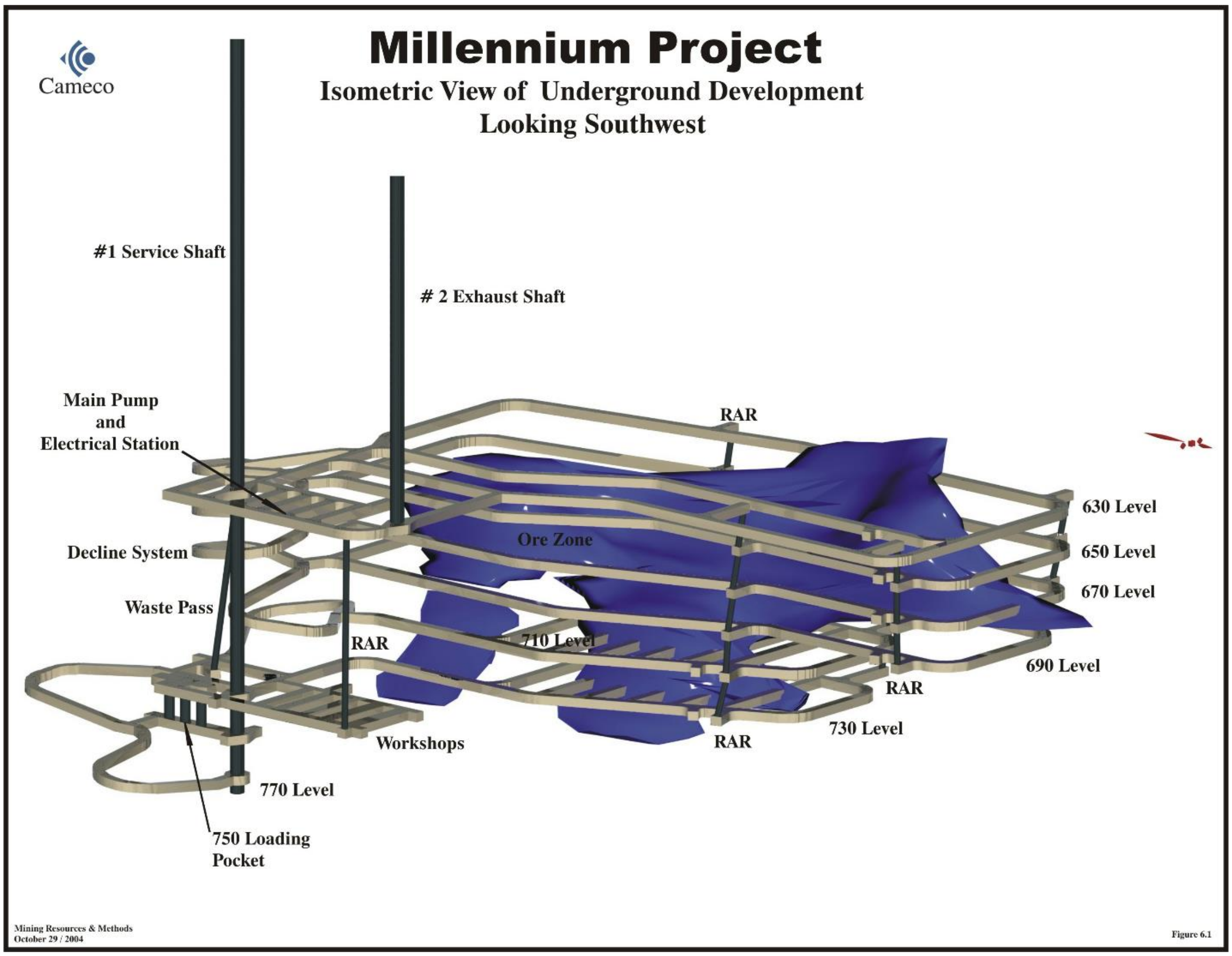

Figure 1 Isometric View of Millennium Project Pre-feasibility Level Mine Design

The Arrow deposit however has two key advantages; shaft sinking is through basement rock, hence no grout covers are necessary when sinking, and the upper mine level is much higher than at Millennium (375 m vs. 630 m). This shortens up the exhaust shaft length and reduces power requirements for pumping mine water to surface for instance. Figure 2 applies this design approach to the Arrow deposit in section. Figure 3 illustrates a conceptual 375 m Level plan.

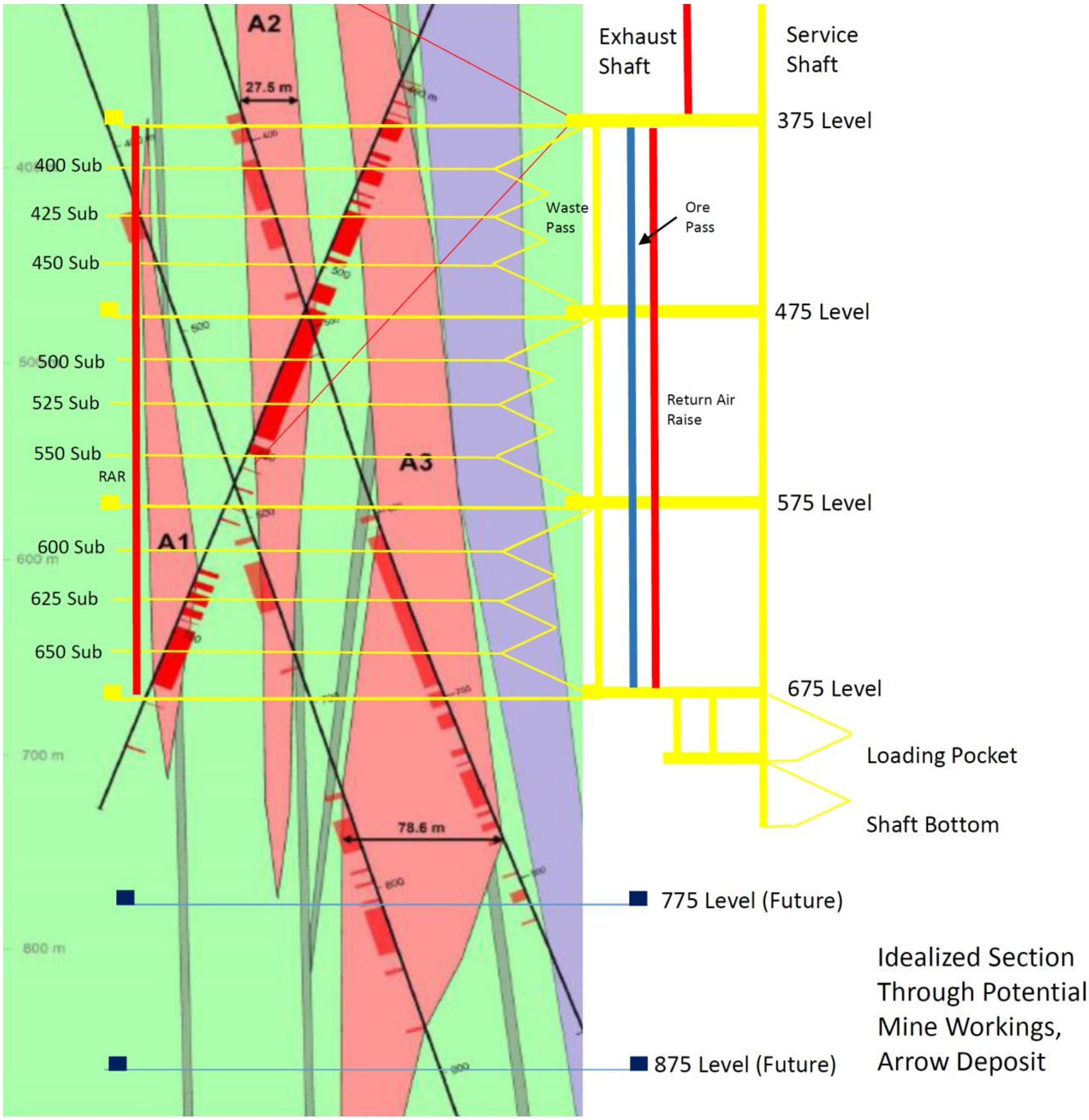

Figure 2 Conceptual Cross Section Through the Arrow Deposit Illustrating Key Mine Infrastructure

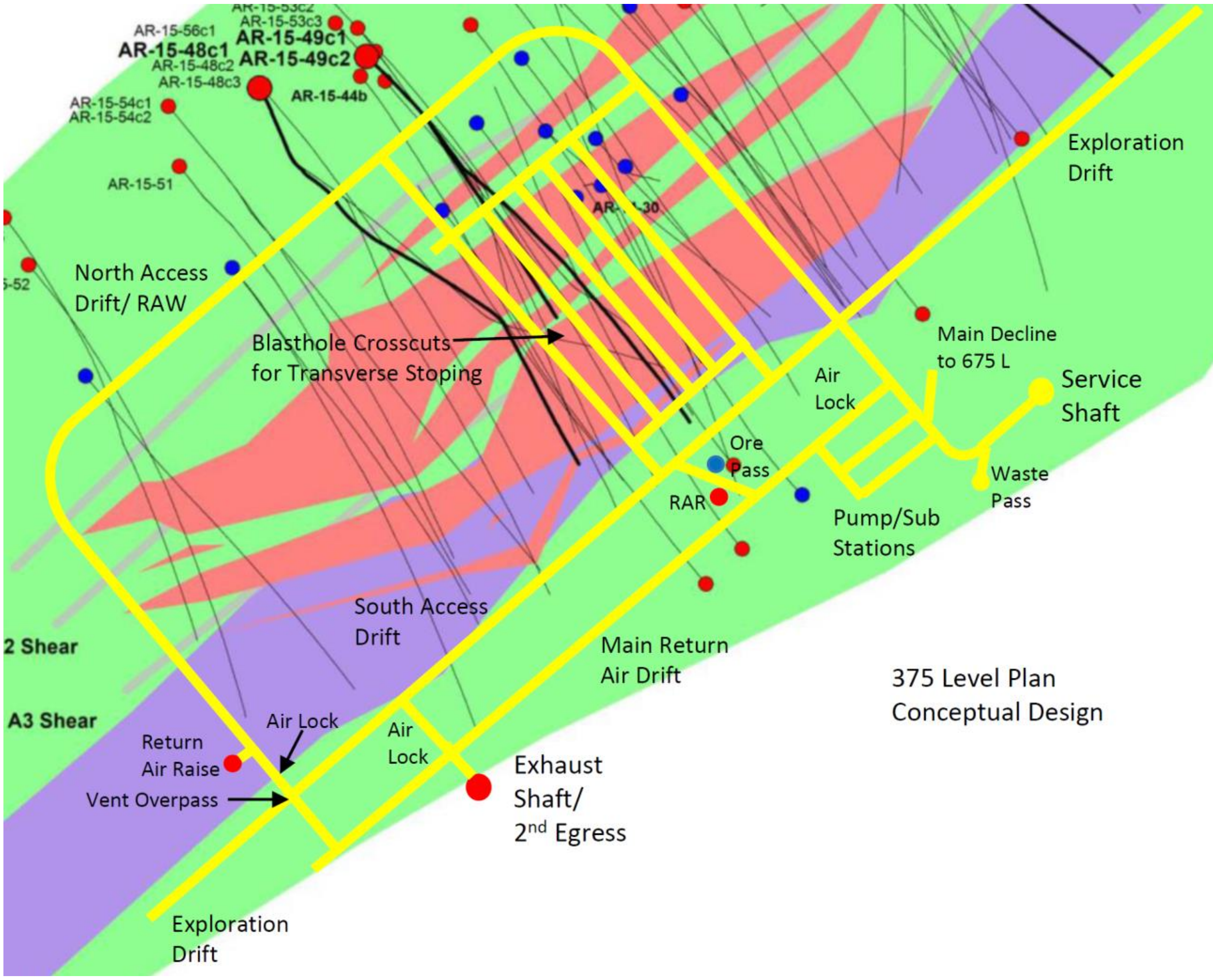

Figure 3 375 Level Conceptual Design for the Arrow Deposit

The basic mine infrastructure plan entails:

1) Sinking a 7 m diameter service shaft to approximately 725 m depth;
2) Sinking a 7 m diameter exhaust shaft to approximately 375 m depth;
3) Establishing main levels from the service shaft on 100 m intervals from the 375 m level downwards;
4) Establishing sublevels on 25 m intervals between main levels and connecting all sublevels via a ramp system;
5) Establishing a loading pocket and loading bins for separate ore and waste streams between the 675 m and 700 m levels;
6) Equipping the service shaft with a cage/counterweight combination, auxiliary cage and balanced 15 tonne skips;
7) Introducing up to 800,000 cfm of air into the mine.

The 375 m level will be utilized for both access and as the exhaust air horizon.

Each main level will be used initially for exploration drilling and then placed into production once all infrastructure is in place and the local mine plan has been finalized.

Ultimately mining below the 675 m level will be conducted by extending the ramp system downwards. Deepening of the shaft may be warranted but daily production volumes will be low enough such that decline mining to 900 m or greater from the 675 m level may be more cost effective.

2.1 Shaft Location Comments

Shafts may be placed on either side of the ore zones. Ground conditions are likely better south of the mineralization, as illustrated in Figure 2, assuming the gabbro has acted as a trap for mineralization and related alteration. However, the thickness of the sand overburden is greater on surface (up to 100 m) and this will impact shaft sinking schedule and costs.

Placing the shafts north of the mineralization, where overburden thickness is less, places the shafts closer to Paterson Lake. A minimum 100 m exclusion zone is required around water bodies so there may be limited locations to spot the shafts. Ground conditions for major infrastructure, such as the ramp system could also be considerably worse. Altered basement rock is typically only one third as strong as unaltered basement rock and this has a substantial impact on ground support requirements and ultimately the project schedule.

For this report the shafts are assumed to be located south of the mineralization as illustrated in Figure 2.

2.2 Development Sequencing

The sequence of mine development events could be as follows:

- Conduct shaft sinking simultaneously for both shafts;
- When the return air shaft reaches the 375 m Level, mine the main access drift and return air drift and connect to the service shaft. Diamond drilling can also be conducted if desired. Commence mining the decline to the 475 m Level. When appropriate schedule wise, cease development and equip the return air shaft for second means of egress and teardown the sinking plant. Build and commission the mine exhaust fans on surface;
- Establish shaft stations at 375 m, 475m, 575m and 675 m levels from the service shaft while sinking. When at the 675 m level, decline to shaft bottom and mine the loading pocket area. Equip the loading pocket and shaft with final guides once shaft bottom breakthrough is made;
- Change over on surface to the final headframe/hoist arrangement;
- Connect up the levels with service shaft area waste passes;
- Commence development on all main levels and the ramp system once waste handling is available. Establish all Level exhaust ventilation to the 375 Level exhaust drift. Commence diamond drilling in areas targeted for initial mining;
- Mine the ore pass system;
- Commence raiseboring between main levels in lower grade ore to prove out procedures;

- Commence raiseboring in high grade ore when lower grade blending ore from development and blasthole stoping is available.

The actual levels and sublevels where initial production commences would require a detailed review of the block model when it is available. However, the high grade core for raiseboring in the A2 shear appears to be between the 475 and 575 Levels.

A rough estimate from commencement of site mobilization for shaft sinking to commencement of production is 4-4.5 years. Collar ground freezing through the sand and sediments is required for both shafts and this adds substantially to the project schedule. Concrete diaphragm walls may be a suitable alternative.

2.3 Site Requirements Prior to Shaft Sinking

The initial site general arrangement for Arrow to support shaft sinking will contain these key capital cost items apart from the shaft area infrastructure:

1) Mine water treatment plant with intake and monitoring ponds;
2) Concrete batch plant, aggregate drying and heated storage;
3) Fresh water intake, storage and distribution;
4) Diesel power generation;
5) Electrical distribution;
6) Propane tank farm(s) and distribution;
7) Construction camp;
8) Temporary offices and mine dry;
9) Cold warehousing;
10) Lined waste pad for mineralized waste rock;
11) Unlined waste pad;
12) Treated water discharge;
13) Off-site pit for aggregate production;
14) Airstrip (or bus from La Loche airstrip).

Some of this infrastructure is temporary only, but if designed correctly, the mine water treatment plant for instance can be permanent infrastructure with modest additional requirements for operations. Infrastructure needs to be well winterized.

3.0 Mining Methods

Blasthole stoping will be used for stopes grading less than 3% U_3O_8. (See separate radiation report for rationale);

Raisebore mining to selectively extract the very high grade core of the ore zones where grades will average 8% U_3O_8. A second pass of stoping is possible adjacent to areas mined by raiseboring at a later stage.

3.1 Blasthole Stoping

Conventional longitudinal and transverse blasthole stoping will be used depending upon orebody width.

Transverse stoping is illustrated in Figure 4. As illustrated in Figure 3, crosscuts through the A2 and A3 shears will be developed on nominal 15 m centres (depending on the outcome of geotechnical investigations). A footwall drift will also be established to allow flow through ventilation. Stope size will depend upon orebody thickness. A waste pillar will be left between the A2 and A3 shears when transverse stoping.

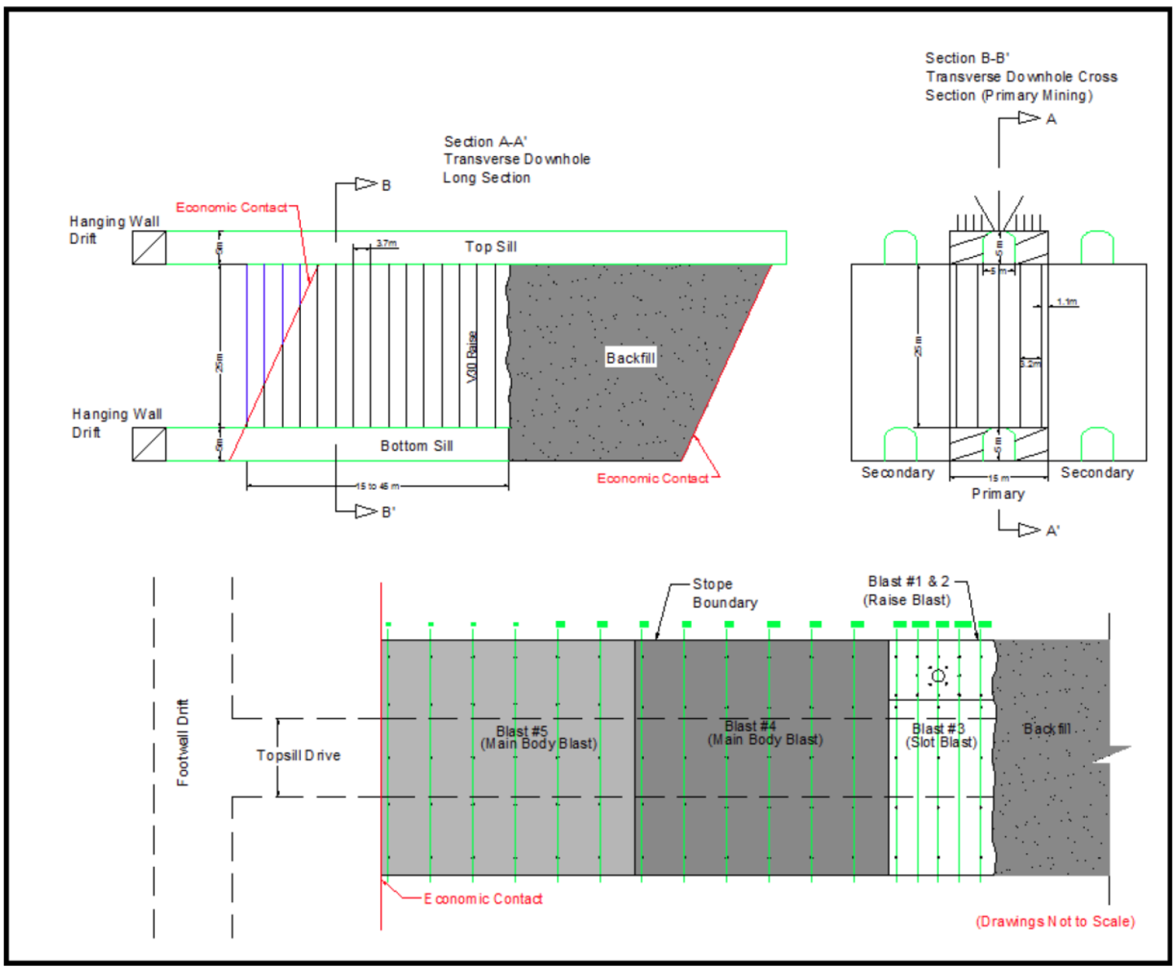

Figure 4 Typical Transverse Open Stoping Method

The stoping sequence will be similar to that illustrated in Figure 5. Cemented paste fill will be utilized to fill mined voids (not rock fill as illustrated in some locations). Sublevel interval is 25 m. For this cost estimate, stopes are assumed to contain 10,800 tonnes each including development tonnes.

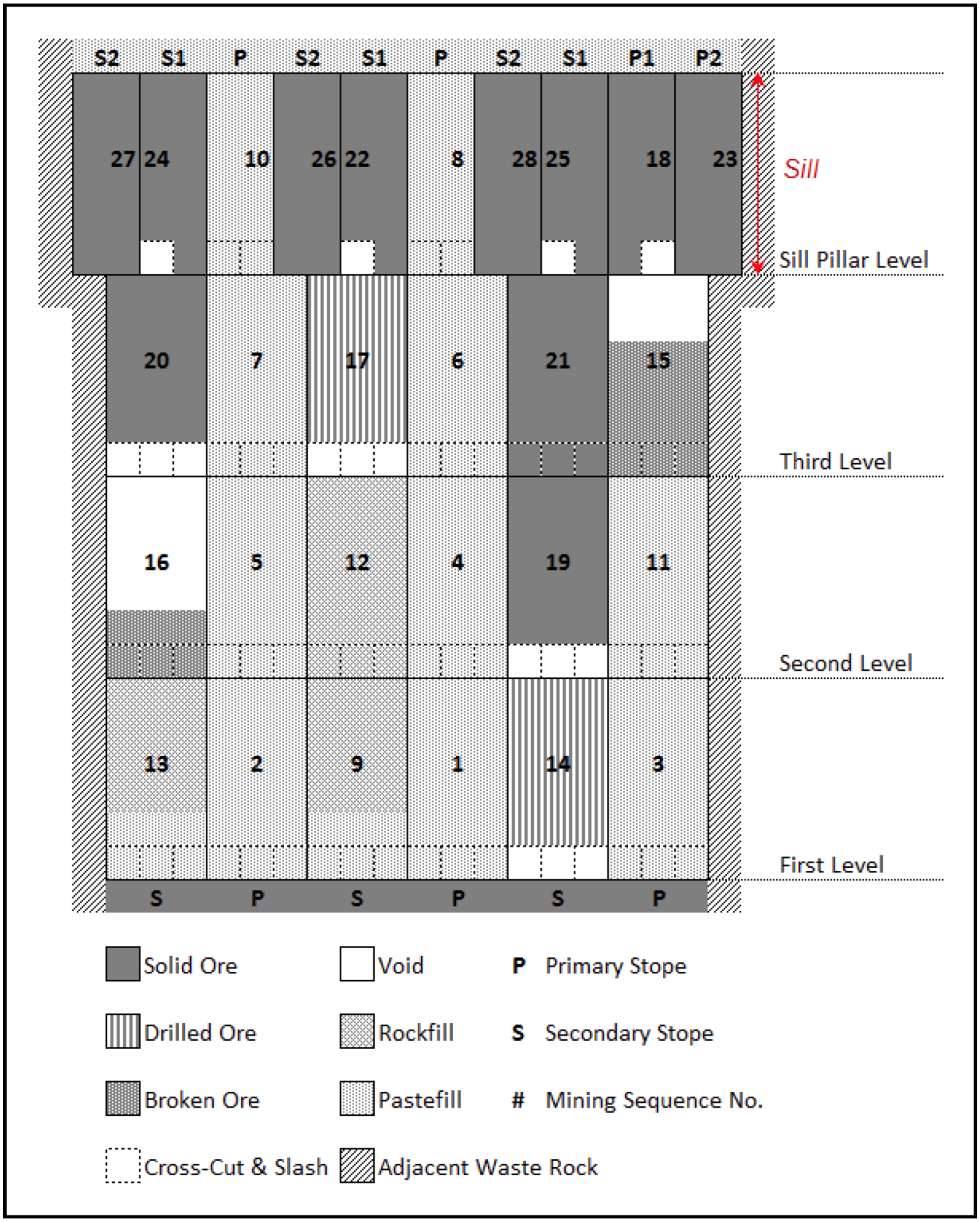

Figure 5 Typical Transverse Stoping Sequence

For narrower portions of the A2 and A3 shears, longitudinal, or sublevel retreat stoping will be used and is similar to that illustrated in Figure 6. The length of stopes will be limited to nominally 25 m before paste filling and re-slotting. Again, geotechnical investigations will confirm these assumptions.

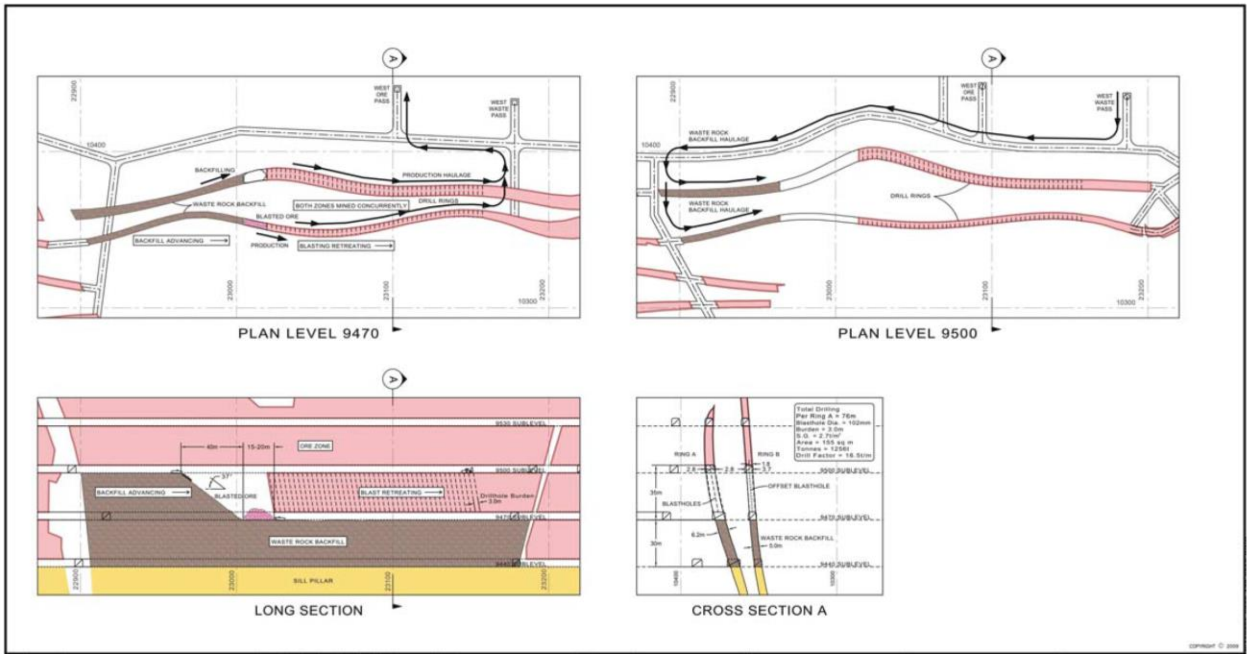

Figure 6 Typical Longitudinal Stoping Method

Photo 1 illustrates a typical drill drift at Eagle Point. For gamma radiation protection, a thick layer of shotcrete is applied to the walls and a concrete floor is poured. The in-the-hole hammer drill illustrated is a reverse circulation drill in order to prevent radioactive cuttings from accumulating on the drift floor. Blast holes are cased with PVC pipe due to their willingness to slough otherwise.

Considerable optimization is possible to further lower worker exposures. Drilling can be largely done remotely for instance.

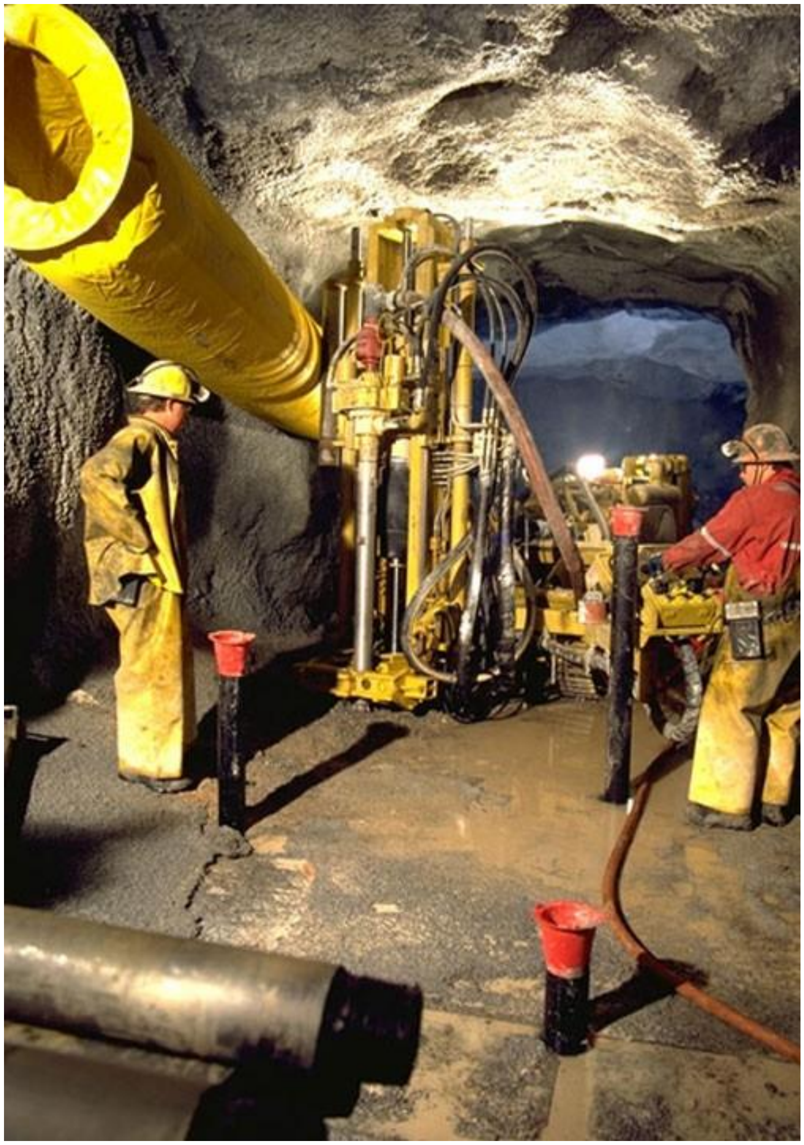

Photo 1 Typical Blasthole Stoping Drill Crosscut at Eagle Point

3.2 Raisebore Mining

The geometry of the shears is ideally suited for raisebore mining. Figure 7 illustrates four possible variations for raisebore mining. The constraints at McArthur River due to the presence of water bearing sandstone are not present at Arrow. Therefore there is much greater flexibility to utilize the method for high grade ore extraction.

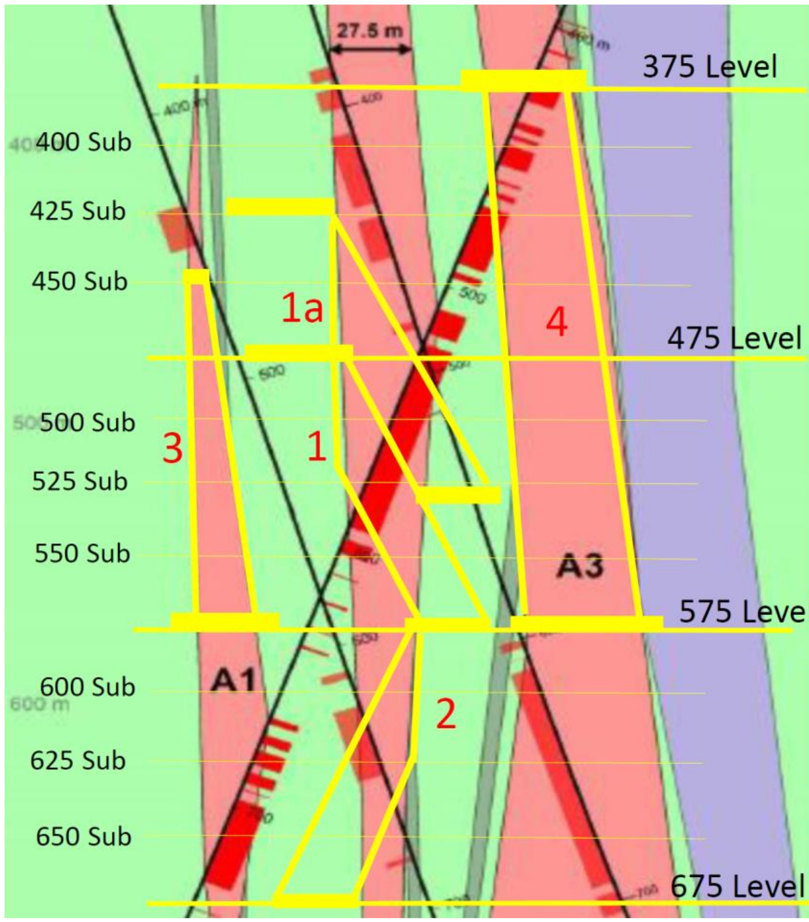

Figure 7 Potential Geometries for Raisebore Mining at the Arrow Deposit

Variation 1: This variation illustrates a raisebore chamber in waste rock north of the A2 shear on the 475 Level and an extraction chamber south of the A2 shear on the 575 Level. Rows of raises are mined from east to west.

Variation 1a: Illustrated is the ability to continue mining up dip (and/or down dip) by establishing additional chambers in waste rock on other horizons. Essentially a mechanized cut and fill method is created where all workers remain outside the very high grade ore zone.

Variation 2: This variation is similar to Method 1 but it illustrates how a previous extraction chamber can be used as a subsequent raisebore chamber by re-excavating the chamber fill as required.

Variation 3: The use of fanned raises is possible. Raises do not need to be parallel. Illustrated is a set up where low grade horizons have been identified above and below very high grade ore which is then extracted without the need for significant waste mining.

Variation 4: “The Holy Grail”. Vertical height is not restricted to that illustrated in previous examples. If the chambers are placed in low grade mineralization with high grade between, it is possible to set up mining areas with metrics significantly better than at McArthur River currently. Raises with a million pounds U_3O_8 or more could be possible without any need for ground freezing. Such a raise could be mined and filled in roughly one month.

As illustrated in Figure 7 there is no need to have the sublevels in place for this method and very long raises are possible. There is no technical reason why raises 200 m long or greater could not be reamed at 3 m diameter. Methods for directional control of pilot holes are available. Hudbay’s Namew Lake mine utilized a variation of this method to mine nickel ore.

The production cycle would be no different to that used currently at McArthur River. In general the raiseboring steps are little different than conventional raiseboring:

1) Pilot hole drill from raisebore chamber to extraction chamber. A full collar blooie arrangement is used to divert cuttings to a collection box or dedicated cuttings borehole. The blooie remains in place during reaming with compressed air introduced into it for raise air changes to avoid the buildup of very high radon progeny concentrations;
2) Following breakthrough, the pilot bit is removed and replaced with a 3.0 m diameter reamer. The reamer is brought into the extraction chamber in the bucket of a large LHD;
3) Reaming commences with cuttings dropping to the concrete floor. Cuttings are removed periodically by a remote controlled LHD and delivered to a radiometric scanner to determine grade. Cuttings are delivered to the location dictated by the respective cut-off grades;
4) Reaming continues to an elevation based on the geological model, radiometric probing through the drill pipe or bucket scanner reading;
5) The reamer is lowered back to the mining level back elevation to provide head cover. A cleanup of the area is conducted using an LHD and fire hose to push remaining ore on the concrete floor beyond the raise opening. The reamer is then lowered to the floor.
6) The LHD then brings the backfill gantry into the chamber and places it under the raise. (The gantry has a horizontal slot in it to fit around the drill pipe). The four legs of the gantry are raised hydraulically remotely to seal against the back. A number of options have been tried to ensure a good seal between the gantry and back;
7) A custom quick attach device is used on the front end of the remote LHD to clamp the drill stem to allow the reamer to be broken away from the drill string. The reamer is then removed;
8) The raisebore retrieves all drill pipe and moves to the next raise location usually a minimum of 7.5 m away and commences the next pilot hole;
9) A concrete pump is brought into the extraction chamber and a 3 m vertical plug of high strength concrete is placed above the backfill gantry in the raise;
10) After a 48 hour cure time, a second pour of lower strength is made in the raise via the pilot hole. The backfill gantry is removed when it is deemed that the initial plug pour is capable of

supporting remaining backfill pours (roughly 4-5 days based on work done at the University of Saskatchewan);

11) Repeat the cycle starting at step 2. It is not necessary to have the previous raise entirely backfilled prior to reaming the next raise though this is desirable to reduce clutter in the raisebore chamber. Backfilling displaces air with extremely high radon progeny concentration so it is critical that negative ventilation ducting be located at the pilot hole collar during backfilling.

Figure 8 is a plan view through Zone 2 mining at McArthur River illustrating the partial overlap of raises for high recovery. Row to row spacing is 2.5 m and collar to collar spacing is also 2.5m. Ore recovery is approximately 95%. Raises dip from 65 to 85 degrees. Pilot holes are approximately 115 m in length depending on dip. Some shallower raises have been mined but pilot hole deviation is a concern. Figure 9 illustrates a cross section through the southern portion of Zone 2.

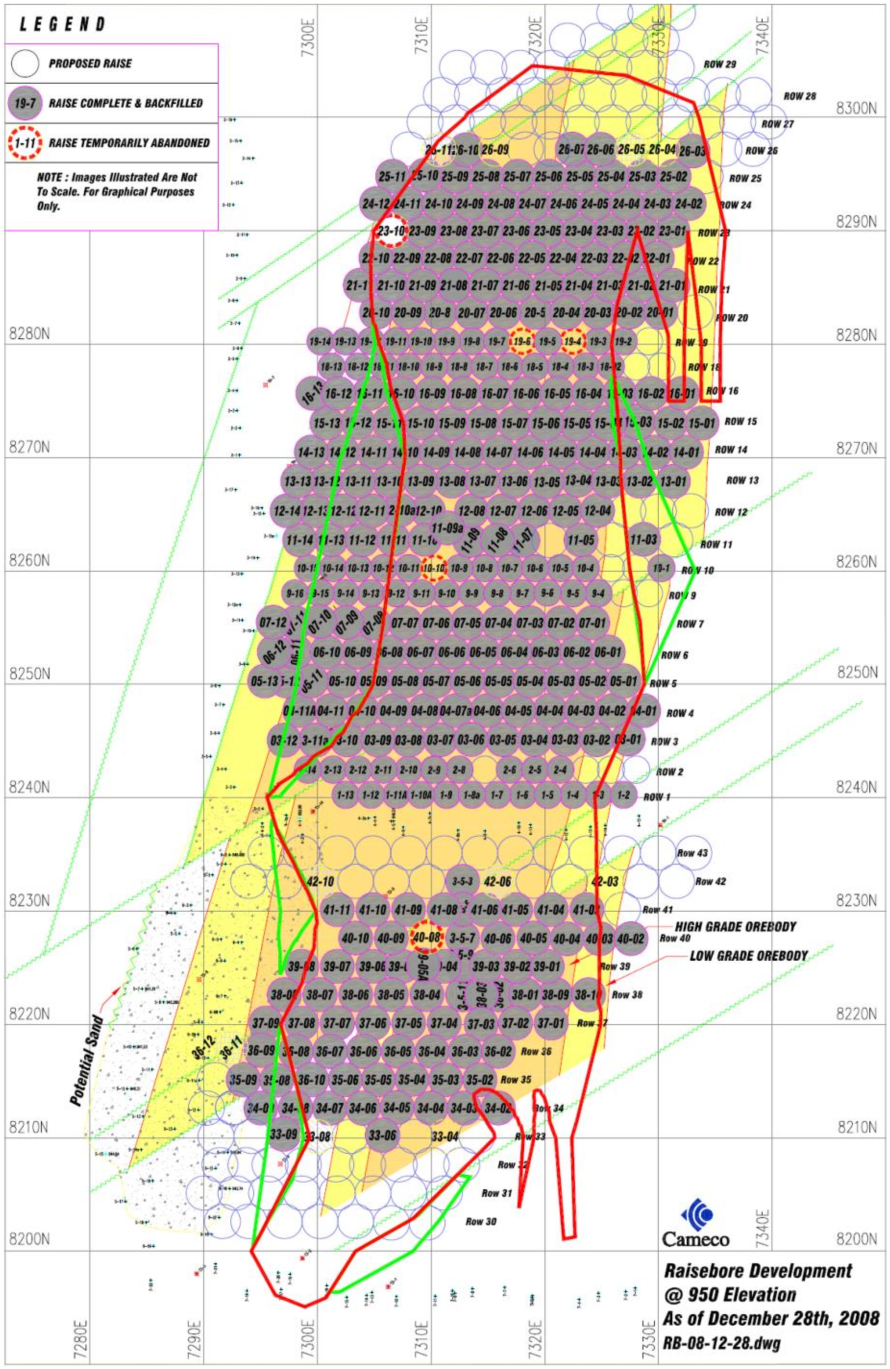

Figure 8 Plan View of Production Raises through Zone 2 at 950 m elevation.

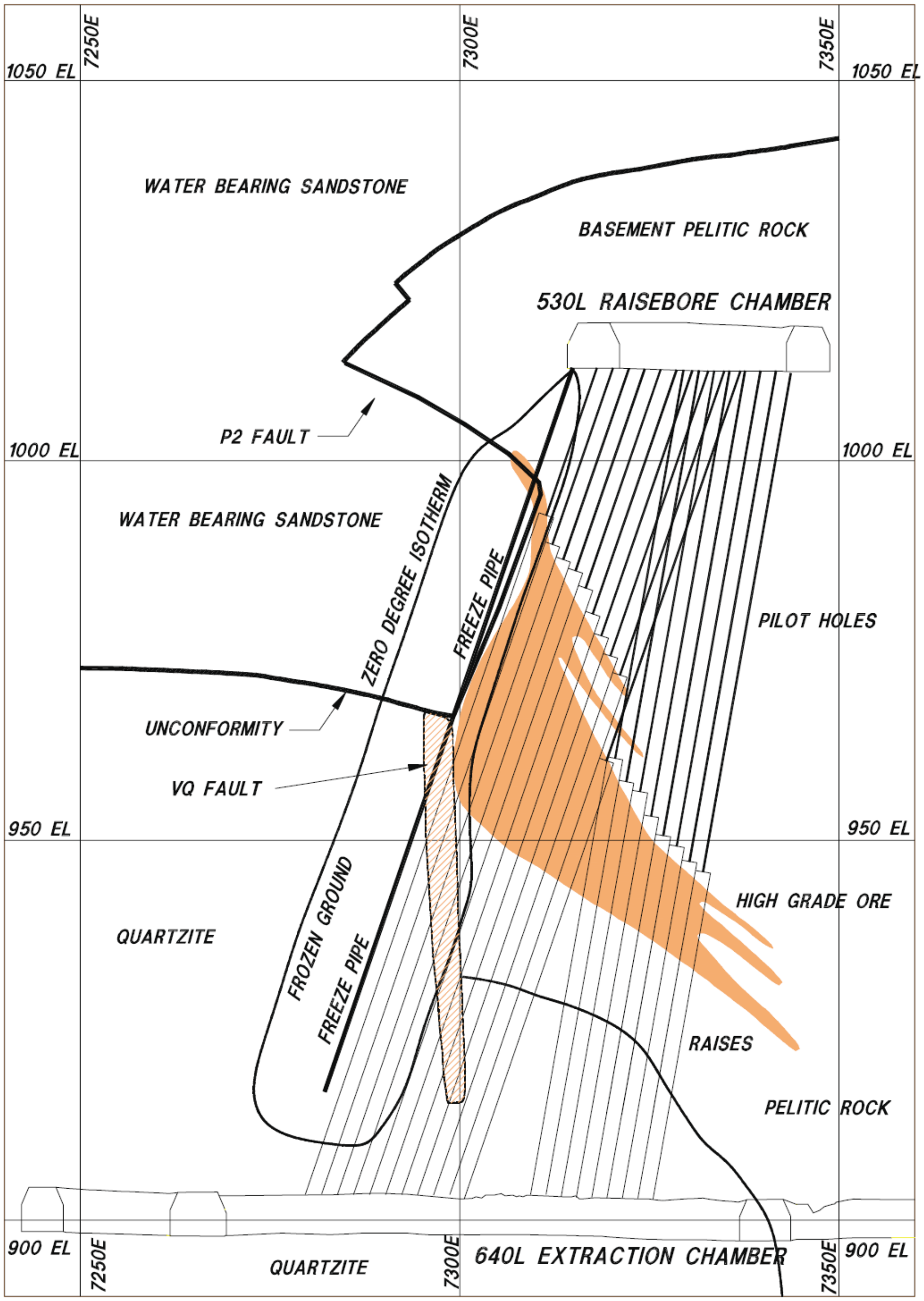

Figure 9 Cross Section Through Panel 3 of Zone 2

A series of photos from Zone 2 mining at McArthur River are provided below. McArthur River commenced reaming at 2.4 m diameter but switched to 3.0 m diameter when the initial raises proved to be very stable.

The raisebore chamber was originally developed for the taller 73 RH raiseborer, therefore it was excessively high for the 53 RH (Photo 2). Due to fair ground conditions, chambers were developed to allow two rows of raises to be mined prior to chamber drift and filling for two additional rows. Initial chambers were 18 m to 30 m below the water bearing unconformity and there was no ground freezing conducted above the original Zone 2 chambers. For reaming the second row of raises the pipe addition arm was relocated to the other side of the raiseborer.

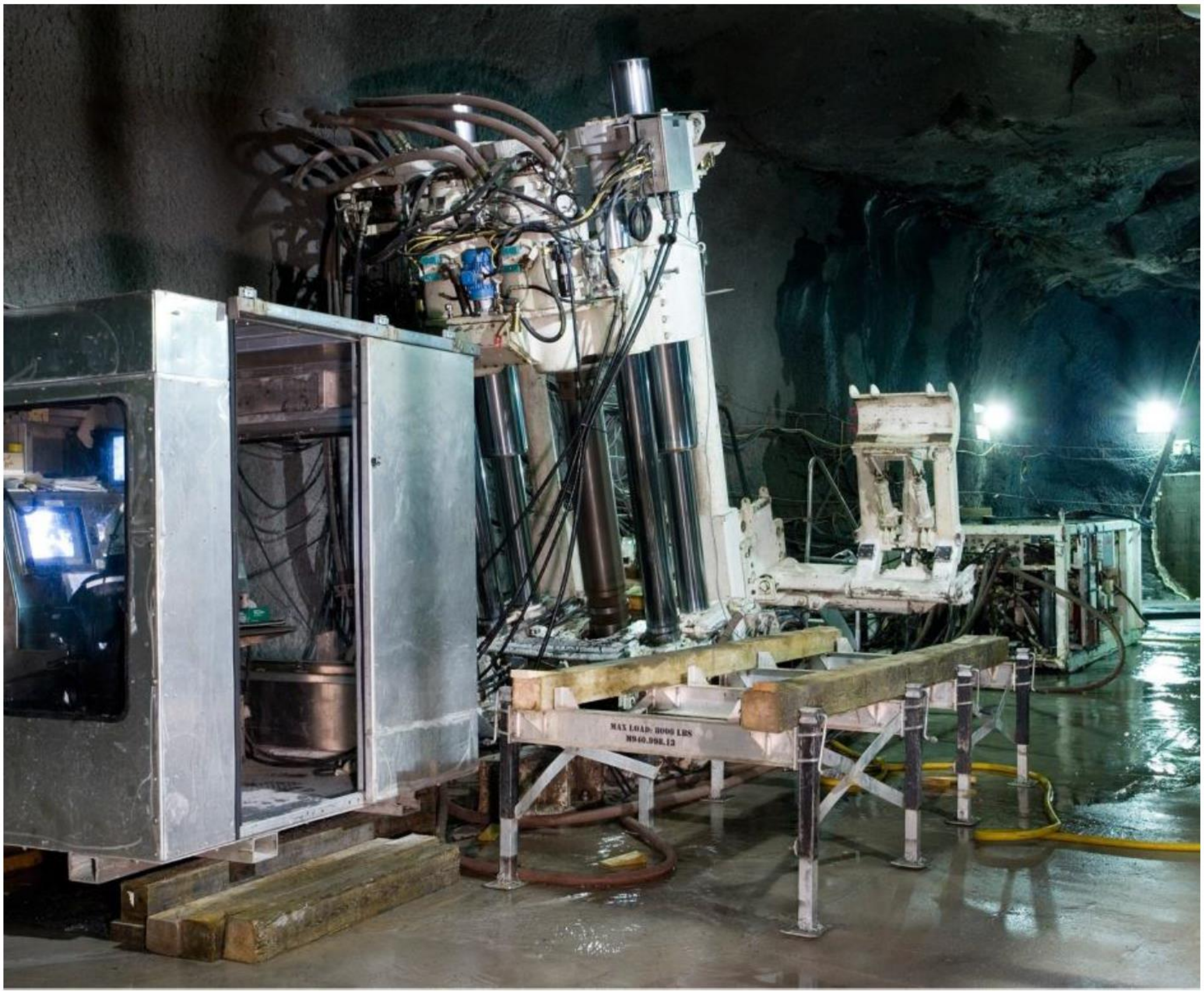

Photo 2 Typical 53 R Raisebore Set up at McArthur River

Photo 3 illustrates the off the shelf 3 m diameter reamer used.

Photo 3 3.0 m Diameter Reamer for Production Raiseboring

The ore collection chute, illustrated in Photo 4, was initially employed at McArthur River to limit the spread of gamma radiation in the extraction chamber. A wet bath dust scrubber is shown to the right. Reaming could only be conducted when the remote controlled LHD was beneath the raise. Cameras were used to indicate when a bucket was near full. After approximately two years of mining, a testing phase was conducted without any of this infrastructure in place in order to demonstrate to the CNSC that worker exposures would remain low (or lower) without it. Conventional remote mucking would be done periodically by scooping muck off the floor. This method change was approved by the CNSC.

Photo 4 Ore Collection Chute below a Production Raise

The operator retrieves the LHD when the bucket is near full as illustrated in Photo 5. At this point, the operator boards the LHD and drives it to the radiometric scanner. Note the backfill gantry in the background in temporary storage. The operator in practice does not stand in this location due to conventional safety concerns. This is more of a public relations photo. An elevated operator's booth is located nearby.

Photo 5 Retrieving the Remote Controlled LHD from the Ore Collection Chute

As illustrated in Photo 6, at the gamma radiation scanner the bucket is elevated to the height of the timber on chains and a gamma reading is taken to determine if this is material for skipping (<2 % U_3O_8) or high grade ore for underground processing (>2 % U_3O_8). On occasion, ore grades on a given shift exceeded 50% U_3O_8 but these grades did not represent a radiation concern due to the shielding provided between ore and LHD operator.

Photo 6 Production Raisebore Cuttings Being Scanned for Ore Grade

Photo 7 illustrates a radon progeny monitor indicating that Working Levels in this upstream area are satisfactory. A raise ready for backfilling is shown in the background. The reamer is likely parked 2-3 m above the back in the raise for head cover. The radioactive spillage has been pushed by an LHD and fire hose beyond the raise opening to the return air side. Note the concrete floor. A radiation technician is taking readings (radon progeny and gamma) to ensure it is acceptable to bring the backfill gantry into the extraction chamber for reamer removal and subsequent raise backfilling. A portable radon progeny monitor would also be installed near the raise. A remote controlled rubber rollup door used to regulate the ventilation flow is illustrated at the end of the chamber. This allows for access to the exhaust drift beyond to allow periodic cleanup by LHD of spillage that has accumulated. The exhaust ducting illustrated is no longer required since it was related to the ore collection chute no longer necessary.

Photo 7 A Radon Progeny Monitor in a Typical Raisebore Extraction Chamber

Upon completion of two rows of raises, formwork is erected in the raisebore chamber to allow the filling of a 5.0 m wide length of this chamber. It is vital to get a tight seal with the back to minimize future ground relaxation. This is done by using more than one fill pipe placed in the highest points of the pour. If necessary, one or more small notches should be mined in the back to locate the outlet of these pipes to ensure a tight seal with the back by gravity. Concrete is placed by standard adjacent cement pump. Concrete strength of 20 MPa is adequate. After pour completion a series of jackleg holes can be angled through the top of the concrete to cross the concrete/rock interface. Cement grout is injected into this interface under low pressure to fill remaining voids. The chamber is then slashed 5.0 m wider using a drill jumbo.

The same drift and fill approach is used in the extraction chamber.

Wear on the reamers at Arrow should be extremely low due to the nonabrasive nature of the ore zone. McArthur River on the other hand reams through quartzite and silicified sandstone for a portion of most raises mined. These are highly abrasive rock types.

4.0 Annual Production Rate

The annual production rate modelled is 10 million pounds per annum U_3O_8 recovered from the mill. Daily mining rate will approximate 500 tpd and will be composed of 420 tpd from blasthole stoping and 80 tpd from raiseboring as illustrated in Table 1.

Table 1 Conceptual Annual Production Schedule for Cost Estimation Purposes

Mining Method	Tonnes	Grade	Million Lbs. U_3O_8
Raiseboring	30,000	8%	5.3
Blasthole Stoping	150,000	1.5%	5.0
Total	**180,000**	**2.6%**	**10.3**
Mill recovery			98.5%
Packaged U_3O_8			**10.1**

5.0 Support Services

5.1 Mine Ventilation

The 7 m diameter shafts will allow for an initial ventilation capacity of 800,000 cfm with the restriction being the velocity of air in the service shaft. Mine air heaters will be located at the service shaft to heat the air then delivered through a subcollar plenum. The ventilation system will be set up to be a pull system. In other words, the exhaust shaft fans will be high negative pressure fans and the intake fans will be low positive pressure fans. Historical propane consumption factors for the underground mines in northern Saskatchewan are 12 litres of propane per annual cfm. Therefore, 9,600,000 litres of propane are estimated per year for mine air heating. Intake air is heated to 4C.

An upgrade later in the life of the mine to 1,200,000 cfm is possible with the addition of a second intake shaft. This shaft can either go to the 375 Level or be as short as 200 m in length provided it is then connected to the mine workings via bored or Alimak raises. Velocity in the exhaust shaft would be increased to accommodate this flow rate.

5.2 Shaft Equipping

The service shaft will be equipped with a main cage balanced by counterweight, two 15 tonne skips in balance and a 5 person auxiliary cage. Therefore, it is necessary to procure two main hoists and an auxiliary hoist. The main hoists will likely be in the 4-4.5 m diameter range.

Fixed steel buntons with wood guides would be installed. The skip compartments would be bratticed off from the main cage compartment.

The exhaust shaft would have an auxiliary cage with adjacent ladderway. An air lock is necessary at surface so this cage would be loaded and unloaded at a subcollar plenum.

5.3 Mine Dewatering

Very little groundwater should be expected due to the distance of mining from the unconformity. The main pump station will be established on the 375 Level and flows from levels below will be pumped to this horizon. A dewatering rate of 50 m^3/h (roughly 200 gpm) is assumed for steady state dewatering and surface water treatment. Pumping capacity of 400 m^3/h however will be installed to handle emergency situations such as intersecting ungrouted diamond drill holes connected to the surface groundwater table.

5.4 Backfilling/Batch Plant

A surface plant capable of making paste fill, shotcrete and concrete is required. The plant products will be pumped by positive displacement pump to either dedicated lined boreholes or shaft slick lines leading to the 375m level. Paste fill will be distributed to stopes requiring filling via a network of fill pipes. Shotcrete and concrete will be delivered to a variety of mobile equipment (transmixer, LHD bucket or truck) or concrete pump depending upon the application.

5.5 Site Power

The existing power lines from Meadow Lake to La Loche (Photo 8) are low voltage (likely 10-20 kVA) and are not deemed by the author to be sufficient for industrial needs related to a large mine/mill complex. This requires confirmation with SaskPower. On site diesel generation is assumed for the mine life. The financial incentive appears to exist ($0.39/kwh vs. $0.06/kwh) to negotiate with the province for the installation of a high voltage line from Meadow Lake assuming there is sufficient spare capacity in this part of the province. Meadow Lake has a 44 MW natural gas fired generating station. Natural gas availability in the Meadow Lake area is excellent since a major transmission pipe line is nearby. LNG may therefore be an option. For a full mine/mill complex, savings of approximately $20 M annually are possible by switching from diesel to grid power.

Photo 8 Power lines south of La Loche

5.6 Ore Handling

A key aspect of the mine plan proposed is to avoid the need for elaborate ore handling underground. No grinding of ore and hydraulic hoisting to a dedicated slurry handling building is proposed. Ore will be skipped up the shaft. This should reduce capital costs by $75-$100M.

An ore pass system will be bored from the 675 Level to the 375 Level. A system of return air raises will parallel the ore pass system nearby. Both raise systems are connected to the 375 Level return air drift and exhaust shaft. Short finger raises on each level and sublevel will connect to the ore pass raises. Each finger raise is a dump point and it will include a hydraulically activated ore pass cover. The ore pass system will be under negative ventilation such that radioactive dust does not accumulate or spread through the level or sublevel when ore is dumped down the pass.

The ore pass system is used exclusively for development ore and blasthole stoping ore. On the 675 Level, the ore will collect at the bottom of the ore pass system and be removed one bucket at a time by the skip tender. This ore will be scanned and then placed in one of three adjacent stock piles; low grade, medium grade and high grade.

Ore from high grade raiseboring will be collected at the bottom of a production raise by remote LHD. It will then be loaded on to a 26 T truck and hauled via the decline system to a 675 Level laydown area adjacent to the loading pocket area. The skip tender will then muck, scan and place this material in a high grade or low grade stockpile dedicated strictly for raisebore cuttings.

Since the average daily production rate for raiseboring is only 80T it is apparent that only a handful of truck trips per day are required. The truck itself could readily be designed with further radiation protection measures such as a hydraulically activated truck bed cover.

For shaft skipping, the skip tender will generally place one bucket of high grade raisebore cuttings for every four or five buckets of development or blasthole muck into the loading pocket ore bin to both blend the grade down and to make the raisebore cuttings more skippable. There will be a separate waste bin.

One 15 T skip will be dedicated to ore skipping and one 15 T skip will be dedicated to waste skipping in order to keep the streams separate. Likewise, on surface, there are a number of options to dump the ore. The simplest method is to dump the ore into a concrete lined bunker (similar to that used for shaft sinking) where the ore is then removed by remote controlled front end loader to a nearby surface stockpile. A separate bunker is used for waste rock. More complex designs (binhouses etc.) will likely come with more complex radiological issues also. The potential to greatly over engineer the solution to a problem such as skip dumping is always present in the northern Saskatchewan uranium mines and local engineering firms will be glad to do so for you.

Periodic gamma radiation surveys will be conducted in the skip compartments to ensure no buildup of gamma radiation is occurring on the sets. A broom is all that is necessary to clean up any buildup during regular shaft inspections. The skip compartments will be bratticed from the main cage compartment. The skip compartments will downcast similar to the main cage compartment. It is not necessary, from a radiological perspective, to exhaust ventilate the skip compartments since the ore generates very little radon gas during transport to surface and, with proper skip design, there is little or no turbulence in the skip itself that could generate radioactive dust into the shaft. McArthur River has amply demonstrated this.

6.0 Conceptual Design Level Mine Operating Cost Estimate

To get a basic understanding of the mine operating costs it is necessary to understand the operating philosophy. Existing mines have been a split between company and contract workers in an often awkward arrangement. For instance, company staff do the production raisebore mining at McArthur River whereas contractors do the mine development. At Eagle Point, contractors do both blasthole stoping and mine development while the company provides some supervision.

The model derived in this report utilizes:

1) Company personnel for production raiseboring. This includes maintaining related production equipment;
2) A mining contractor for mine development and blasthole stoping. All mobile equipment is supplied by the owner but maintained by the contractor. Therefore, equipment rental rates are not included in the model;
3) Contractors for diamond drilling, camp catering/site janitorial and site security services.
4) Company personnel for all other aspects.

This is not necessarily the recommended approach but it is adequate for cost modelling purposes. Responsibilities are summarized in Table 2.

Table 2 Responsibility Check List

Area	Responsibility
Production Raisebore Mining (including maintenance)	Company
U/G fixed plant maintenance	Company
Shaft Operations	Company
Admin/Eng./Geol.	Company
Surface Services	Company
Radiation/Environment/Safety	Company
Mine Water Treatment	Company
Surface Maintenance	Company
Batch/Backfill plant	Company
Mine Development (including maintenance)	Contractor
Blasthole Stoping (including maintenance)	Contractor
Diamond Drilling	Contractor
Camp catering/janitorial	Contractor
Security	Contractor

The mining contractor is essentially providing a labour contract in this cost model.

For mining contractor accounts, labour costs have generally not been split out by account. Unit rates are used instead. A labour cost reconciliation is then conducted for all mining contractor accounts in order to compare costs to current industry labour charge out rates.

Maintenance costs for the equipment the contractor is using and maintaining, are itemized in a separate account (210) and then allocated on a percentage basis to all stoping and development accounts. Maintenance costs include labour, parts and consumables.

Likewise, company maintenance costs are itemized in one account (501) and then allocated across many different surface and underground accounts.

Material costs for stope backfill, production raise lean concrete, drift and filling, roadbed concrete and development shotcrete are reflected in related accounts. The surface backfill/batch plant costs are strictly for plant operation and maintenance.

It is necessary to understand the amount of mine development required per year to open up areas for mining in sufficient time for production.

Mine development is subdivided into three categories:

1) Mine development in or near mineralization for blasthole stoping;
2) Mine development including chamber drift and filling for production raiseboring;
3) Mine development to establish new sublevels and mining areas sufficient to provide ample future production sources. This also includes exploration development on strike, ramps and declines and raises for ventilation and passes. This area could be referred to as sustaining capital development but is included in this operating cost model.

Rough calculations performed by the author suggest:

1) 800 m of mine development for blasthole stoping is required annually;
2) 3,000 m^3 of drift and filling for existing raisebore and extraction chambers is sufficient to maintain these production sources. 200 m of access development annually is adequate to access new production raisebore areas from existing lateral development;
3) 1,500 m of lateral development and 200 m of raise development is adequate annually to open up new areas of the mine for future production.

Production metrics for both production mining methods are summarized in Table 3 and 4.

Table 3 Production Raisebore Metrics Modelled

Raise Metrics	Qty.	Units
Pilot Hole Length	100	m
Raise Length	80	m
Ore	65	m
Waste	15	m
Diameter	3	m
Average Ore Grade	8%	U_3O_8
Specific Gravity-Ore	2.8	
m^3 reamed/m raise	7.1	m^3/m
Tonnes ore/raise	1,286	t
Pounds U_3O_8/raise	226,882	lb U_3O_8
Annual Metrics	**Qty.**	**Units**
Active Raiseborers	2	
Raises per year	23.3	
Tonnes ore/year	30,000	t
Pounds U_3O_8/year	5,290,920	lb U_3O_8
Pilot holes/year	2,332	m
Ore reaming/year	1,516	m
Waste reaming/year	345	m
Waste reaming/year	6,676	t

Table 4 Blasthole Stoping Metrics Modelled

Stope Metrics	Qty.	Units
Tonnes per Stope	10,800	t
Average Ore Grade	1.5%	U_3O_8
Pounds U_3O_8/stope	357,129	lb U_3O_8
Annual Metrics	**Qty.**	**Units**
Stopes/year	14	
Tonnes/year	150,000	t
Pounds U_3O_8/year	4,999,806	lb U_3O_8
Blastholes	21,600	m
Slot raises	280	m
Cable bolting	11,200	m
Paste Fill	53,571	m^3

A rough calculation performed by the author indicates that 35,000 m of underground diamond drilling annually for the first 5-8 years should be sufficient to outline at least 15 years of ore for production. Zones would be defined on a 15 m x 15 m toe spacing.

6.1 Manpower

A total of 287 persons are included in the cost model. Table 5 lists the breakout.

Table 5 Manpower Included in the Cost Model

Salaried Employees	
Mine Administration, Engineering, Geology	22
Site Administration	6
Environment and Safety Services (incl. Radiation)	17
Site General Services	4
Mine Maintenance	14
Subtotal	**59**
Hourly Employees	
Drymen	2
Shaft Operations	14
Backfill Plant	8
Production Raiseboring	20
Water Treatment	4
Warehousing	4
General Site Services	14
Electrical Supply (powerhouse)	4
Mine Maintenance	36
Subtotal	**104**
Contract Employees	
Mining Contractor (development and stoping)	84
Diamond Drilling	14
Catering/Janitorial	22
Site Security	4
Subtotal	**124**
TOTAL	**287**

The average fully burdened cost per employee is $110,500. Staff and contractors work a 7 day in/7 day out schedule apart from 17 individuals who work a 4/3 schedule. Therefore only roughly half of those listed above are on site at any given time. Twelve hour shifts are modelled.

6.2 Cost Model Summary

The cost model assumes that the mine has reached a steady state annual production rate of 180,000 T of ore grading 2.6% U_3O_8. The year 2022 was used, although costs should be considered to be in 2015 dollars. All model details and calculations are provided with this report in an Excel spreadsheet.

No milling costs, nor site support services for a mill, are included in the model. Milling comments are made in Section 6.3. The Level 1 summary of costs from the model are illustrated in Table 6.

Table 6 Level 1 Summary of Arrow Operating Costs in Canadian Dollars

Area	Description	Annual Cost
2xx	Underground Mining	$ 52,776,187
3xx	Water Treatment	$ 2,154,910
4xx	Site Administration, Fuels and Utilities	$ 27,328,445
	Subtotal	**$ 82,259,542**
1xx	Offsite Overheads @6%	$ 4,935,572
	Total	**$ 87,195,114**
	Cost/ lb U_3O_8 recovered	**$ 8.63**

Offsite costs include numerous items such as warehousing and marshalling in Saskatoon, accounting and other corporate services. Not included are costs related to marketing and transport of yellowcake to conversion facilities. Note that for a single asset company offsite costs will be considerably greater than illustrated since all head office, marketing and finance costs will likely be assigned to the single mine/mill complex. Paladin Resources is a good example of a head office consuming any potential mine site free cash flow.

Table 7 illustrates the Level 2 breakout of operating costs at the Arrow site only.

Table 7 Level 2 Operating Costs in Canadian Dollars

Area	Description	Annual Cost
201	Mine Administration General, Geology and Engineering	$ 2,917,411
204	Building Operation	$ 522,033
212	Shaft Operations	$ 2,751,332
216	Backfill/Batch Plant	$ 1,787,710
223	Underground Mine Infrastructure	$ 1,733,165
240	Mine Vertical Development	$ 783,000
241	Mine Lateral Development	$ 23,274,900
242	Production Raiseboring	$ 8,577,932
245	Production Slot Raises	$ 560,000
247	Cable Bolting	$ 1,193,200
251	Production Drilling and Blasting	$ 1,862,000
253	Stope Mucking	$ 1,132,800
254	Stope Backfilling	$ 2,005,704
270	Underground Diamond Drilling	$ 3,675,000
2xx	**Underground Mining Subtotal**	**$ 52,776,187**
3xx	**Water Treatment**	**$ 2,154,910**
401	Site Administration	$ 3,986,922
411	Commuting	$ 4,815,106
412	Camp Operation	$ 3,751,008
421	Environment and Safety Services	$ 3,681,111
441	Warehousing and Freight	$ 1,087,422
461	Site General Services	$ 3,337,165
470	Diesel, Propane and Gasoline	$ 6,669,711
471	Electrical Distribution (including diesel)	$ 14,323,332
4xx	**Site Administration, Fuels and Utilities**	**$ 27,328,445**
	Total Site Costs	**$ 82,259,542**

The unit cost assumptions for propane and diesel are important:

Propane	$0.50 /litre
Diesel	$1.30/litre

Costs include freight to site.

The annual power demand for the mine and site is estimated at 36.3 million kWh (see cost centre 470) consuming 9.4 million litres of diesel which equates to $0.35/ kWh. Additional costs to operate and service the generators on site bring total cost to $0.39/ kWh.

The current provincial industrial rate for power is roughly $0.06/ kWh. Savings of approximately $10 M are possible annually by using grid power for the areas above. Conceivably, similar savings would be possible for the mill and related services.

This was the rationale for the installation of the power line to Key Lake and McArthur River in the early 1990's (when diesel was $0.50/litre). Spare power existed at the Island Falls hydroelectric station north of Flin Flon. Both Cameco and the province therefore benefited by displacing third party diesel fuel.

6.3 Milling Cost Comments

The author has no expertise in the area of uranium milling and therefore has not incorporated this area into the cost model. The author can recommend an expert in this area if desired.

Both McArthur River and Cigar Lake have filed NI 43-101 reports in the past three years with annual milling operating cost estimates included. These would be presumably based on recent experience hence are likely far superior than included in a typical PEA for instance. Both Key Lake and McClean Lake mills benefit greatly from economies of scale at full (anticipated) mill production rates of 17-18 million pounds U_3O_8 per year.

The 2012 NI 43-101 report for McArthur River/Key Lake states that mill site costs in 2015 will be:

Administration	$60.2 M
Milling	$84.5 M
Subtotal	$144.7 M
Corporate Overheads	$9.7M
Total	$154.4 M

Packaged product is 18.7 million pounds U_3O_8. This leads to an overall cost of $C8.25 /lb and site cost of $C7.74/lb. This represents a cost escalation of roughly 70% since 2005 presumably due to the ill effects western Canada has experienced related to the oil sands boom.

The 2012 NI 43-101 report for Cigar Lake, with milling at McClean Lake, states that milling costs at McClean Lake will be approximately $135 M per year when full annual production of 17.6 million pounds U_3O_8 is reached (2018 data was used). This translates into $C7.70/lb site costs. Cameco may have merely applied Key Lake's cost structure to McClean Lake for the report.

A fixed and variable cost model would likely be necessary to estimate mill costs at Arrow but indications are it would likely be in the $C10-11/lb range particularly if diesel generated power is used.

It would be legitimate to add this cost to the derived mining cost to come up with an overall unit cost. Some mine/mill synergies exist to reduce costs (warehousing for instance) but the Key Lake budget above also likely subtracts some back charges to McArthur River for services provided by the well-equipped shops and laboratories at Key Lake.