Mine Development and Operations Plan

Quartz Mining Licence QML-0009

December 2009

BELLEKENO PROJECT

Mining Study Report
Bellekeno Ag-Pb-Zn Project
Yukon Territory

Report Prepared for

Alexco Resource Corp

Report Prepared by

SRK Consulting
Engineers and Scientists

October 14, 2009

Mining Study Report

Bellekeno Gold Project

Yukon Territory

Alexco Resource Corp

Suite 1150 - 200 Granville Street
Vancouver, BC, Canada
V6C 1S4

SRK Consulting (Canada) Inc.
Suite 2100 – 25 Adelaide Street East
Toronto, Ontario M5C 3A1
Tel: 416.601.1445 Fax: 416.601.9046
E-mail: toronto@srk.com Web site: www.srk.com

SRK Project Number 3CA020.002

October 14, 2009

Ken Reipas

Prepared by

K. Reipas, P.Eng. - Principal Mine Engineer

Reviewed by

J.F. Couture - Principal Geologist

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CAUTIONARY STATEMENT

This mining study, prepared in support of a preliminary economic assessment technical report, is not adequate to allow the estimated mineral resources to be categorized as mineral reserves. A feasibility study or pre-feasibility study is required for that purpose.

1 Introduction

1.1 Terms of Reference

SRK Consulting (Canada) Ltd. (“SRK”) was commissioned by Alexco Resource Corp. (“Alexco”) in November 2008 to prepare the mining related sections of a Preliminary Economic Assessment (“PEA”) technical report on the Bellekeno underground silver-lead-zinc project, located in northern Yukon.

The current scope of work is intended to update SRK’s previous technical report, “Bellekeno Preliminary Economic Assessment,” prepared for Alexco and dated June 2008. This update is based on the results of an underground development program and rehabilitation of historical working for an underground in-fill drilling and pre-production development program completed since the previous study.

SRK started work on the current commission in mid-December 2008.

1.2 Qualifications of Consultants

SRK is an independent, international consulting company providing focused advice and problem solving. SRK provides specialist services to mining and exploration companies for the entire life cycle of a mining project, from exploration through to mine closure. Among SRK's 1500 clients are most of the world’s major and medium-sized metal and industrial mineral mining houses, exploration companies, banks, petroleum exploration companies, agribusiness companies, construction firms and government departments.

Formed in Johannesburg, South Africa, in 1974 as Steffen, Robertson and Kirsten, SRK now employs more than 750 professionals internationally in 30 permanent offices on six continents. A broad range of internationally recognized associate consultants complements the core staff.

SRK employs leading specialists in each field of science and engineering related to the minerals sector. Its seamless integration of services and global base have both made the company the world's leading practice in due diligence, feasibility studies and confidential internal reviews.

The SRK Group’s independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues.

This Mining Study Report was prepared by Ken Reipas, P.Eng (APEO) with the assistance of Mr. Bruce Murphy and Mr. Ross Greenwood for the geotechnical assessment.

Mr. Ken Reipas is a Principal Mine Engineer and has been employed by SRK since 2001 and has over 28 years experience in mine engineering, mine production and consulting. Prior to joining SRK, he worked at several open pit and underground mining operations in Canada involved in the bulk mining of iron, coal, gold and base metals. Positions held included Chief Engineer and Mine Superintendent. Since 1997, his consulting projects have included technical studies and reports, mine planning and reserves, project economics, and due diligence reviews involving northern projects, mine rehabilitation, mine reopening, and care & maintenance.

Mr. Reipas is the principal author of this report.

Mr. Bruce Murphy, Principal Consultant, Rock Mechanics, with the assistance of Mr. Greenwood, prepared section 4.1 “Mine Geotechnical” of this report.

Mr. Ross Greenwood, P.Geo, is a Senior Consultant in rock mechanics.

By virtue of his education, relevant work experience and affiliation to a recognized professional association Mr. Reipas is an independent qualified person as this term is defined by National Instrument 43-101.

This report benefited from the review of Dr. Jean-Francois Couture, P.Geo.

1.3 Basis of the Report

This Mining Study Report is based on the following sources of information:

- SRK’s previous technical report (June 2008) referenced above;
- Mineral resource block models and wireframes of the Bellekeno 48 vein and 49 vein provided by Alexco and based on the recent drilling campaign, underground mapping and resampling of exposed vein material;
- The results of geotechnical studies undertaken by SRK, based on the recent drilling and underground geotechnical evaluation of new and historic areas;
- 3D wireframes of existing Bellekeno mine workings that include recently resurveyed areas;
- Certain site specific cost information provided by Alexco including summary level results of mining contractor quotations;
- Net smelter return (“NSR”) models provided by Alexco and reviewed by SRK;
- Discussions held with Alexco management and technical staff;
- Site visits to the project site, including extensive inspections of the underground mine.

1.4 Site Visit

The following SRK consultants made multiple visits the Bellekeno project site in northern Yukon. The most recent visits include:

Ken Reipas, Principal Mining Engineer, July 14-18, 2009
Bruce Murphy, Principal Consultant, Rock Mechanics, December 14-17, 2008
David Keller, Principal Resource Geologist, August 5-7, 2009
Ross Greenwood, Consultant, Rock Mechanics, June 23 to July 7, 2009.

1.5 Terminology and Definitions

Metric units of measure, including metric tonnes, are used in this report. Currency amounts are expressed in Canadian dollars unless otherwise stated (such as for quoting metal prices).

2 Declaration and Reliance on Other Experts

SRK's opinion contained herein and effective October 13, 2009, is based on information provided to SRK during the course of SRK's investigations as described in report section 2.3. This, in turn reflects the various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time.

SRK is not an insider, associate or an affiliate of Alexco Resource Corp, or of its affiliates in connection with the Bellekeno project. The results of this technical study by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

Technical economic projections include forward-looking statements that are not historical facts. These forward-looking statements are estimates and involve a number of risks and uncertainties that could cause actual results to differ materially.

This report includes technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK relied on the work of Alexco who prepared the mineral resource block model on which the underground mine plan is based.

3 Mineral Resource and Mineral Reserve Estimates

3.1 Mineral Resource Statement

Table 1 provides a summary by zone of the publically disclosed Classified Mineral Resources for the Bellekeno project.

The resource estimate was prepared by Mr. Stan Dodd, P.Geo. V.P. Exploration, Alexco Resource Corp. Mr. Dodd is a Qualified Person as defined in National Instrument 43-101. The mineral resources for the Bellekeno project were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101.

Table 1: Mineral Resource Statement, Bellekeno Project, Yukon Territory, Alexco Resource Corp., October 5, 2009

Indicated	**Tonnes**	**Ag (gpt)**	**Pb (%)**	**Zn (%)**
East 48 Upper	17,026	996	3.7	9.9
East 48 Mid	80,210	520	3.8	7.2
East 49 all	24,325	643	4.2	2.1
99 all	91,724	995	7.5	4.2
SW all	221,177	986	12.5	7.1
Total Indicated	**434,462**	**883**	**9.0**	**6.4**
Inferred	**Tonnes**	**Ag (gpt)**	**Pb (%)**	**Zn (%)**
Total (Lower East)	**82,981**	**289**	**2.8**	**22.2**
Total Indicated & Inferred	**Tonnes**	**Ag (gpt)**	**Pb (%)**	**Zn (%)**
Inferred	82,981	289	2.8	22.2
Indicated	434,462	883	9.0	6.4
Total All Categories	**517,443**	**788**	**8.0**	**8.9**

Resources are reported based on an NSR cut off value of $185 per tonne. NSR values were calculated on an in-situ (undiluted) basis using the price and exchange rate inputs shown in Table 2.

Table 2: Metal Prices and Exchange Rate

Zone	**USD:CAD exchange**	**Ag US$/0z**	**Pb US$/lb**	**Zn US$/lb**
SW	0.90	$15.25	$0.675	$0.80
99	0.90	$15.25	$0.675	$0.80
East	0.90	$14.50	$0.600	$0.90

3.2 Mineral Reserve Statement

A Preliminary Economic Assessment does not support an estimate of Mineral Reserves. A pre-feasibility study or feasibility study is required to support an estimate Mineral Reserves.

4 Mining

4.1 Mine Geotechnical

4.1.1 Introduction

The Keno Hill Mining camp has long been recognized as a polymetallic silver-lead-zinc vein district with characteristics possibly similar to other well known mining districts in the world. Examples of this type of mineralization include the Kokanee Range (Slocan), British Columbia, Coeur d'Alene, Idaho, and Fresnillo, Mexico.

The Bellekeno vein system consists of at least 11 individual veins with variable strike, dip and thickness. The target vein for mining is 48 vein and associated vein splays.

The 48 vein system is sub-divided into three segments: the Southwest ("SW"), 99 and the East zones. Refer to Figure 1. Planned mining areas are shown in cyan colour.

A number of data sources have been employed to gain the best understanding of the possible ground conditions at the Bellekeno Project. Underground resource definition drill holes from the 2009 drilling campaign and underground exposures, in addition to geological mapping and logging, and core photographs, have been used as the primary sources of data for both the geotechnical evaluation and structural updates.

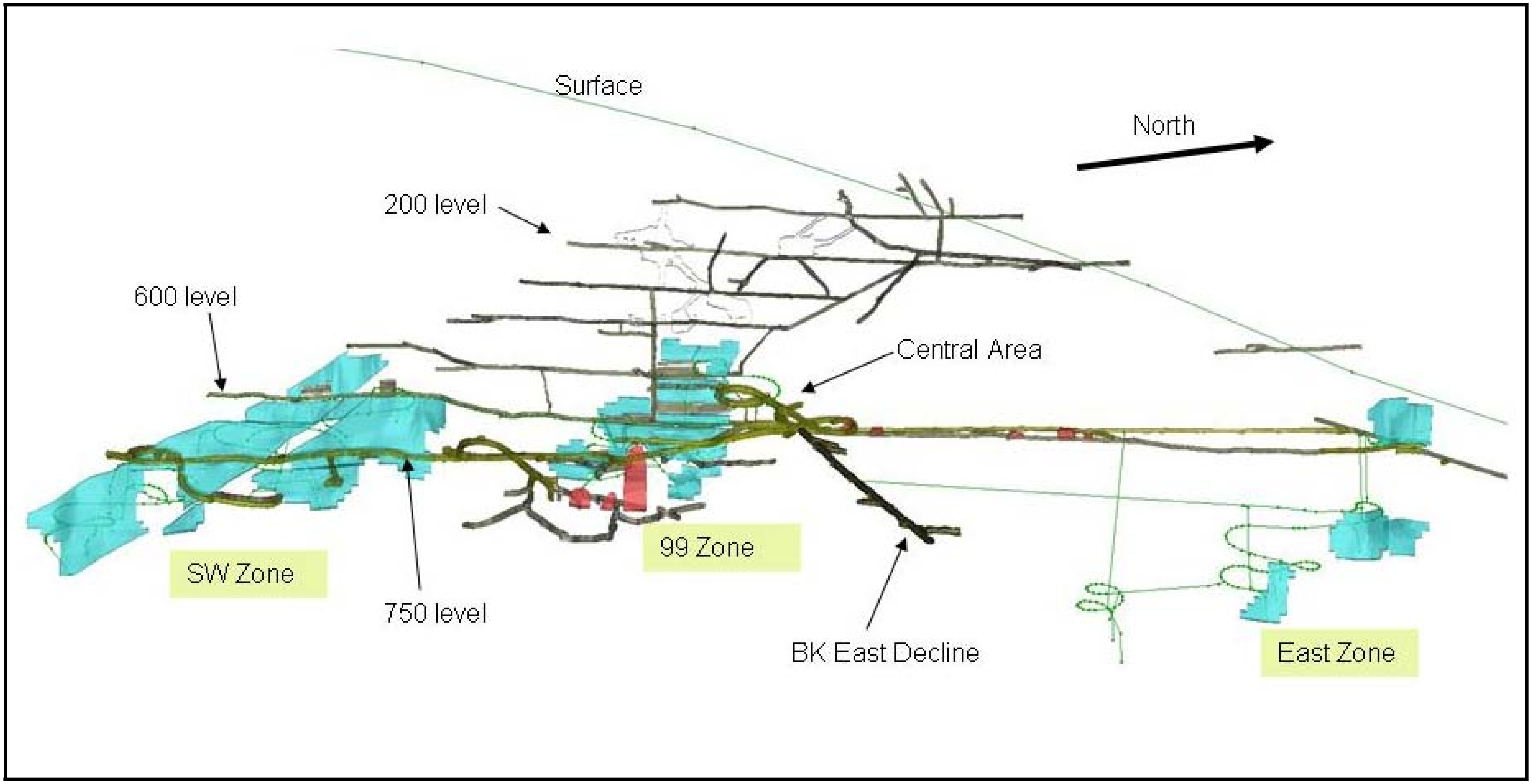


Figure 1: 3D view of Bellekeno Mine (looks west)

4.1.2 Previous Mining

Historically, underground mining at Bellekeno was completed by drifting on vein and shrinkage stoping using tracked equipment; all development was supported with square set timbers and timber pole lagging. Few areas were impassable using this method and where instability on the vein was an issue, the heading was abandoned and a bypass commenced in more competent foot wall ("FW") lithologies.

The most recent mining has been completed using an overhand cut and fill method utilizing rubber tired equipment. Support for drifting on the vein (for exploration purposes) was completed using friction anchors and weld-wire mesh, with varying degrees of success achieved in the variable vein conditions. The use of shotcrete was implemented only in the final stages before mine closure to successfully mine through the poor ground conditions encountered on vein, on the 800 level.

The 6-99-N cut and fill stope is observed with support installed in the sidewalls only. The back of this excavation is still standing approximately 20 years after completion.

4.1.3 Structural Geology

At the request of SRK, a structural investigation and interpretation has been completed for the Bellekeno Mine area. Alexco Resource Corp contracted Structural Geologist Bruce Otto, P. Geo. to complete the investigation. Mr. Otto's initial scope of work was to investigate available data sources and compile a revised structural interpretation for the incorporation with geotechnical findings; provide detail on the structural and geological controls of the uncontrolled caving initiated during mining on the 800 level; and comment on the existence of similar conditions susceptible to caving elsewhere in the Bellekeno mining areas.

Structural Results – 99 zone

The following paragraphs (in italics) are excerpts from the report authored by Bruce R. Otto (October 8, 2009) entitled "Structural and stratigraphic relationships at the Bellekeno Mine; their control on grade and thickness and their implications for exploration." The full report is included as Appendix XX.1.

The 48-vein fault forms a 3-dimensionally curviplanar surface that ranges planimetrically in strike from 025° to over 065°, averaging 038° through the extent of mine workings. It moved sinistrally a distance of approximating 35 meters along a 080° / -65° vector. Initial movement of the fault where its strike was north of 38° azimuth formed dilational zones while segments with strikes east of 038° resulted in less dilation, and perhaps transpression.

The mine section occupies the top third of the central quartzite and consists of massive quartzite with interlayers of variably carbonaceous schist and laminated to thin-bedded quartzite. A thick greenstone sill occupies a stratigraphic interval above the east zone and below the 99 zone. The sill occurs near the boundary of a fundamental vertical change in the ratio of quartzite to schist. The section below the sill consists primarily of massive quartzite with only one 12-meter-thick graphitic schist interval. The section above includes multiple intervals of schist that separate massive quartzite units. The 99 zone includes parts of four quartzite intervals separated by three of schist, and that hosting the southwest zone includes a diversity of massive quartzite beds separated by abundant intervals of interbedded schist and thick units of intricately interlayered thin-bedded quartzite and schist.

Bypass Schist
The bypass schist separates the two quartzite members of the East zone. Contact relationships are unclear because it has not yet been mapped at its only exposure, in the 600 footwall bypass drift and formerly mapped exposures in the main 600 drift have been compromised by faulting. Drill logs show that it consists of 12 meters of laminated carbonaceous schist with thinly interlayered and thin-bedded to laminated quartzite. Its base forms the relatively sharp stratigraphic top of the East zone mineralized shoot.

COF Schist
The COF (Cause of Failure) schist, named for associated severe caving on the 800 level is the first significant unit of schist above the bypass schist in the east zone. It consists of a 4-meter-thick section of strongly graphitic schist with very little interleaved quartzite. The COF schist lies in sharp contact with the underlying 600 quartzite and grades upward into thin-bedded quartzite-bearing schist and thence gradationally into the overlying 650 quartzite. Where exposed in the 750 decline it is highly foliated and displays a superimposed steeper dipping cleavage that lies at a 20° angle to the primary foliation. It is distinctly more carbonaceous than the other schist intervals, and forms an aquaclude; quartzitic strata directly above generally carry abundant water, generally causing a rain storm wherever workings pass through it.

Vent Raise Schist
The vent raise schist, named for exposures near the vent raise in the 750 decline, is an 8-meter thick unit consisting of basal graphitic schist grading upward to a section of thin-bedded to laminated fine quartzite with graphitic partings and thence upward to thicker quartzite beds with schistose partings. The basal graphitic schist lies in sharp contact above the thick-bedded sands of the 650 quartzite and its top grades upward into the thicker quartzite beds of the 700 quartzite. The vent raise schist passes through the 800 level in the 48-vein hanging wall, where it is associated with unstable backs and caving.

700 Schist
The 700 schist, named for exposures near the 700 level in the 750 decline, is a 5-meter thick unit consisting of basal graphitic schist with minor laminations and thin beds of quartzite that lies in sharp contact with the underlying 700 quartzite, and grades upward to interlayered thin-bedded quartzite with 1 - 20 cm graphitic schist interlayers, thence gradationally up to medium- to thick-bedded quartzite of the Bankers quartzite. The middle of the 700 schist contains one anomalous thick-bedded interlayer of quartzite.

Previous Instability – 800 Level Self Mining
Ground instability on the 800 level occurred during mining in the late 1980s. Refer to Figure 15. Monthly and weekly reports of development document three attempts to access the vein, resulting in "severe overbreak…and extensive cave-in" of the footwall to the 48 vein. Mining during the late 1990s once again entered the vein on the 800 level, applying shotcrete immediately after mucking to support the excavation. An attempt to mine through the self-mined zone from the 700 level resulted in further instability and abandonment of the heading. The following paragraphs (in italics) are excerpts from the report authored by Bruce R. Otto (October 8, 2009).

Severe caving on the 800 level occurs in a lithologically diverse part of the section where a number of structural and stratigraphic geometries coalesce. It is difficult to ascertain the specific cause without the ability to observe the caved area first-hand. Caving occurred primarily in schistose strata, and that is clearly one of the most important factors. Caving occurs along the fault at the northeast edge of the 99 zone where it curves almost imperceptibly *to a more easterly strike as it enters the 600 quartzite. Curvature to this geometry would form a region of strike-parallel, or perhaps transpressional displacement. The combination of schistose strata deformed by transpressional displacement may have caused intense fracturing that when mined through could not hold a back. When faulted schistose strata tend to drag into the fault zone. The vertically extensive geometry of caving suggests that schistose strata may have been dragged into the fault zone. The most reasonable explanation for the 800-level caving would therefore involve transpressional faulting of a strongly schistose section, perhaps where broken schist forms the primary proto-lithology of the fault gouge.*

The propensity to cave in vertically continuous zones therefore appears to require three spatially overlapping factors: the intersection of abundant schistose strata with the 48-vein fault surface, segments of the fault where these strata were draped during early transtensional faulting, and later transpressional displacement.

The current geological interpretation indicates three schist packages intersecting the 48 vein in the vicinity of the 99 zone. Due to the proximity to transpressional zones, these packages are expected to have similar ground conditions along strike as those intersected during previous mining in the 99 zone. Figure 2 shows the modelled schist packages and zones of transpression.

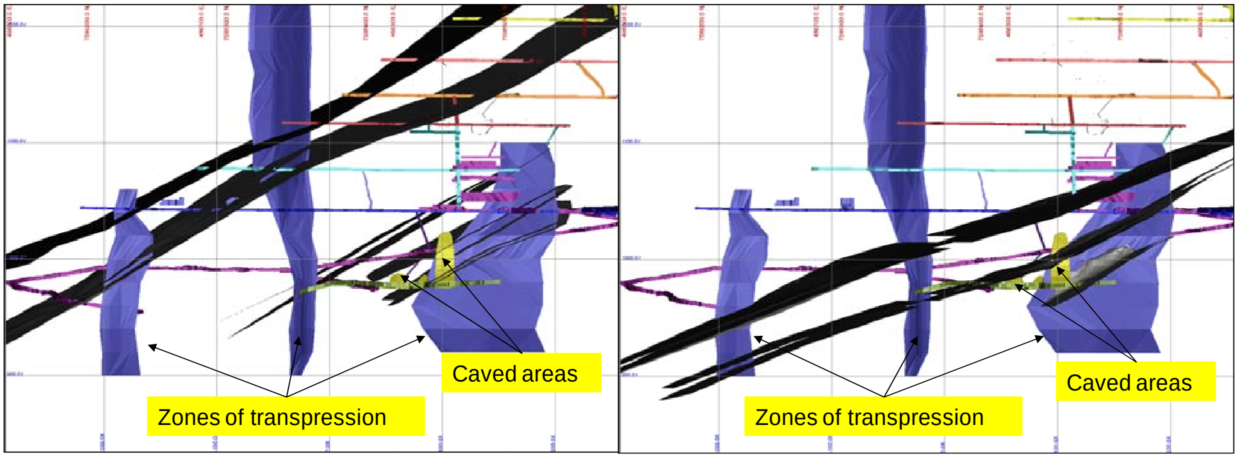


Figure 2: Bellekeno 99 zone. Zones of transpression and interpreted schist packages (FW left; HW right)

Discussion

Based on the results of the structural investigation, schist packages are the primary source of historic ground instabilities at Bellekeno. The fact that most instabilities (including zones of overbreak) have occurred in the schist should be viewed in relation to the limited suite of mining and ground support methods available during the previous mining undertakings.

The conditions resulting in the self-mining from the 800 level are specific to the 99 zone. This is the only area where interpreted FW and HW schist packages exist at similar elevations across the 48 vein-fault system, within a zone of transpression. Additional geotechnical factors that may have assisted in the initiation of self-mining are:

- Lack of geological understanding of the Bellekeno area (specifically the 99 Zone);
- Untimely installation of ground support;
- Lack of preparedness to handle these types of ground conditions;
- Limited availability of ground support methods/techniques;
- Hydrogeological conditions.

As stated previously, mining was successfully completed through the vein elsewhere on the 800 level using shotcrete, bolts, and mesh. It is apparent that the timing of support installation is crucial in supporting weak ground areas. Additionally, the lack of geological knowledge and preparedness did not allow for alternative methods of support to be provided, or for miners to completely avoid the area.

4.1.4 Rock Mass Classification

The 48 vein-fault system is hosted by a series of moderately dipping quartzite, schist, and greenstone units which have been offset in a sinestral (left lateral) strike-slip movement along the 48 fault. The particulars of each zone are described below. Figure 3, Figure 4 and Figure 5 illustrate the interpreted structural geology and ground conditions for the SW, 99, and East Zones, respectively.

SW zone

The SW Zone is predominantly medium to thick bedded quartzite, with two significant graphitic schist units (Figure 3). The quartzite in general is of fair to good rock mass quality, although more broken zones exist proximal to the vein. The stratigraphically lower schist unit (the 750 schist) is considered to be of fair to poor rock mass quality: the poorer quality material is observed in drillcore intercepts only at this stage. An interbedded carbonaceous quartzite/schist, and a thin bedded schist unit form the upper unit, and define the southern extent of known mineralization. These units are of fair rock mass quality.

Support installed in the 750 decline consists of friction anchors, and in some areas additional welds-wire mesh.

Good exposures of the mineralized vein are available in the 750 vein access, Level No.1 cross-cut. At this intersection the vein is approximately five meters wide, and comprises siderite and sphalerite (both intact and sanded), massive galena veins, clay gouge, and quartz-carbonate contact breccia. Variability is evident from left-rib to right-rib, with vein components pinching and swelling across the three meter wide cross-cut. Drill holes completed adjacent to the cross-cut indicate that these conditions are not representative of all areas of the vein in the SW zone.

99 zone

The 99 zone contains the majority of the schist in the Bellekeno area (Figure 4). Three distinct schist packages intersect the 99 zone: the 700 schist and Vent Raise schist are of fair rock mass quality, while the Cause of Failure (COF) schist is a 4 metre (“m”) zone of poor to very poor rock mass quality.

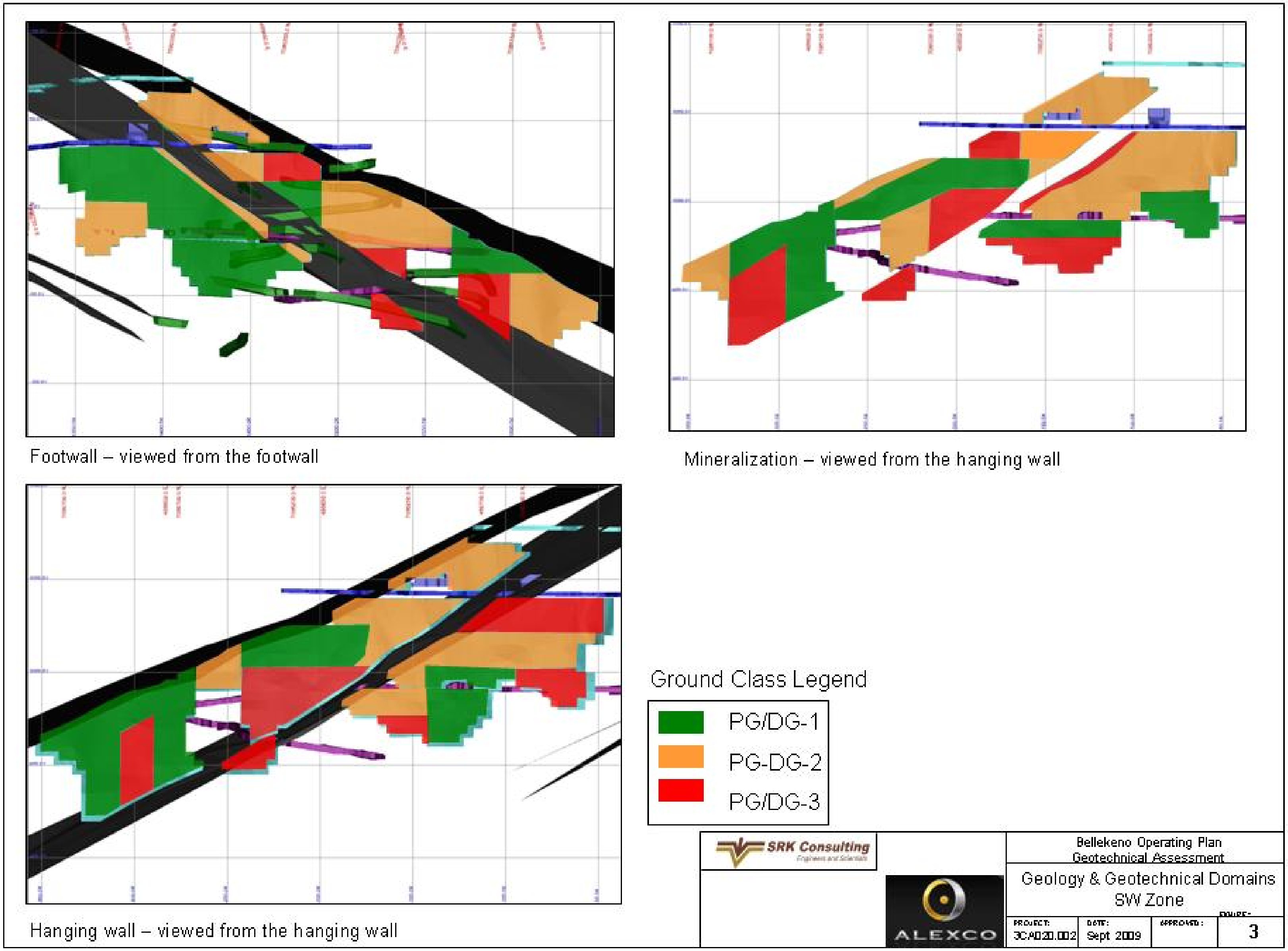


Figure 3: Geology & Geotechnical Domains – SW Zone vertical projections

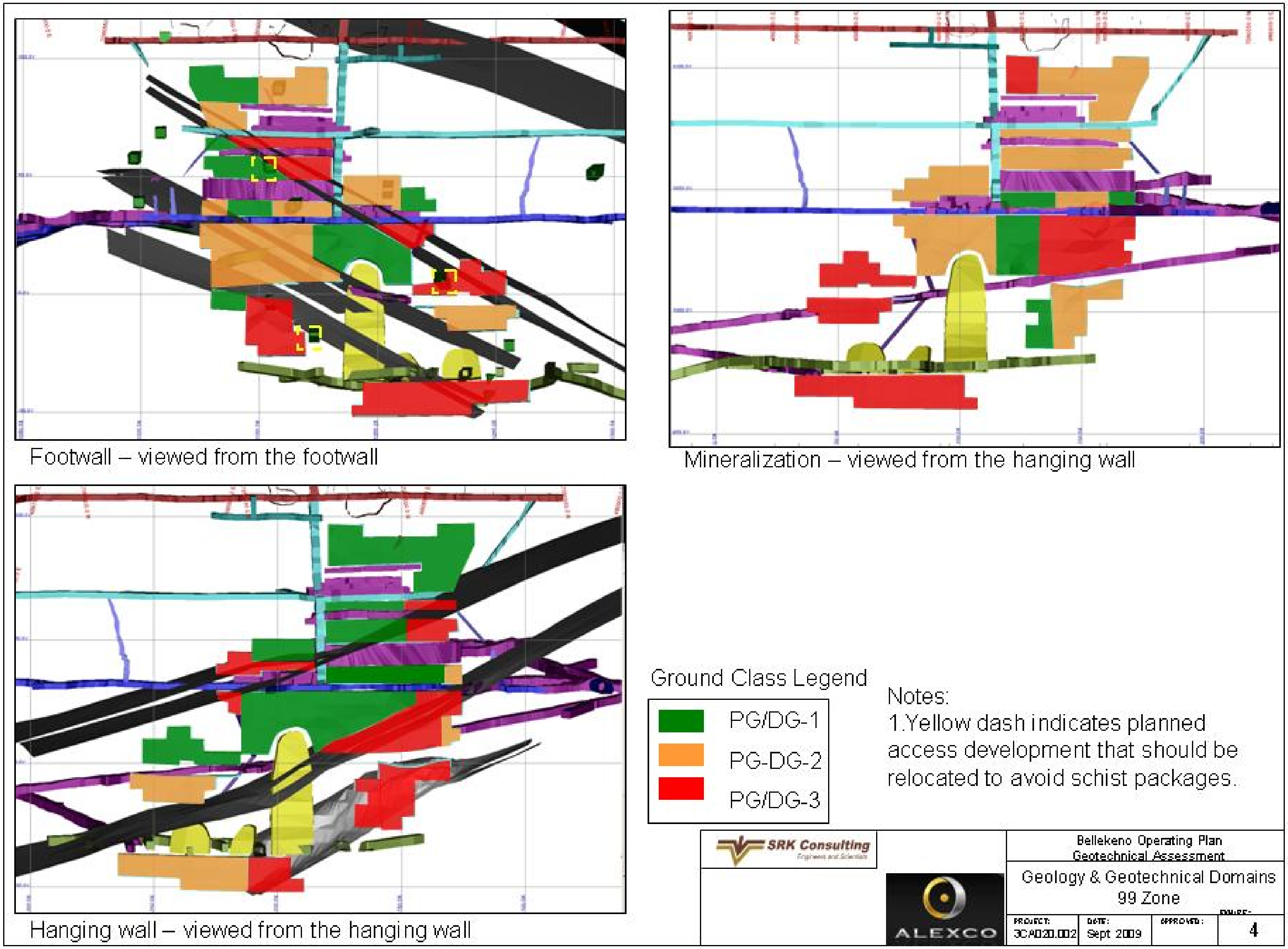


Figure 4: Geology & Geotechnical Domains – 99 Zone vertical projections

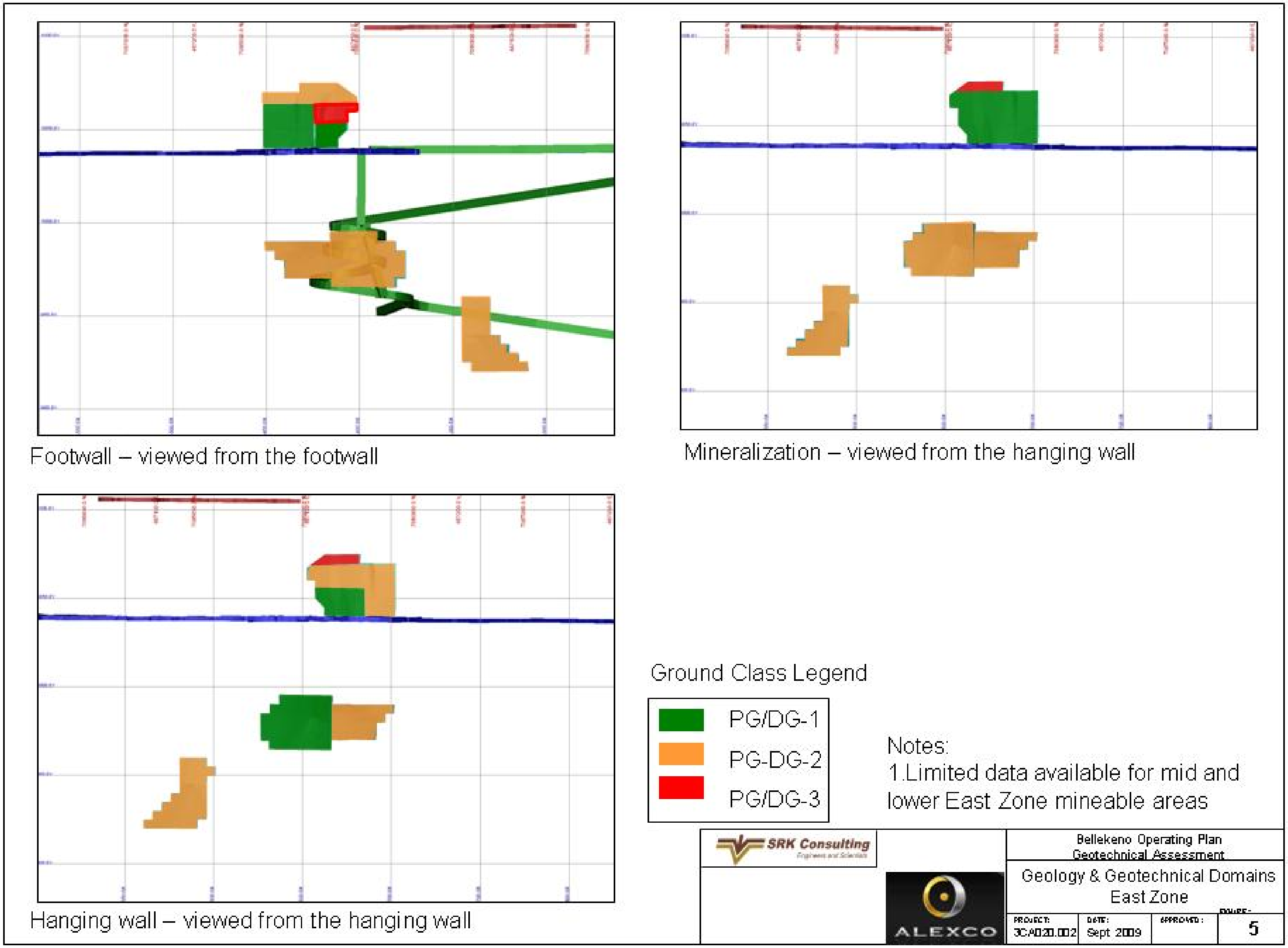


Figure 5: Geology & Geotechnical Domains – East Zone vertical projections

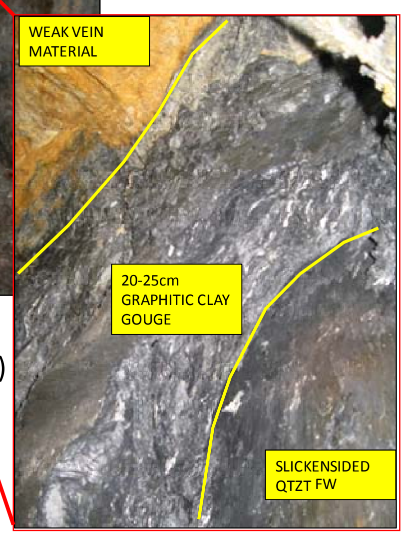
These weaker units (relative to the quartzite) are interpreted to be a significant factor in the self-mining initiated from the 800 level. Footwall and hangingwall quartzite throughout the 99 zone is considered to be of fair to good rock mass quality.

Where development headings have encountered schist packages in the 99 Zone, these have been controlled with friction anchors and mesh. Where schist forms the HW or FW to the 48 vein, it is expected that ground control will become more difficult, requiring shotcrete and possibly spiling to maintain stability.

The COF schist package in particular should be approached with caution, or avoided entirely. Where exposed in the 750 decline, this unit serves as an aquitard; the quartzite unit above forms an aquifer and running water is observed.

Vein materials have been observed both underground, and in drill core in the 99 zone. The most significant difference in the 99 zone is the intense oxidation of the vein; the weaker materials possibly resulting in poor recoveries during the 2009 underground drilling program. In underground and drill core exposures, the vein exhibits high variability both along strike and dip (on a scale of less than five meters). Weak vein contacts with thicknesses between 20 centimetres ("cm") and 1.5m have been observed. Figure 6 shows the 6-48-S drive developed on vein, and the ~1.5m of clay gouge and slickensided FW contact. A ~1.0m wide quartz-carbonate breccia is observed on the HW contact, and it is unclear whether this has been removed as part of the mining width. Based on SRK's findings, it is not possible to delineate contact zones from drill holes.

6-48-S drive was abandoned (on vein) in poor ground ~ 20m beyond this point (intersection with KOD schist?).



- Historic support: Timber sets and lagging
- Rehab support: Screen and split sets (not a travel way)
- 1.5m unmineralized vein material and 20-25cm graphitic clay gouge on FW (above and right)
- Quartzite FW slickensided but intact and very strong
- Undulating rough HW contact. Quartz-carbonate breccia possibly removed as part of mining width here.

Figure 6: Vein and contact zone in the abandoned 6-48-S heading

East zone

Geology in the East Zone comprises predominantly quartzite of fair to good rock mass quality with two schist packages; Figure 5 presents the interpreted geology and ground conditions in the East Zone. Target mineralization is contained within the 48 East vein. Conditions in the east upper zone have been interpreted from detailed photo reviews of drillcore, and geotechnical data collected by Alexco. Conditions in the East Mid zone (upper and lower blocks below 600 level) have been interpreted from adjacent drill core intercepts. These conditions should be reviewed in detail when more drilling is completed in these areas.

Vein materials in the East zone have observed in core photographs only, and are considered to be of 'poor' rock mass quality. Weak to moderate oxidation of the vein and adjacent HW/FW zones is evident, although not as intense as the 99 zone. Vein materials generally appear to be intact, with indications of contact zone breccias observed in some holes.

Ground Classes

Based on the interpreted geotechnical conditions at Bellekeno, the following ground classes have been defined. Refer to Table 3. These are based on the lithology determined from the face of the advancing heading.

Table 3: Ground Classes

Area	Ground Class	Typical Conditions
Development Headings	DG-1	Quartzite with less than 20% interbeds of schist (graphitic, chloritic). RQD* 70 – 90%, and intact rock strength ("IRS") 100 – 150MPa.
	DG-2	Quartzite with 20 – 80% interbeds of schist (graphitic, chloritic). RQD 60 – 80%, and IRS 40 – 90MPa.
	DG-3	Fault/shear zones comprising predominantly graphitic schist. RQD <50% and IRS 15 – 40MPa.
Production Headings (incl. vein cross-cuts)	PG-1	Predominantly intact vein materials with RQD 50 – 60% and IRS 20 – 40MPa. Weaker materials comprise <10% of vein; HW and FW units are competent and intact.
	PG-2	Predominantly intact vein materials, with brecciated or sheared HW and/or FW contacts. RQD 20 – 50% and IRS 15 – 30MPa. Weaker materials comprise 10% - 40% of vein width.
	PG-3	Predominantly soil strength materials in vein. HW and FW units are broken or sheared. RQD 0 – 30%, and IRS <15MPa. Excavation potentially does not require the use of explosives.

* "RQD" Rock Quality Designation

4.1.5 Mining Method Selection

Following a preliminary review of the deposit characteristics by SRK (2008), three possible mining methods were proposed for the Bellekeno Mine:

- Overhand or underhand cut and fill;
- Shrinkage stoping;
- Longhole stoping.

These options were chosen on the basis of a ‘weak vein, strong HW/FW’ understanding. Underhand mining methods were considered appropriate to avoid having to use heavy support in the vein back following each round of advance.

Following the 2009 geotechnical investigation, and based on the current understanding of the 48 vein and adjacent vein contacts, an underhand cut and fill method is no longer considered suitable for mining at Bellekeno. Geotechnical conditions indicate that an overhand cut and fill method is more appropriate for the observed ground conditions and geometry. The main factors contributing to this decision are:

- Variability of vein contact conditions: the observed vein contacts are extremely variable in nature both along strike and dip. Conditions present include clay and slickensided contacts through to sharp hard rock contacts. It is not considered feasible to delineate these contacts due to the changes in contact on the metre(s) scale;
- Variability of vein and contact thickness: significant thicknesses of unmineralized contact zone are evident at Bellekeno including quartz/carbonate breccias, and clay. When combined with the variable vein width, significant dilution would be included into the mining sequence. This would be increased in the narrower segments of the vein where the minimum mining width is dictated by equipment dimensions;
- Fall of ground risk: due to the weak nature of the vein contacts it is not considered safe to ‘hang’ backfill in an underhand method. The risk related to possible shear failure along the backfill/weak contact is considered to be high, and instability related to poor backfill performance and placement could compromise worker safety and mine production.

Figure 7 illustrates an example of an approximately 2m wide weak graphitic schist zone within the hanging wall of the SW zone. This type of ground is considered unsuitable for the placement of a stable backfill for use in an underhand method, and it is not practical to remove this material to provide a solid hangingwall contact.

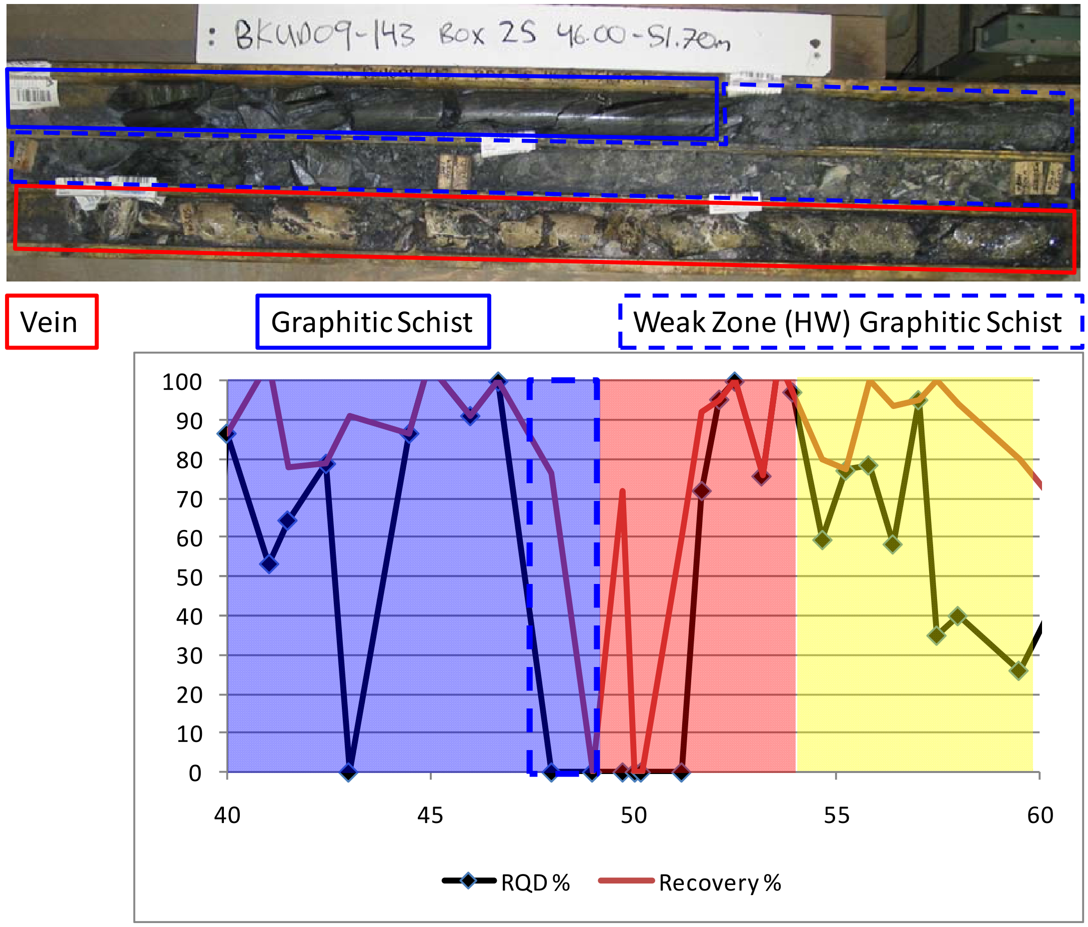


Figure 7: Drillhole photograph and graphical representation on the weak graphitic schist material found on the HW contact (SW zone drill hole)

4.1.6 Geotechnical Risks

99 zone

- Mining below 800 level self-mined caved zone – mineable tonnage exists beneath the caved areas on the 800 level. Ground conditions are anticipated to be similar to those encountered in the 800 level where self-mining was initiated during raise development. Support in the form of spiling, shotcrete, bolts and mesh will be required to safely access and develop these areas. Even with such support, a reduced advance and conservative estimate of final extraction tonnage should be assumed for mine planning;
- Access to the 800 level self-mined areas – currently these are timbered up, and it may not be feasible to fill these areas prior to mining. Fill that is placed without removing the timbers (and rock debris) may not be of sufficient quality to form a sill pillar beneath which mining can take place. If this is the case, the possibility of leaving a 5m sill pillar beneath the caved areas should be investigated;
- Mining within schist units – several planned mining areas within the 99 zone have portions of HW and FW bound by schists. These areas are considered to be higher risk due to the weaker ground conditions observed within the schist packages. Specifically, the COF schist package, where it intersects the 48 vein is of very poor to poor rock mass quality. Although

these packages are reasonably well defined in the HW, FW intersections are sparse and some inferences have been made regarding the exact contact locations;

- Dilution – reduced core recoveries in the 99 zone were observed in the 2009 drilling program. Following a review of a series of twinned NQ/HQ drillholes, it was concluded that core recovery within the vein was not dictated by material strength. Zones of core loss have been considered to be of poor rock mass quality, and the possibility of leaving unmineralized vein materials on the HW/FW is not considered feasible at this stage of the evaluation;
- Ramp access – these are considered to be long term man entry excavations, and should be located within the better quality quartzite unit where possible. Mining along weak schist zones will require increased ground support (shotcrete, potentially spiling) and may potentially risk the completion of the access.

SW zone

- Mining within schist units – observed schist ground conditions in development headings are generally of fair quality; however several intercepts in drill core showed zones of poorer quality schist. While these intercepts are outside of the planned mining areas, it is possible that there are areas where schist ground conditions are worse than observed;
- Water inflows – water inflow from drillholes has been observed in the SW zone flowing from drillholes. Valves installed on drillholes are opened periodically, and while initial flows are significant, they are observed to drop-off fairly rapidly. Inflows of water to production headings have been noted in historic mine records from Bellekeno, and dewatering of the vein using pre-production drillholes should be completed to allow adequate time for water to drain from the vein. Water intersected during production will adversely affect weaker clay and incohesive vein materials. A dewatered vein will also benefit the application of shotcrete during support installation.

East zone

- Mining in East Mid zone – limited geotechnical data exists for the East Zone Mid (upper and lower) mining blocks. Conditions are interpreted to be fair to good; more recent underground drillholes indicate that weaker areas do exist in the East Zone. Mining is not expected to encounter weaker altered areas associated with the greenstone sill;
- Development headings in schist – production headings are not anticipated to intersect schist units in the East zone. Access development for the East zone will traverse the Bypass schist unit. As this heading will be a man-access, increased support will be required to prevent deterioration of the heading.

4.1.7 Support Requirement Evaluation

Development and production support requirements are based on the ground classes that are determined to be representative of the likely rock mass conditions.

Mining practices in and around the deposit will need to be cautious and excavation size and overbreak limited as much as possible to maintain the stability of the excavations. Mining rates within the vein are expected to be low due to the requirement of short face rounds, and the high levels of support to be installed.

Support Classes

Support classes have been determined for the ground classes at Bellekeno. Options have been provided in some classes to allow for flexibility in the selection of mining equipment. Support standards are attached as Appendix XX.2.

Development support requirements

In general, the infrastructure is considered to be open for the long term situation, and support has been designed accordingly. The infrastructure has been designed to avoid areas with potential poor ground conditions; in some situations this is unavoidable and support will be increased to provide long term stability. Table 4 outlines the recommended ground support for development headings.

Intersection Support

Where intersections will be formed in PG-1 and PG-2 ground conditions, a standardized bolting program will be required. In addition to the standard Support Class requirements, any span opened over and above the standard development width (4.5m) should be supported with additional grouted rebar bolts installed with mesh straps throughout the intersection. It is a requirement that permanent support, suitable to the final excavated dimension, be installed prior to the breakaway being taken. Table 5 outlines the support requirements for large intersections.

Table 4: Support Classes for Development Headings

<table>
<tr><th>Area</th><th>Ground Class</th><th>Support Class</th><th>Support Requirements</th></tr>
<tr><td rowspan="4">Development Headings</td><td rowspan="2">DG-1</td><td>DS-1</td><td>1.8m friction anchors on 1.2x1.2m diamond spacing across back and shoulders. #6 galvinized welded wire mesh across back and shoulders.
Additional spot bolting down ribs as required</td></tr>
<tr><td>DS-2</td><td>1.8m grouted rebar on 1.2x1.2m diamond spacing across back and shoulders. #6 galvinized welded wire mesh across back and shoulders.
Additional spot bolting as required</td></tr>
<tr><td>DG-3</td><td>DS-3</td><td>1.8m resin grouted rebar on 1.2x1.2m diamond spacing down to 1.4m above floor. #6 galvinized welded wire mesh down to 1.2m above sill. Additional spot bolting as required. Mesh straps as required</td></tr>
<tr><td>DG-4</td><td>DS-4</td><td>25mm flash-coat shotcrete in back and ribs. 2.4m resin grouted rebar on 1.0x1.0m diamond spacing down to 1.2m above floor.
#6 galvanized welded wire mesh down to 1.0m above sill. Mesh straps as required. 50-75mm additional shotcrete in back and ribs.
If required: spiling at 30cm centres with 4.5m self-drilling or grouted hollow bar spiles</td></tr>
</table>

Table 5: Support Requirements for Large Intersections

<table>
<tr><th>Area</th><th>Ground Class</th><th>Support Requirements</th></tr>
<tr><td rowspan="3">Development Headings</td><td>DG-1</td><td rowspan="2">2.4m grouted rebar on 3.0m diamond spacing throughout intersection
Mesh Straps
Final support installed before breakaway taken</td></tr>
<tr><td>DG-2</td></tr>
<tr><td>DG-3</td><td>4.5m cable bolts on 2.0m diamond spacing throughout intersection
Mesh straps
Final support installed before breakaway taken</td></tr>
</table>

Production Support Requirements

Support design for production headings has been based on observed ground conditions, historic support performance, and anticipated ground conditions. It should be assumed that an increase in ground support will delay advance rates. Table 6 outlines support for production headings.

Table 6: Support Classes for Production Headings

Area	Ground Class	Support Class	Support Requirements
Production Headings (incl. vein cross-cuts)	PG-1	PS-1	1.8m friction anchors across back on 1.2x1.2m spacing; rib bolting as required. If required: #6 galvanized welded wire mesh across back and shoulders. If required: 25mm flash-coat shotcrete in back; rib coverage as required.
		PS-2	1.8m friction anchors across back on 1.2x1.2m spacing; rib bolting as required. #6 galvanized welded wire mesh across back and shoulders. If required: 25mm flash-coat shotcrete in back; rib coverage as required
	PG-2	PS-3	1.8m resin grouted rebar* across back and either friction anchors or grouted rebar in ribs on 1.0x1.0m spacing #6 galvanized welded wire mesh across back and shoulders. Mesh straps as required.
	PG-3	PS-4	25mm flash-coat shotcrete on back and ribs. 1.8m grouted rebar* on 1.0x1.0m spacing in back and ribs. #6 galvanized welded wire mesh down to 0.8m above the sill 50-75mm shotcrete in back and walls If required: spiling at 30cm centers with 4.5m self-drilling or grouted hollow bar spiles. Mesh straps as required.

4.1.8 Shotcrete Requirements

Where required, shotcrete will be used to provide short and long term stability to access and production headings. In production headings, the main purpose of the shotcrete will be to prevent progressive ravelling of the rock mass. Where required, a 25 millimetre (“mm”) flash-coat of shotcrete can be applied immediately after mucking to tie the rock mass together prior to the installation of conventional bolts and mesh support. Additional shotcrete can be applied if ground conditions dictate. In areas where shotcrete is required, it is important that shotcrete is applied as soon as possible following blasting and mucking to control the behaviour of the rock mass and prevent unravelling of the rock mass before final support is installed.

In excavations expected to be open for the long-term, shotcrete can be used to prevent rock mass dilation and ravelling (e.g. where schist packages are intersected). In this situation, shotcrete can be applied as a flash-coat, with additional shotcrete thickness (50-75mm) added following the installation of conventional support.

Dry-mix shotcrete is the recommended product for use at Bellekeno due to the requirement for compact equipment that can be rapidly moved around the mine. It is recommended that multiple dry-mix shotcrete machines are available for back up, and to support the need for multiple headings to be operating at one time. For most applications, additives will be used to provide the fast set-up times required to prevent ravelling of vein materials prior to conventional support installation.

Shotcrete Application

The quality of applied shotcrete is highly dependent on the preparation and application methods used. The following notes should be considered for shotcrete application:

- Rock surfaces should be cleaned and scaled prior to shotcrete application (where possible). Water can be used to complete both of these processes. Dust and gouge materials should be removed from surfaces to allow proper adhesion of the shotcrete;
- Shooting distances should be maintained according to equipment and mix specification. This will prevent excessive rebound and provide proper adhesion of the shotcrete to the rock mass;
- Operators should be well trained in the use of equipment and in the practice of spraying test panels to provide quality control on the final shotcrete product. A quality control program should be developed to suit the needs of the mine;
- Drainage should be provided where water bearing rock masses are intersected.

4.1.9 Backfill

Cemented Tails/Rock Fill

Cemented tails and rock fill is now the preferred backfill method for Bellekeno. Where sill pillars are required, a cemented fill will be used to provide a stable back to mine up to from beneath. Extraction of the vein from the final lift requires that the pillar is self-supporting and maintains integrity while the heading is active. The quality and the placement of the fill are both important factors in this application. An increased cement content of between four and five percent will be required to provide the required strength of the pillar. In areas where additional caution is required during final lift extraction, the lift will be mined using up-holes and remote mucking.

Careful preparation of the excavation where cemented fill is to be placed will be required, including blasting beyond the vein contacts to provide a clean, rough surface for the fill to hang on. The floor should be cleaned prior to placement to prevent material falling from the back following mining. An appropriate lead time should be provided to allow set-up and cure for the cemented fill. Standard quality control procedures (e.g. unconfined compressive strength and slump tests) should be completed during batching and following placement of cemented tailing fill materials.

Uncemented Rock/Tailings Fill

For the overhand cut and fill mining method, uncemented rock fill/development waste and/or tailings fill will likely be utilized. These materials should be placed into headings as tight to the back as possible.

Paste Fill

A paste backfill system had initially been considered a likely option if the underhand cut and fill method was deemed to be the most suitable mining method. As mentioned previously, the occurrence of weak contact zones and weak wall/country rocks has indicated that an underhand method maybe too risky because of the possible instability of the placed backfill. As the mining method is to be an overhand method, the need for a paste fill plant and paste backfill has been discounted as an option for Bellekeno

4.1.10 Conclusions and Recommendations

Mining conditions at Bellekeno will be locally difficult due to the poor ground conditions within the vein and in some instances within the hangingwall and/or footwall units. Based solely on vein conditions, an underhand method would be appropriate for mining the deposit; however, because weak country rock conditions exist, an overhand method using high quality support practices, combined with cemented tails and rock fill in sill pillar areas is the required mining method.

Rock mass conditions have been assigned for development and production headings, and support systems designed for the interpreted ground conditions. The application of careful mining, the early use of a high quality shotcrete and the combinations of friction anchors, grouted rebar, welded wire mesh and spiling are expected to handle the anticipated ground conditions.

Previous instability (self-mining) initiated from the 800 level is attributable to three specific factors: the intersection of abundant schistose strata with the 48 fault surface; segments of the fault where these strata were draped during early transtensional faulting, and later transpressional displacement. These conditions are interpreted to be specific to the 99 Zone. Areas with possible similar ground conditions have now been identified and will require specific attention prior during pre-production to further assess the mining risk and what tactical mining strategies are required to mine these areaes if the decision is taken to do so.

Mining practices in and around the deposit will need to be cautious and excavation size and overbreak limited as much as possible to maintain the stability of the excavations. Mining rates within the vein are expected to be low due to the requirement of short face rounds and the high levels of support required to be installed. To achieve safe and productive mining the following recommendations need to be considered:

- The ground class that is being predicted to be mined in the various mining blocks has been determined from definition drilling and will need to be further evaluated with additional infill drilling, evaluation at the mining face and from cover/probe drilling during pre-production/production. Cover drilling is essential during access cross-cut development to aid in the assessment of the ground and hydrological conditions for the timely implementation of the correct support for the ground conditions ahead of the face;
- An ongoing program of face mapping and probe drilling of the advancing face should be carried out to assist with prediction of future face conditions, confirm and record current ground conditions and to determine the appropriate support class to be used;
- Excavation performance and the performance of the installed support should be regularly assessed using monitoring stations installed in the more critical areas of the deposit drives and intersections. The behaviour of the implemented ground support should be audited to assess support type suitability and provide recommendations for future support;
- Due to the locally difficult ground conditions it will be important to also have additional support options available if ground instability is experienced. The following options should be seriously considered: a shotcrete product with accelerated set up times and better wet ground performance; cementitious foam material and application machine for void filling and adequate shuttering material that can be used in the very weak areas.

4.2 Planned Mining Methods

4.2.1 Mining Context of Bellekeno 48 Vein

The relevant characteristics of the Bellekeno 48 vein from a mining method selection perspective include:

- On a deposit wide basis, a pronounced level of variability in metal grade distribution, with SW and 99 zones rich in silver-lead, and lower parts of East zone tending to be zinc rich;
- Average silver grades ranging from 250-2100 grams of silver per tonne ("gpt Ag") in the stoping blocks studied;
- Vein geometries with variable dips of 73-81 degrees for SW and 99 zones, and approximately 60 degrees for East zone. Strike direction is uniform on the large scale with some modest undulations on a stope mining scale;
- Vein true width generally ranging from 1.5m to 3.5m. On a small scale (metres) the vein can be highly variable in width and geotechnical properties;
- Continuity of the vein itself is good based on historic mining results. Grade continuity appears to be good on a stope mining scale, but mining grades are variable across the vein on a large scale;
- The Bellekeno deposit is characterized by one principal vein with some vein splays and the presence of sub-parallel veins located up to 300m into the hanging wall and footwall. These ancillary features have not been modeled in the latest resource estimate because of limited data. These features are not expected to have a significant effect on mineral resources or mineable tonnes;
- The vein is cross-cut by bedding plane faults, which predate vein emplacement. Reactivation of these structures has occurred resulting locally in variable levels of damage to the rock mass. Some of these structures may carry water;
- Expected ground conditions in the vein and along the vein contacts varying from good to very poor. The conditions can vary substantially over short distances (5m);
- The FW is often characterized by competent jointed quartzite, but can be weak in some areas. The HW varies from competent quartzite to a significant layer of weak quartz breccia with clay filled shear bands. These clay bands are expected to be present in most mineralised areas;
- In areas of weak HW and FW conditions the use of an underhand mining method may be a risk as a result of poor backfill cohesion to the sidewalls and a lack of confinement stress as a result of the shallow depth;
- Geological contacts at hanging wall and footwall can often be visually identified, but can be faulted or fractured contacts with fault gouge and breccias. Mineralization contacts are less clearly defined and are based on a combination of structure, vein mineralogy and metal grades;
- There is some historic evidence that the vein system is locally water bearing, and may need time to drain when accessed by mine development;
- Vein depths are shallow with stress not being a factor in mine planning.

In summary there is a high level of variability within the vein. For this reason, mining methods were assessed separately for each zone, considering geotechnical domains, and particularly focusing on the mineable portions of the 48 vein above cut off.

The structural evaluation completed by Mr. Otto, P. Geo. (refer to report section 4.1.3) has increased the knowledge base significantly as a result of studying fault mechanics and precursor conditions. The study has facilitated the overall understanding of the factors contributing to those areas with poor ground conditions and should enable the mine operator to avoid, anticipate or prepare for such conditions.

4.2.2 Study Mining Methods – Introduction

This section describes the mining assessment of the Bellekeno 48 vein. The planned mining methods described below support the estimate of mineable tonnes in the subsequent report section.

The factors that most significantly affected the choice of mining methods are:

- The high value of the resource, demanding a high mining recovery;
- The variable nature of ground conditions in the vein and the need for safety in the stopes along with predictable mining, keeping the excavation under control;
- Weak materials locally occurring along the vein contacts. This can potentially impact the stability of overhead cemented backfill in an underhand mining method;
- Flexibility to chase sinuos veins and vein splays;
- The narrow vein with variable geometry.

The main mining method planned is mechanized cut and fill in 3.5m cuts, with overhand sequencing. In addition, there may be limited application for some shrinkage stoping and/or longhole stoping.

Backfill will consist of a mix of development waste rock and dry (filtered) pyritic tailings from the planned processing plant. Cement will be added and the backfill will be placed in stopes by LHD.

4.2.3 Mechanized Cut and Fill – Overhand

Cut heights are planned at 3.5m. Total stope lengths average 60m, ranging from 25 to 100m. A central cross cut will access the cut near the center of its strike length to allow mining faces to be opened in two directions. Each face in the vein will be marked up by a geologist before drilling starts utilizing an electric/hydraulic single boom face jumbo. Load-haul-dump (“LHD”) size (and capacity) will vary depending on the average vein width in the stope, and cross cuts will be sized to accommodate the selected LHD. The two LHD sizes selected to handle most of the mining are:

- LHD 3.0m^3, capacity 6.7 tonne, unit width 2.23m;
- LHD 1.6m^3, capacity 3.5 tonne, unit width 1.48m.

Cross cuts averaging approximately 60m in length will be driven from existing and planned footwall ramp systems to the vein. They will be driven at -15 percent (“%”) and then back slashed to access successive cuts. Typically, a cross cut approaching the vein at -15% over a length of 35m will be back slashed three times, accessing a total of four cuts, advancing 14m vertically through the vein from the sill of the primary cross cut.

Remuck bays will be established at the intersection of the main ramp and the stope cross cut. Blasted vein material will be trammed by LHD from the face to the remuck bay at the ramp for later haulage by small diesel truck.

Stope blasting costs are estimated based on 80% anfo and 20% cartridge emulsion.

An auxiliary ventilation fan will pick up fresh air from the ramp for delivery into the stope by ventilation ducting.

Planned stope production rates are based on cycling an independent stope face every three shifts (11-hour shift basis). The overall production plan incorporates several available stope faces to provide mining flexibility.

Variable and sometimes difficult ground conditions are expected in the cut and fill stopes. A number of ground support classes have been established and they will be employed depending on ground conditions. Alexco's technical staff will direct supervisors and stope miners regarding the appropriate support class to be installed. In the weakest ground it will be necessary to shotcrete the back and walls as soon as possible after blasting (flash coat), and then install the bolting through the shotcrete.

The 48 vein is known to be locally water bearing to varying degrees, based on historic mining. It is expected that water outflows from the vein will diminish if given time to drain after mining intersects the vein at planned stope sill cut elevations. Allowing time for initial drainage will improve geotechnical stability, and such drainage time is included in the development and production schedule.

The underground mine will be supplied with dry filtered pyritic and non-pyritic tailings back hauled in returning haulage trucks from the processing plant which is located about 5 kilometres ("km") by road from the BK East mine portal. The dry tails will be stored in a centrally located cross cut before being transported to empty stopes for backfilling. The backfill will be a mix of the dry tails and development waste rock (typically, potentially metal leaching "PML"). Development waste rock will also be stored in a centrally located cross cut. Refer to Figure 8.

Cement will be delivered to the mine in bulk bags through the BK East decline. Cement grout will be prepared using a mobile slurry plant, and sprayed in metered amounts into the bucket of the LHD placing the fill material.

Some sills will be created where cut and fill mining will later come up from below. These sills will be filled with cemented waste rock with no dry tails mixed in to achieve the best strength. Over the life of mine, the average cement content of all backfill is estimated at 2% by weight.

Table 7 shows the life of mine planned quantities of development waste rock and dry tails to be used in backfill.

Table 7: Planned Quantities of Waste Rock and Dry Tails

Waste rock broken at development faces:				
Area	**Lateral tonnes**	**Raise tonnes**	**Total tonnes**	
SW	133,903	4,096	137,999	
99	49,003		49,003	
East	63,704	3,291	66,995	
Central	4,223		4,223	
Total broken	**250,833**	**7,386**	**258,219**	
Waste rock used as backfill:				
Area	**tonnes**	**distr.**	**volume m3**	**distr.**

SW	57,964		28,982	
99	14,798		7,399	
East	12,059		6,029	
Sub-total	**84,821**	43%	**42,410**	45%
Dry tailings used as backfill:				
Area	**tonnes**	**distr.**	**volume m3**	**distr.**
SW	71,784		33,113	
99	24,009		11,075	
East	18,044		8,323	
Sub-total	**113,837**	57%	**52,511**	55%
Total backfill	**198,658**	100%	**94,921**	100%

Backfill will be pushed up to the back of the 3.5m high cuts using an LHD equipped with a push plate commonly known as a backfill “jammer”.

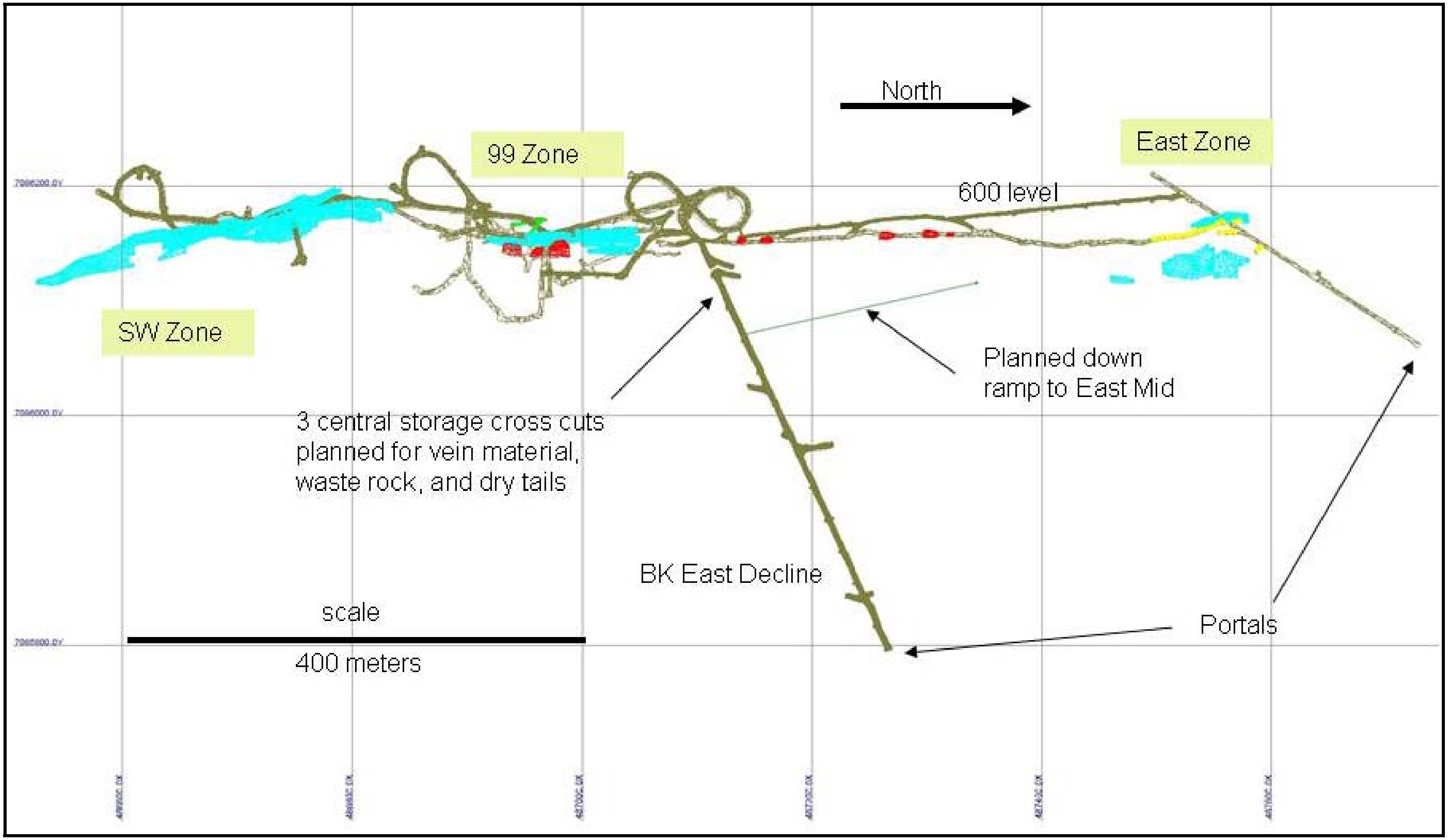


Figure 8: Plan View of Underground Mine

Figure 9 shows the general configuration for cut and fill mining in the SW and 99 zones where the average in-situ true vein width is 2.5m. Face drilling will be by electric/hydraulic single boom jumbo, and mucking will utilize a 1.6 m^3, 3.5 tonne capacity LHD (2 yard LHD).

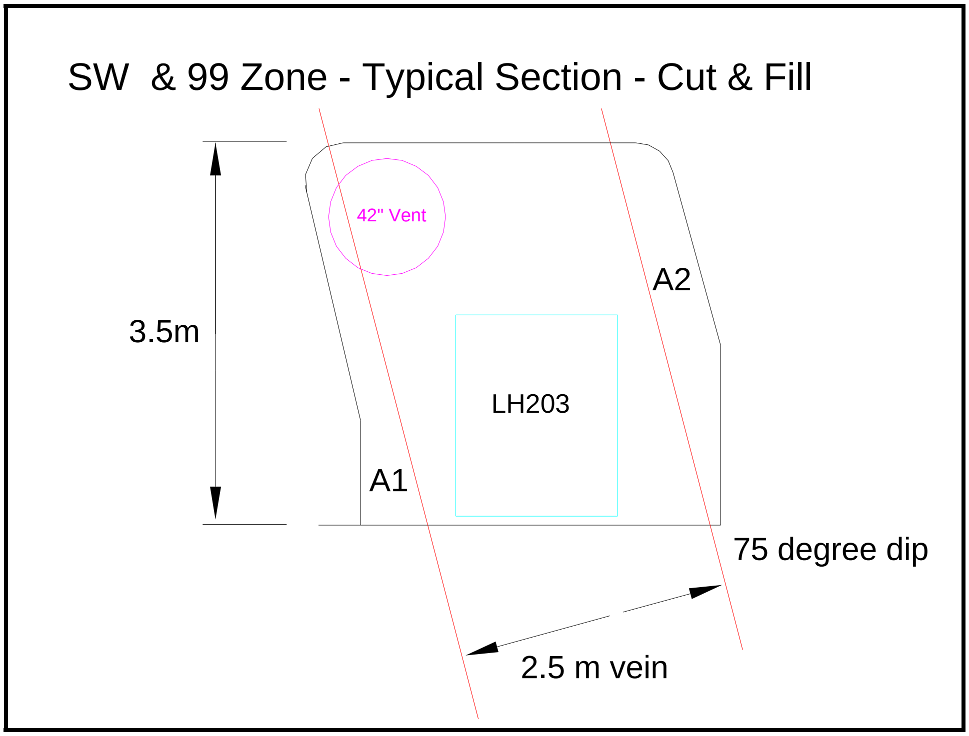


Figure 9: Typical Cut and Fill Cross Section

4.2.4 Mechanized Cut and Fill – Underhand

Underhand cut and fill was considered in the study. It would be an effective means of handling poor (weak) ground conditions in the vein – particularly in the back.

This method has been excluded from the current mine plan for Bellekeno 48 vein due to the risk of backfill instability created by the very weak rock that locally occurs along the vein contacts. In such conditions there is a risk that overhead cemented backfill could become unstable due to a shearing movement along the vein contacts.

4.2.5 Shrinkage Stoping

There may be limited application for shrinkage mining at Bellekeno in recovering the currently defined mineral resources.

In the East mining zone SRK defined a mining block situated above the existing 600 level. It is named 48 Upper and contains an estimated 14,100 tonnes mineable at an NSR value of $453 per tonne. The block extends 27m vertically above the existing level.
This block has an average in-situ true thickness of 1.9m. Mechanized cut and fill mining with footwall ramp access will work to recover this block, but a shrinkage method is likely more economic as the cost of the footwall ramp can be eliminated.

The current mine plan incorporates shrinkage mining for East 48 Upper block. Mined vein material will be trammed a short distance by LHD from the draw points on 600 level and transferred down a planned raise bore hole to the East Mid development planned below 600 level.

4.2.6 Longhole Stoping

There may be limited application for longhole mining at Bellekeno in recovering the currently defined mineral resources.

In 99 mining zone the uppermost (in elevation) mining block defined by SRK is situated above mine grid elevation 1078. It is named 99_B and contains an estimated 5,700 tonnes mineable at an NSR value of $377 per tonne. The block extends 27m vertically with an average in-situ true thickness of 1.6m.

Mechanized cut and fill mining with footwall ramp access is not attractive for this block due to its limited tonnage.

It is planned that this block be recovered using a longhole method, with raise access to two drilling subdrifts. The existing 99 timbered raise can be used for this purpose. An existing sublevel would be extended by 16m and a new subdrift 60m in length is also required.

Drilling will be by bar and arm (or small mobile drill) for lengths of approximately 10m. The mine design includes two ramp access draw points at the bottom elevation of 99-B block for mucking the vein material. Refer to Figure 10 .

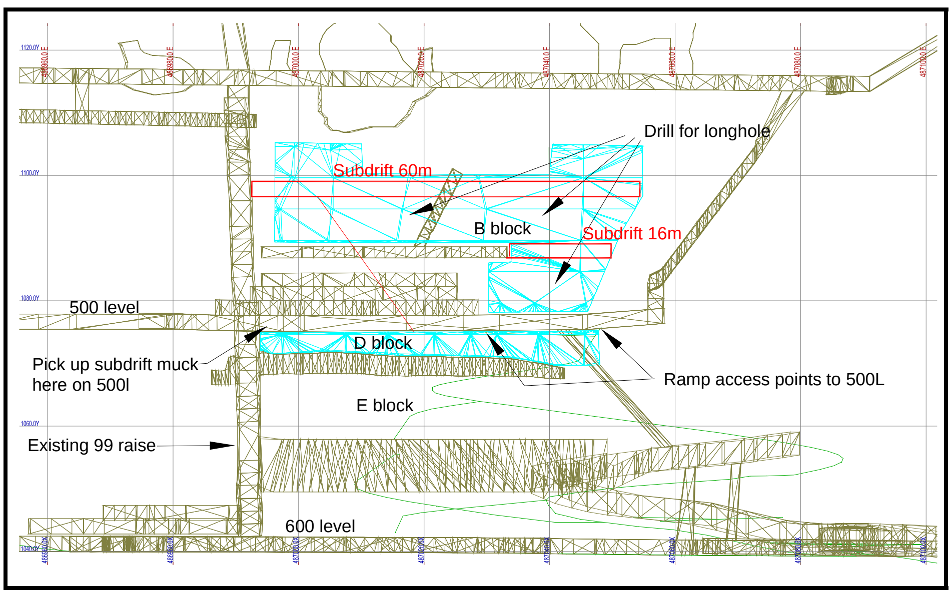


Figure 10: Longhole Method 99_B Block – Vertical Section

4.3 Estimate of Potentially Mineable Tonnes

4.3.1 Introduction

This section presents the methodology used to estimate potentially mineable tonnes for the study. The term "potentially mineable tonnes" is used because this Mining Study does not support the estimation of a Mineral Reserve.

The steps used in converting the mineral resources to potentially mineable tonnes are listed below:

- Estimation of a site operating cost;
- Selection of metal prices and exchange rate;
- Estimation of average external dilution, and metal grades in the dilution material;
- Assessment of minimum mining width and application to the target vein(s);
- Preparation of a net smelter return calculation;
- Determination of an economic cut off criteria; an NSR value (C$/tonne);
- Application of the cut off NSR value to the resource block model to identify areas above cut off;
- Creation of clipping polygons to outline practical mining shapes that enclose as much of the above cut off material as possible;
- Assessment of the mining shapes against the existing mine workings (previous stoping) and modification of the mining shapes as necessary due to nearby historic workings;
- Estimation of the in-situ tonnes and metal grades within each mining shape using block modelling software;
- Application of estimated external dilution percentages and dilution metal grades to the in-situ material, followed by application of mining loss factors;
- Initial economic assessment of isolated (or other) areas to verify they can carry planned development costs;
- Tabulation of the resulting tonnes and grades by mining shape. This data will feed into the production schedule.

These steps are described in the following sections.

4.3.2 Study Metal Prices and Exchange Rate

Alexco provided SRK with the metal prices to be used in the Bellekeno mine planning. The anticipated timing of metal production from SW and 99 zones occurs sooner than does the expected metal production from the East zone. For this reason, two sets of metal prices were prepared, both based on published metal price forecasts available in late August 2009.

The following prices were used for SW and 99 zone mine planning.

- Silver US$15.25/ounce;
- Lead US$0.675/pound ("US$/lb.");
- Zinc US$0.80/lb.

The following prices were used for East zone mine planning.

- Silver US$14.50/ounce;
- Lead US$0.60/lb;
- Zinc US$0.90/lb.

Alexco also provided an estimated exchange rate of $0.90 USD to $1.00 CAD for the mining study.

Just for reference, the current prices and exchange rate at the time of writing, early October 2009, are shown below.

The following prices and exchange are current as of October 5, 2009.

- Silver US$16.20/ounce;
- Lead US$1.00/lb;
- Zinc US$0.86/lb;
- $0.93 USD to $1.00 CAD.

The current prices are generally higher than the study prices. For the SW zone, the above set of prices and exchange would add over 14% to the average NSR value of the mineable vein material.

It is noted that there is some gold content to the mineable vein material, averaging 0.42 grams per tonne ("gpt") on a diluted basis. Gold does not make a significant contribution to the value of the vein material, and does not need to be considered in the mine planning NSR calculation.

4.3.3 External Dilution Estimate

Estimated values for external dilution were prepared in order to convert in-situ vein metal grades and NSR values to diluted plant feed values.

For this study SRK has defined external vein dilution as:

Dilution % = (dilution tonnes) / (dilution tonnes + vein tonnes) x 100.

For this project the in-situ vein material has a significantly higher average density than the wall rock. Dilution expressed as a percentage based on tonnes is lower than the same dilution expressed as a volumetric percentage. In the SW zone for example, a volumetric dilution of 25% is equivalent to a dilution of roughly 19% expressed on a tonnage basis.

Sources of vein external dilution that were considered include:

- Wall rock from hangingwall and footwall contacts. A layer of rock 0.70m thick was modelled to represent the sum of the wall dilution from both vein contacts. This material carries some very low metal grade;
- Backfill dilution from the floor. With the planned method of overhand cut and fill, there will be some mixing of mined vein material and backfill along the floor. A continuous layer of backfill 150mm thick is included as dilution;
- Some additional vein dilution was added to represent the additional dilution that will be experienced when mining advances along strike through zones of poor vein continuity.

“Extra” dilution will be experienced as the cut and fill mining progresses through areas of weak vein continuity. The mined grade will be reduced, but it will still be above cut off and the material will be part of plant feed. These types of occurrences represent an increase in external dilution beyond the provisions for wall rock and backfill described above.

Table 8 shows the estimates of external dilution by zone.

Table 8: Estimated External Dilution

Zone	In-situ True Vein Thickness (m)	Estimated External Dilution (%)
SW	2.50	23.0%
99	2.51	25.0%
East Upper	1.69	32.0%
East Mid	2.33	26.0%
East Deep	2.70	24.0%

For each zone, the estimate of external dilution was based on the mineable portion of the 48 vein wireframe. The true thicknesses shown represent the mineable portion of the vein. The average external dilution for the life of mine production plan is 25% on a tonnage basis.

The external dilution estimate used in the June 2008 Preliminary Economic Assessment was an average of approximately 14% on a tonnage basis. The average interpreted thickness of the mineable portions of the vein (wireframe) has decreased significantly (approximately reduced from 4.4m to 2.5m) since the 2008 PEA, based on additional 2009 drilling and geologic modelling.

Dilution metal grades estimates are shown in Table 9. These grades were based on a review of drill hole intercepts located just outside of the mineralized wireframes.

Table 9: Estimated Grade of External Dilution

Metal	Units	SW and 99 Zones	East Zone
Silver	opt	4	6
Lead	%	0.05	0.04
Zinc	%	0.40	0.31

4.3.4 Minimum Mining Width

It is planned to be able to achieve a horizontal mining width in the range of 2.2 meters for mechanized cut and fill mining using a 1.0m^3 LHD with a capacity of 2 tonnes.

The SW zone E block has an average in-situ vein width of 1.3m. It will be mined using a small LHD as shown in Figure 11, with cut height reduced to 2.7m. Face drilling will employ jackleg drills. For E block the external dilution was increased to 35%, comprised of 26% wall dilution shown in the Figure, 4% backfill floor dilution and an additional 5% external dilution allowance for areas of weak vein or grade continuity. The fully diluted NSR average for E block is $475 per tonne.

The application of a minimum mining width criteria has an insignificant impact on estimated mineable tonnes. No areas of vein above cut off NSR have been excluded on the basis of being too narrow in true thickness.

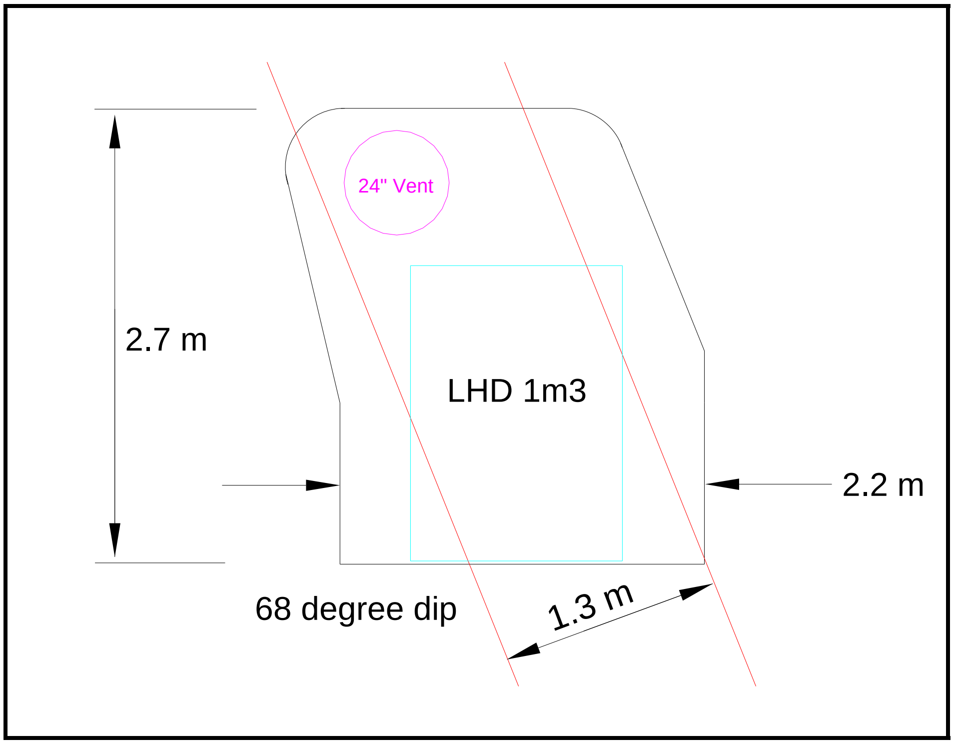


Figure 11: SW E block Mining Cross Section – Minimum Mechanized Width

4.3.5 Net Smelter Return Estimation

Mined vein material will be processed on site to produce two concentrates – a lead concentrate and a zinc concentrate. Key metals of economic interest are silver, lead, and zinc with gold making an insignificant contribution. Concentrates will be transported off site for smelting and refining.

Alexco provided SRK with three net smelter return ("NSR") models to be applied as follows:

- SW and 99 model;
- East Upper and East Mid model;
- East Deep model.

A number of input parameters vary among the models including metal prices (as previously discussed), concentrate grades and metal recoveries, and one penalty element.

SRK reviewed the models and added external dilution as an input item. The models take in-situ grades from the resource block models as input and yield diluted plant feed NSR values ($/tonne) as output.

Alexco has not finalized a concentrate uptake agreement at the time of SRK's mining study, but has provided TC/RC data based on recent discussions and proposals from a number of smelter agents.
SRK created NSR factors from the models that provide a quick way of assigning an NSR value to each bock in the resource block models. Refer to Table 10.

An example of applying the NSR factors to a set of in-situ metal grades is shown below for the SW and 99 models. The NSR formula is:

NSR ($/t) = 0.289(Ag gpt) + 6.08(Pb%) + 3.29(Zn%)

With the in-situ metal grades inserted the NSR formula becomes:

NSR ($/t) = 0.289(1314) + 6.08(16.7) + 3.29(5.26) = $498.59 per tonne plant feed

With the NSR entered as a model item it is possible to view the target vein on the basis of colour contoured NSR values.

Table 10: NSR Factors

	NSR Factors		
Zone	**Ag**	**Pb**	**Zn**
SW and 99	0.289	6.08	3.29
East Upper	0.237	3.47	4.24
East Mid	0.258	3.78	4.61
East Deep	0.250	4.09	4.32

4.3.6 Application of Cut-Off NSR and Creation of Mining Shapes

The economic cut off criteria was set equal to the selected estimate of site operating cost of C$230 per tonne processed (value rounded off). No capital costs were included in the cut off estimate.

Plots were prepared to show the 48 vein, and where appropriate the 49 vein, NSR values in vertical long section views. The colour legend for the NSR plots is:

- black $150 to $230 NSR ($/tonne);
- red $230 to $500 NSR;
- yellow $500 plus NSR.

The target colours for mining are red and yellow, representing vein material above cut off.

Figure 12 is the NSR plot for the SW zone. Figure 13 shows the planned cut and fill mining block outlines created for the SW zone.

Figure 14 and Figure 15 show the NSR plot for 99 zone and the planned mining blocks (highlighted red).

Figure 16 and Figure 17 show the NSR plot for the 48 vein in the East zone, and the planned 48 vein mining block outlines.

Figure 18 and Figure 19 show the NSR plot for the 49 vein in the East zone, and the single planned mining block (highlighted red).

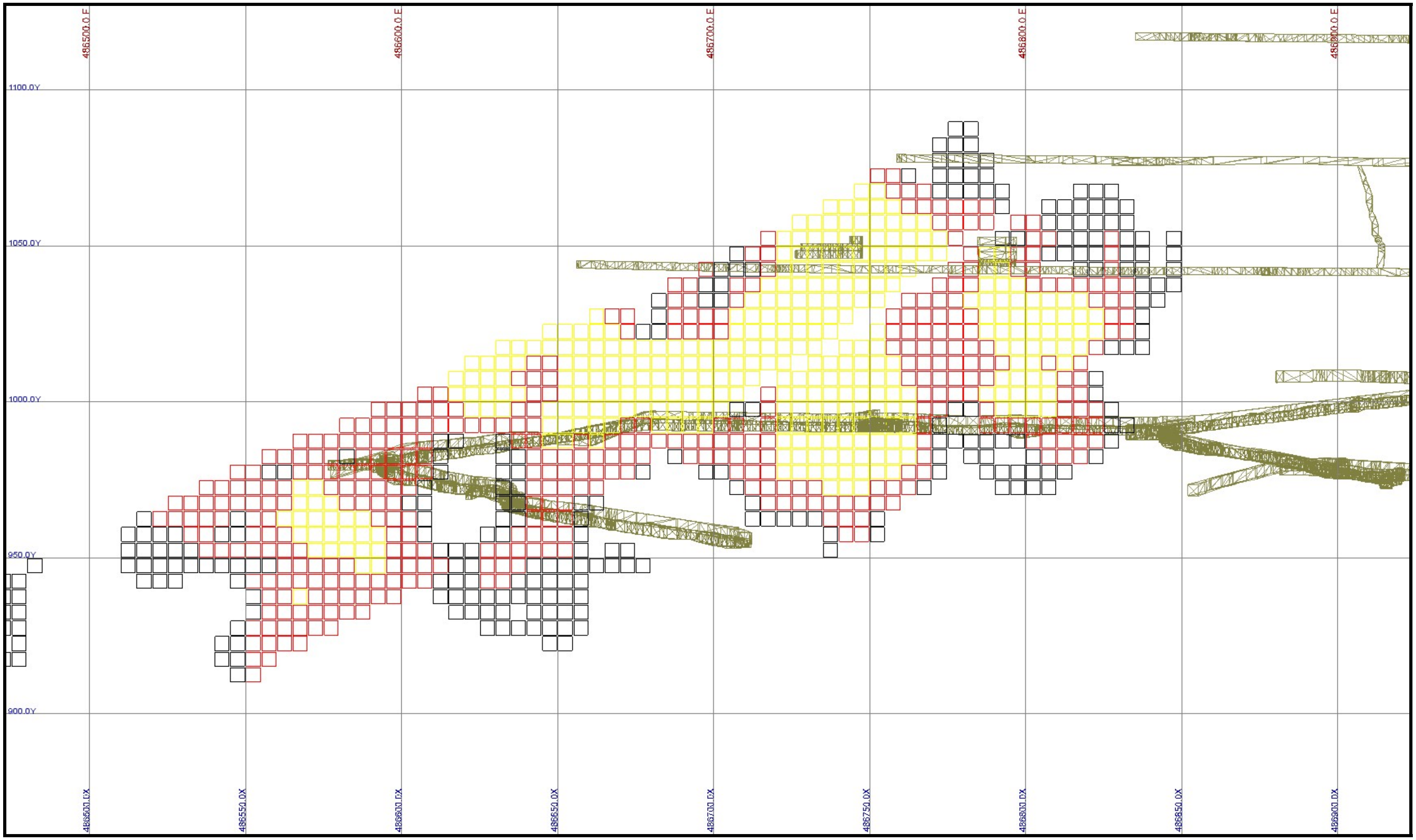


Figure 12: SW Long Section with NSR Block Values Colour Contoured – Looks West

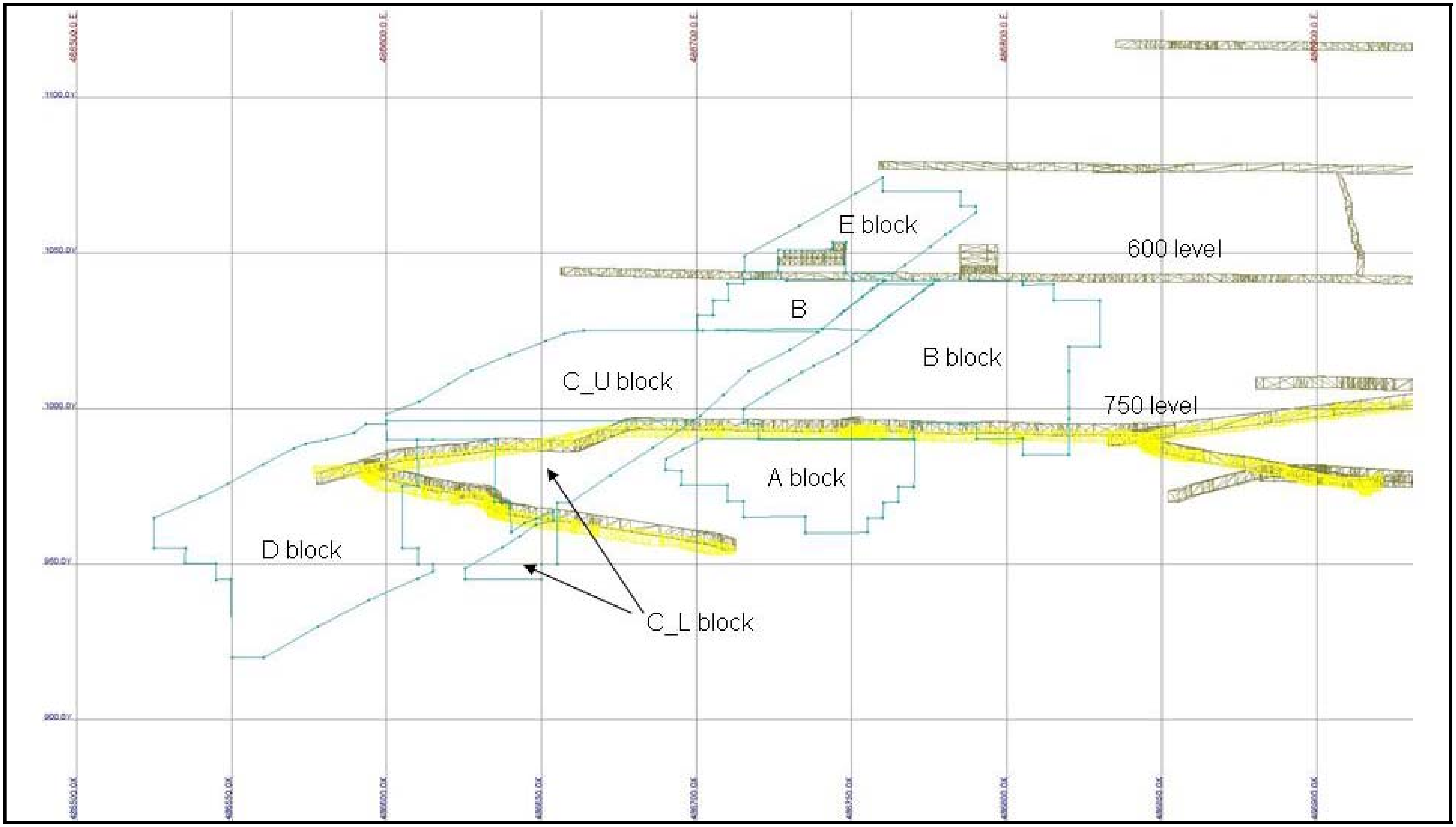


Figure 13: SW Long Section with Mining Blocks – Looks West

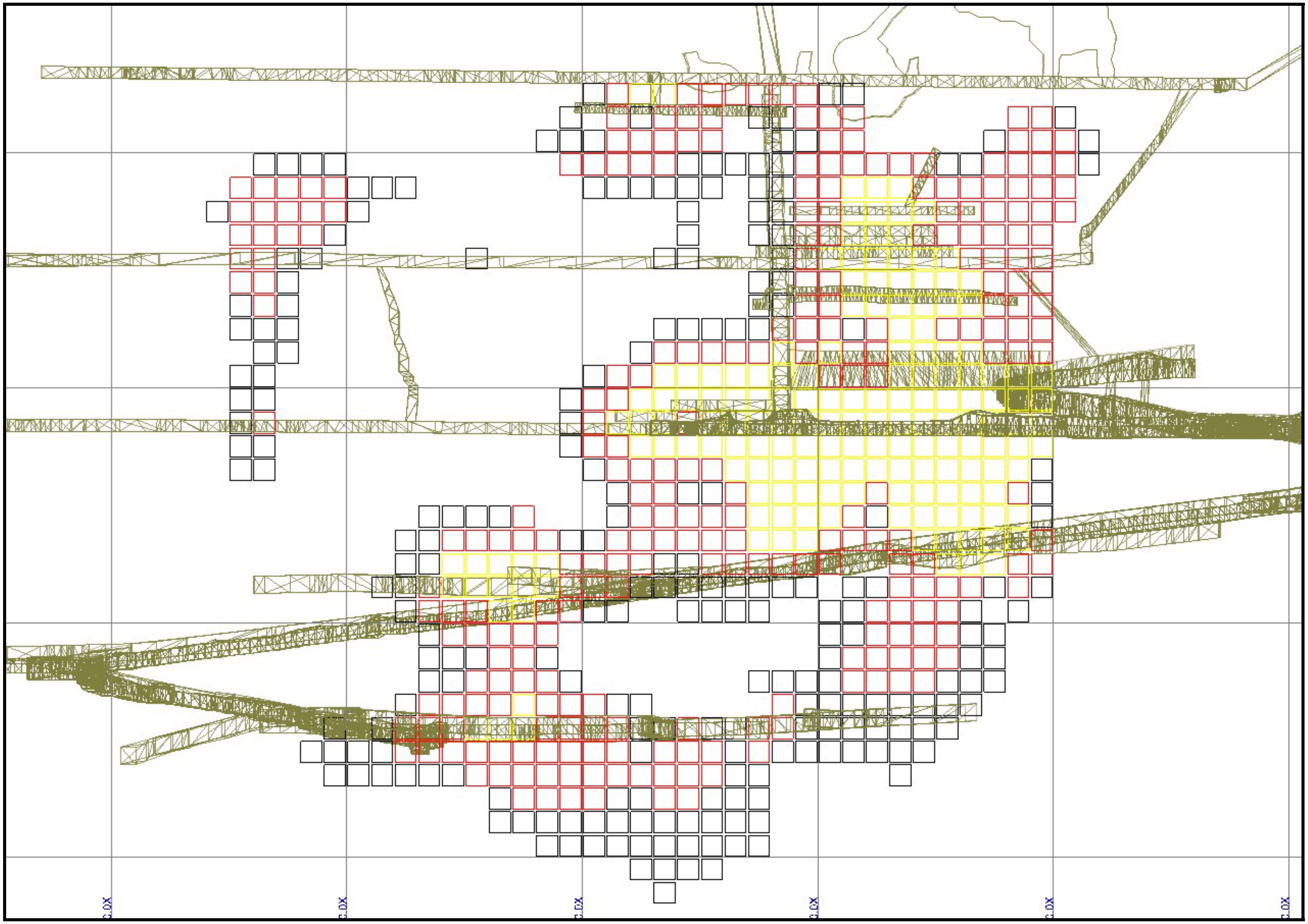


Figure 14: 99 Long Section with NSR Block Values Colour Contoured – Looks West

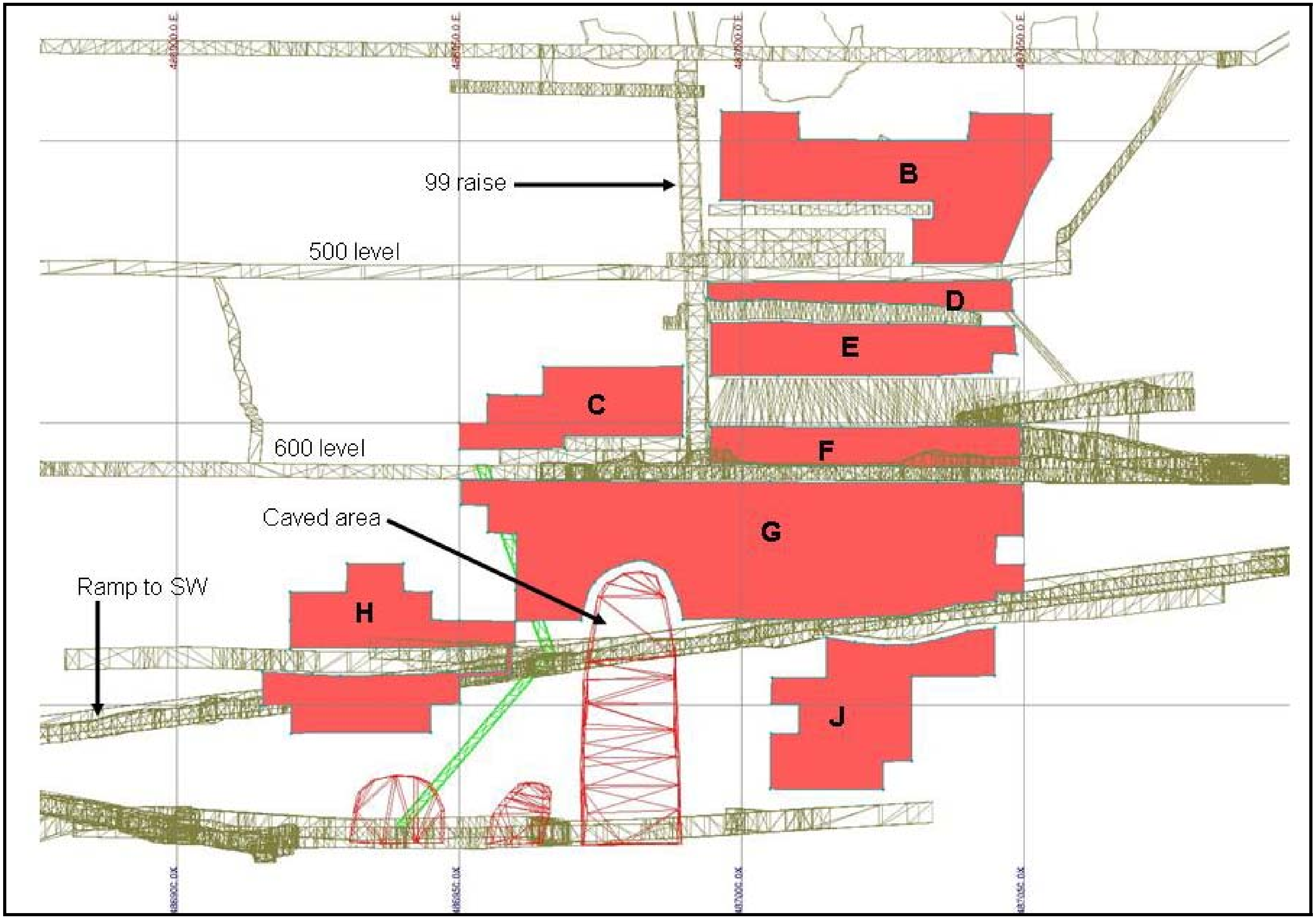


Figure 15: 99 Long Section with Mining Blocks (red) – Looks West

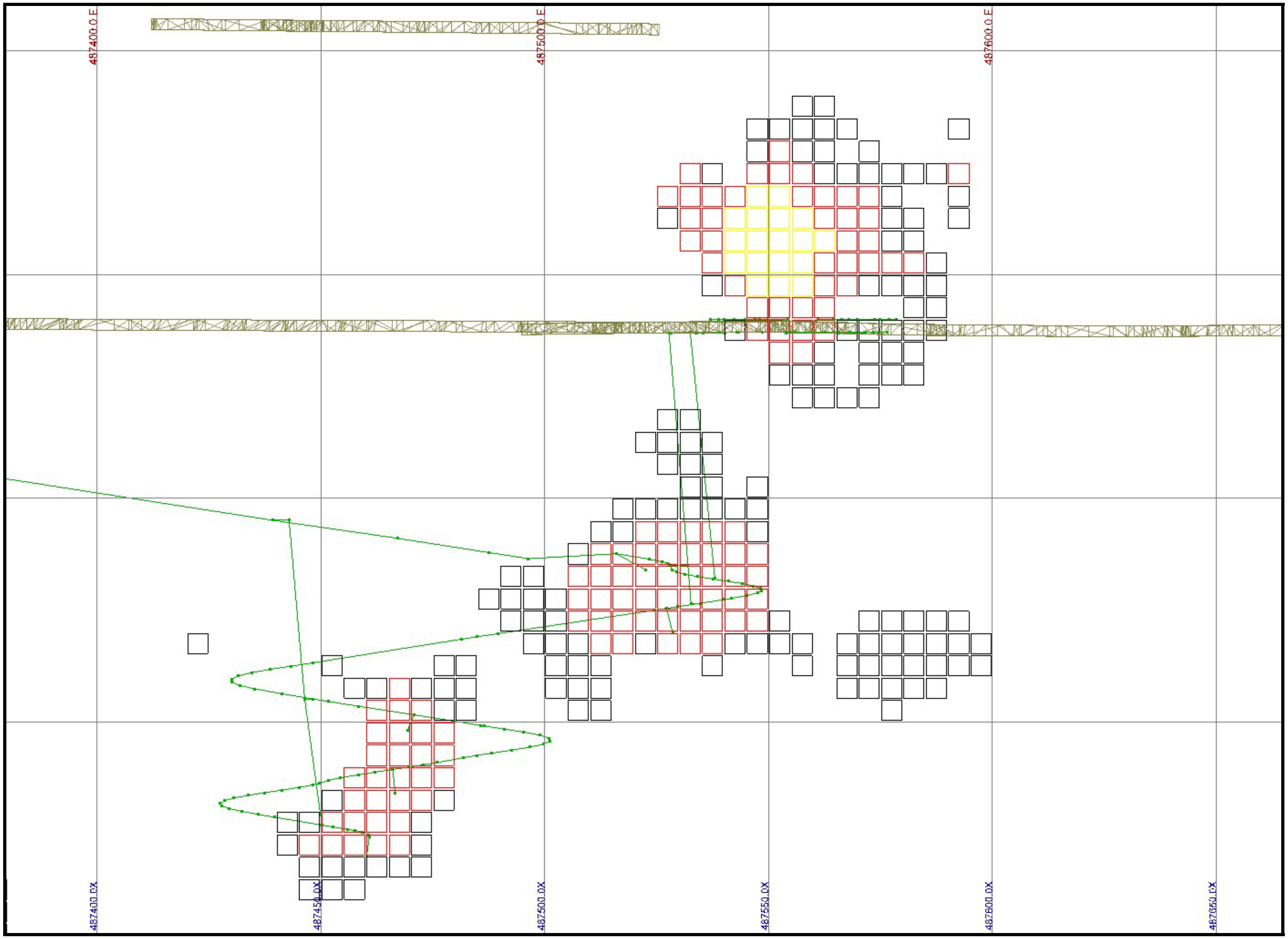


Figure 16: East 48 Long Section with NSR Block Values Colour Contoured – Looks West

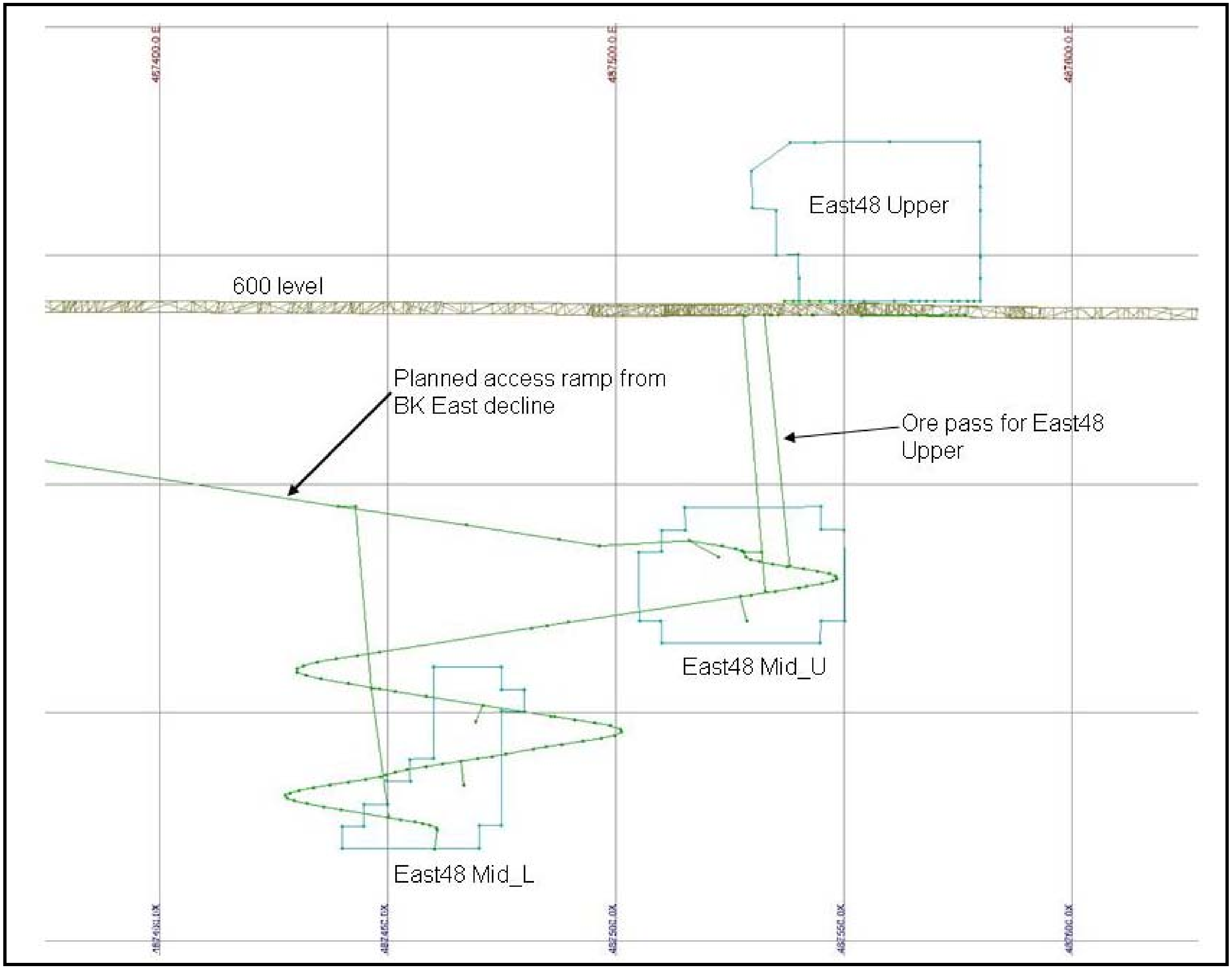


Figure 17: East48 Long Section with Mining Blocks – Looks West

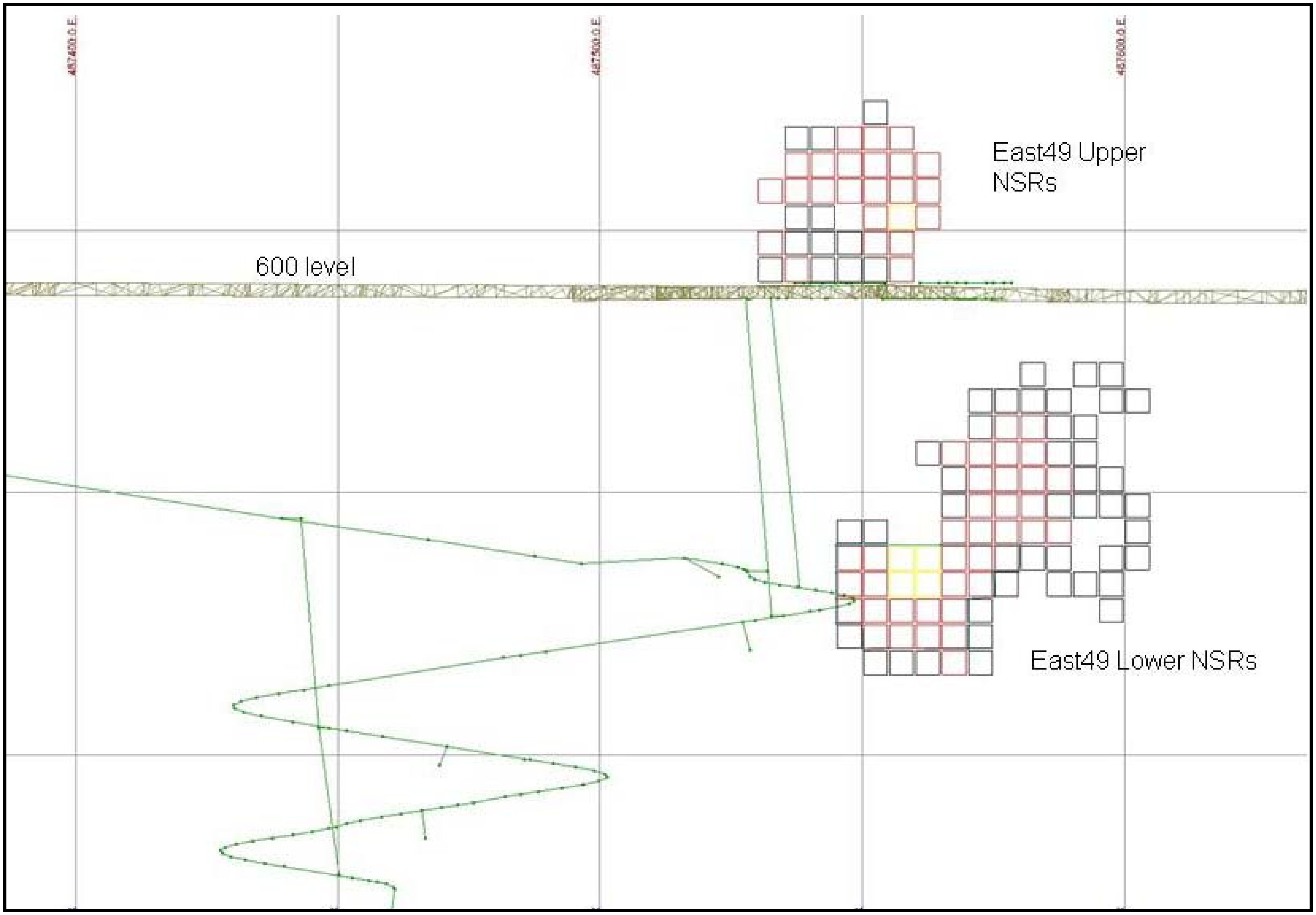


Figure 18: East 49 Long Section with NSR Block Values Colour Contoured – Looks West

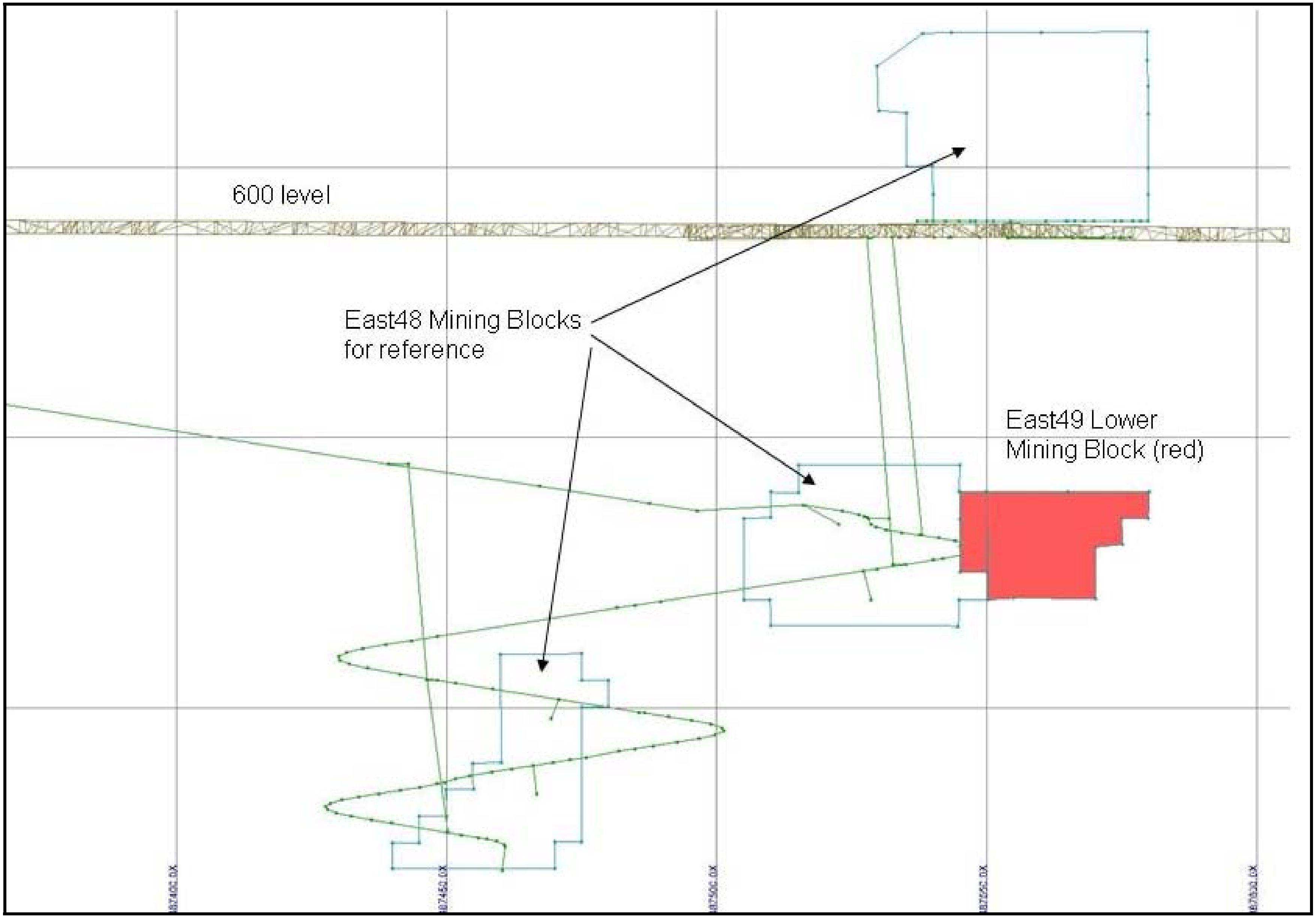


Figure 19: East49 Long Section with single Mining Block – Looks West

4.3.7 Estimated Mineable Tonnes

Table 11 shows the conversion of in-situ vein material in the mining blocks to potentially mineable tonnes. The Table provides a summary of the 17 mining blocks (also referred to as practical mining shapes) contained within SW zone, 99 zone and East zone. The individual mining block names and the mineable tonnage represented appear in the production schedule section of this report.

In general, a 95% mining recovery factor was applied for the cut and fill mining method planned.

For the 99 Zone, SRK applied a lower mining recovery factor because some of the planned mining will be near the existing workings and previously caved areas.

The E block in SW zone will also be mined next to an old stope, and its mining recovery was reduced.

In the East Upper zone a significant portion of 49 vein material which was above the NSR cut off, was judged to be uneconomic as it is very thin, and would require relatively expensive ramp access. This is reflected in the 76% mining recovery shown for that mining block.

The East Lower inferred mineral resources were excluded from the study because they are uneconomic.

Table 11: Conversion of In-Situ Tonnes to Potentially Mineable Tonnes

Mining Area	From Gemcom - In situ				External	External	Dilution Grade			Diluted	Mining	Mineable	Mined Grades			Diluted
	Tonnes	Ag	Pb	Zn	Dilution	Dilution	Ag	Pb	Zn	Tonnes	Recovery	Tonnes	Ag	Pb	Zn	NSR
SW Zone Blocks	(tonne)	(gpt)	(%)	(%)	(%)	(tonne)	(gpt)	(%)	(%)	(tonne)	(%)	(tonne)	(gpt)	(%)	(%)	($/t)
A	**23,563**	1227	15.39	8.24	24%	7,441	4.0	0.05	0.40	**31,004**	95%	**29,500**	933	11.7	6.36	$560
B	**56,978**	1272	13.23	7.43	24%	17,993	4.0	0.05	0.40	**74,971**	95%	**71,200**	968	10.1	5.74	$560
C_Upper	**35,311**	1317	20.45	6.88	24%	11,151	4.0	0.05	0.40	**46,462**	95%	**44,100**	1000	15.6	5.33	$618
C_Lower	**25,980**	819	12.13	9.02	24%	8,204	4.0	0.05	0.40	**34,184**	95%	**32,500**	623	9.23	6.95	$396
D	**25,781**	813	11.90	7.67	24%	8,141	4.0	0.05	0.40	**33,922**	95%	**32,200**	619	9.05	5.92	$388
E	**5,775**	1273	14.22	3.29	35%	3,110	4.0	0.05	0.40	**8,884**	90%	**8,000**	829	9.26	2.28	$475
Sub-total SW Zone	**173,400**	**1140**	**14.7**	**7.6**	**24%**	**56,000**				**229,400**	**95%**	**218,000**	**860**	**11.1**	**5.8**	**$519**
99 Zone Blocks																
B	**4,799**	978	3.61	1.39	24%	1,516	4.0	0.05	0.40	**6,315**	90%	**5,700**	744	2.75	1.15	$377
C	**3,907**	1268	7.65	2.10	24%	1,234	4.0	0.05	0.40	**5,141**	90%	**4,600**	965	5.82	1.69	$508
D	**1,152**	1462	7.36	2.76	24%	364	4.0	0.05	0.40	**1,515**	90%	**1,400**	1110	5.61	2.20	$578
E	**2,509**	1164	6.56	2.57	24%	792	4.0	0.05	0.40	**3,301**	90%	**3,000**	886	5.00	2.05	$466
F	**4,557**	2105	12.86	6.53	24%	1,439	4.0	0.05	0.40	**5,996**	90%	**5,400**	1600	9.78	5.06	$854
G	**23,008**	1589	14.00	6.37	24%	7,266	4.0	0.05	0.40	**30,274**	90%	**27,200**	1210	10.7	4.93	$675
H	**5,175**	876	6.998	2.13	24%	1,634	4.0	0.05	0.40	**6,809**	90%	**6,100**	667	5.33	1.71	$364
J	**4,049**	702	4.24	5.27	24%	1,279	4.0	0.05	0.40	**5,328**	90%	**4,800**	535	3.24	4.10	$295
Sub-total 99 Zone	**49,200**	**1380**	**10.3**	**4.7**	**24%**	**15,500**				**64,700**	**90%**	**58,200**	**1050**	**7.8**	**3.7**	**$572**
East Zone Blocks																
Upper 48 + 49	**12,252**	1265	3.93	11.42	34%	6,374	6.0	0.04	0.31	**18,626**	76%	**14,100**	895	2.86	8.78	$454
East_Mid_U	**15,435**	937	4.97	6.96	27%	5,709	6.0	0.04	0.31	**21,144**	95%	**20,100**	686	3.63	5.16	$345
East Mid_L	**9,356**	613	7.44	10.45	26%	3,287	6.0	0.04	0.31	**12,643**	95%	**12,000**	455	5.51	7.81	$271
Sub-total East Zone	**37,000**	**960**	**5.2**	**9.3**	**29%**	**15,400**				**52,400**	**88%**	**46,200**	**690**	**3.9**	**7.0**	**$359**
Total Project	**259,600**	**1160**	**12.5**	**7.3**	**25%**	**86,900**				**346,500**	**93%**	**322,000**	**870**	**9.5**	**5.6**	**$506**

Table 12: Reconciliation of Resource Contained Metal to Plant Feed

METAL RECONCILIATION - shows metal grades												
	Reported Resource $185				Mining Cut Off NSR Applied $230				Mining Block			
Indicated	In-situ (t)	Ag (gpt)	Pb (%)	Zn (%)	In-situ (t)	Ag (gpt)	Pb (%)	Zn (%)	In-situ (t)	Ag (gpt)	Pb (%)	Zn (%)
East 48 Ind.	97,236	603	3.78	7.69	28,732	1,010	5.62	10.42	32,505	945	5.31	10.09
East 49 all	24,325	643	4.21	2.09	6,501	994	5.25	3.27	4,538	1,101	4.80	3.77
99 all	91,724	995	7.46	4.16	70,283	1,262	9.60	4.21	49,156	1,379	10.30	4.74
SW all	221,177	986	12.45	7.14	172,615	1,154	14.82	7.63	173,388	1,139	14.66	7.56
Totals	**434,462**	**883**	**9.00**	**6.35**	**278,131**	**1,162**	**12.33**	**6.95**	**259,586**	**1,159**	**12.49**	**7.28**
	Recovery of resource metal to + $230 cut off →					**84%**	**88%**	**70%**				
					Recovery of + $230 metal to mining blocks →					**93%**	**95%**	**98%**
									Diluted (t)	**Ag (gpt)**	**Pb (%)**	**Zn (%)**
Diluted Plant Feed									**322,000**	**870**	**9.5**	**5.6**
Recovery of Mining Block Contained Metal to Plant Feed										**93%**	**94%**	**95%**
Recovery of Reported Resource Contained Metal to Plant Feed										**73%**	**78%**	**65%**

METAL RECONCILIATION - shows units of contained metal (thousands)												
	Reported Resource $185				Mining Cut Off NSR Applied $230				Mining Block			
Indicated	In-situ (t)	Ag (oz) 000's	Pb (lb.) 000's	Zn (lb.) 000's	In-situ (t)	Ag (oz) 000's	Pb (lb.) 000's	Zn (lb.) 000's	In-situ (t)	Ag (oz) 000's	Pb (lb.) 000's	Zn (lb.) 000's
East 48 Ind.	97,236	1,885	8,113	16,486	28,732	933	3,560	6,600	32,505	987	3,806	7,232
East 49 all	24,325	503	2,257	1,123	6,501	208	752	469	4,538	161	480	377
99 all	91,724	2,934	15,087	8,420	70,283	2,851	14,875	6,523	49,156	2,179	11,161	5,140
SW all	221,177	7,010	60,711	34,819	172,615	6,402	56,410	29,048	173,388	6,350	56,044	28,909
Totals	**434,462**	**12,332**	**86,168**	**60,849**	**278,131**	**10,393**	**75,597**	**42,640**	**259,586**	**9,677**	**71,492**	**41,658**
	Recovery of resource metal to + $230 cut off →					**84%**	**88%**	**70%**				
					Recovery of + $230 metal to mining blocks →					**93%**	**95%**	**98%**
									Diluted (t)	**Ag (oz)** 000's	**Pb (lb.)** 000's	**Zn (lb.)** 000's
Diluted Plant Feed									**322,000**	**9,000**	**67,200**	**39,700**
Recovery of Mining Block Contained Metal to Plant Feed										**93%**	**94%**	**95%**
Recovery of Reported Resource Contained Metal to Plant Feed										**73%**	**78%**	**65%**

Table 12 presents a reconciliation that compares contained metal in the stated Indicated Mineral Resource to:

- Contained metal within the above $230/t diluted NSR (mining cut off);
- Contained metal within the designed mining blocks;
- Contained metal within the plant feed.

The table information is presented in two ways. On the upper half of the table the reconciled tonnages show the metal grades. On the lower half of the table, the corresponding quantities of contained metal are shown. In Table 12 above, the Indicated Mineral Resource matches the figures shown in Table 1.

Table 12 shows a significant reduction in contained metal in comparing the Indicated Mineral Resource to the vein material above the $230 NSR mine planning cut off.

The mine planning cut off NSR estimate accounts for average external dilution of 25% whereas the resource NSR estimate does not. For example, this reduces the NSR value of each block in the resource models by an average 25%. In addition, the mine planning NSR cut off applied was set at a higher value than the resource reporting cut off, $230/tonne for planning versus $185/tonne for resource reporting.

Table 13 also provides a measure of the efficiency of the mining block designs. It shows that not all of the above cut off material (+$230/tonne material) was included in the mining blocks. For example, the mining blocks include 92% of the silver metal that is above the mining cut off. Further optimization of the mining plan will address and possibly improve this efficiency.

An analysis of the data shown in the table reveals the following:

- Metal recovery of + $230 material into the mining blocks is very good for SW zone and for East Indicated (East Upper plus East Mid). The metal recoveries are close to 100%;
- For 99 zone and East 49, the recoveries are much lower, in the mid to upper 70s.

The +$230 NSR areas that were excluded were judged to be uneconomic and typically involved small groups of isolated blocks, areas of very thin vein, areas affected by previous mining or caving in 48 vein, or areas with NSR values only modestly above cut off.

In Table 12, the plant feed contained metal can be easily interpreted to be less than the contained metal in the mining blocks due to the application of mining recovery factors that averaged 93% on a tonnage basis.

4.3.8 Planned Production Rate

The June 2008 PEA technical report incorporated an initial production rate of 250 tpd, increasing to 400 tpd, with a total mineable tonnage of 614,000 tonnes.

The current estimate of mineable tonnes is 322,000 tonnes. This will support a mining rate of 250tpd for 3.6 years. SRK prepared the production schedule on the basis of 250tpd. The achievability of this mining rate was validated by considering stope cycle times and the number of independent mining areas as described below.

The approach taken was to estimate stope mining cycle times to determine the average production rate that a typical stope would yield over its entire life and to estimate how many stopes would have to be available to reliably achieve the planned production rate. Refer to Table 13.

Table 13: Stope Cycle Model for Overhand Cut and Fill

Stope Data	Units	Typical
Vein width, undiluted	m	2.5
Height of cut	m	3.5
Complete cut length	m	60
Width, diluted	m	3.2
Density, diluted	t/m3	3.37
Tonnes/Cut	t	2,263
Tonnes per blast	t	91
Mined void - total	m3	672
Backfill:		
Waste rock fill needed	t	605
Dry tails needed	t	809
Production phase		
Face shifts per blast	#	3
Independent faces	#	1.5
Tonnes per day during breasting	tpd	91
Cut breasting duration	day	25
Backfill phase		
Strip services in stope	day	1
Backfilling duration	day	4
Backfill fill cure time	day	7.0
Sub-total duration per cut	day	37.0
Unexpected delay	%	10%
Total duration per cut	day	**41**
Average stope production rate	**tpd**	**56**
Planned Production rate	tpd	250
Stopes cycling needed	#	4.5
Contingency	%	30%
Available stopes required	**#**	**5.8**

Table 13 shows a typical case for a 2.5m in-situ vein width. The Table indicates that 5 to 6 stopes should be available throughout the mine life, depending on the mining widths. A 30% contingency is included in the estimate to ensure there will be flexibility in the production plan. Based on the available work places in the mine layout, this level of stope availability has been scheduled, and the 250tpd rate is expected to be achievable.

Alexco intends to operate the mine using an extended shift schedule utilizing two 11 hour shifts per day, seven days per week. This schedule has been used in the current mining study.

4.4 Underground Mine Model

4.4.1 General Description

Figure 20 is a 3D view of the existing Bellekeno mine workings, looking at the mine from the footwall side, showing the names and relative locations of the three mining zones discussed in this report. Historical stopes are visible as outlines in the area of 200 level and 400 level. Currently planned mining solids (portions of 48 vein) are shown in cyan.

The mine access was historically via the 600 level portal and the 600 level track drift (2.7 x 2.4m, W x Ht). Refer to Figure 21. During past mining, exhaust ventilation air was routed up through ventilation raises and out to surface via 200 level. There has been some caving on 200 level, just in from the portal, which has been inspected recently from surface and from underground. This area is planned for rehabilitation which will re-open 200 level to surface for the purpose of exhaust ventilation.

In 2008, Alexco undertook the development of a new trackless 618m decline to access the central area of the Bellekeno mine, referred to as the BK East decline. This new decline provided access for dewatering, significant rehabilitation of existing workings, and diamond drilling. This work program started in late 2008 and was completed in mid-2009.

It is significant to note that there is currently safe and useable mine access through both 600 level portals shown in Figure 21, and also the historic internal trackless ramp system is in good condition, having been rehabilitated during 2009.

All of the levels above 600 level, up to and including 200 level, have been accessed internally by manway raises and inspected during 2009. In many areas, such as in the previous square set mining area at 500 level, timber has been preserved due to high moisture levels and lack of air movement.

4.4.2 Underground Mine Access

The main access to the mine is the new BK East decline with nominal dimensions of 4.6 x 4.6m and also by the historic 600 level portal and track drift with dimensions of 2.7 x 2.4m (W x Ht).

The existing ramps that have been incorporated into the mine plan are nominally sized at 3.0 to 3.2m in height with an average width of 3.7m. Tight sections of these ramps may be slashed out to accommodate a 15 tonne class underground truck, or alternately, smaller trucks may be used. New ramp development is planned at dimensions of 3.7 x 3.7m. The main areas of new footwall ramp development are shown in Figure 21 in dark red.

Ventilation options that have been considered all involve fresh air entering the BK East decline and exhausting by one of three possible routes:

- Out through the 600 level and 600 level portal (currently available);
- Exhaust via a new Alimak ventilation raise driven to surface (new raise required);
- Exhaust via the historic route – raises and out to surface on 200 level (rehabilitation needed).

These are discussed further in the ventilation section. These routes are also the secondary escapeway options for the mine.

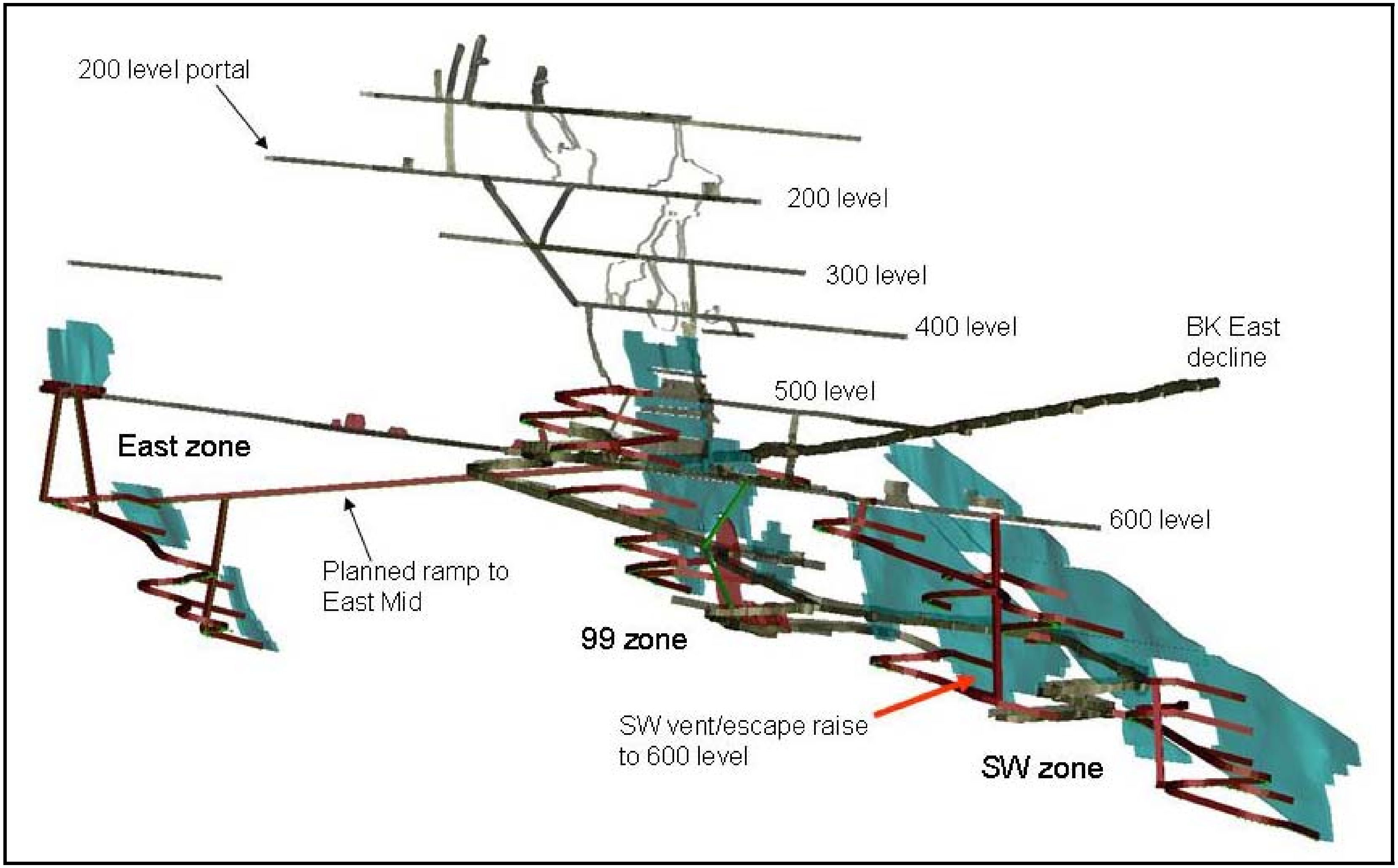


Figure 20: 3D View of Mine Workings looking NE from the footwall side

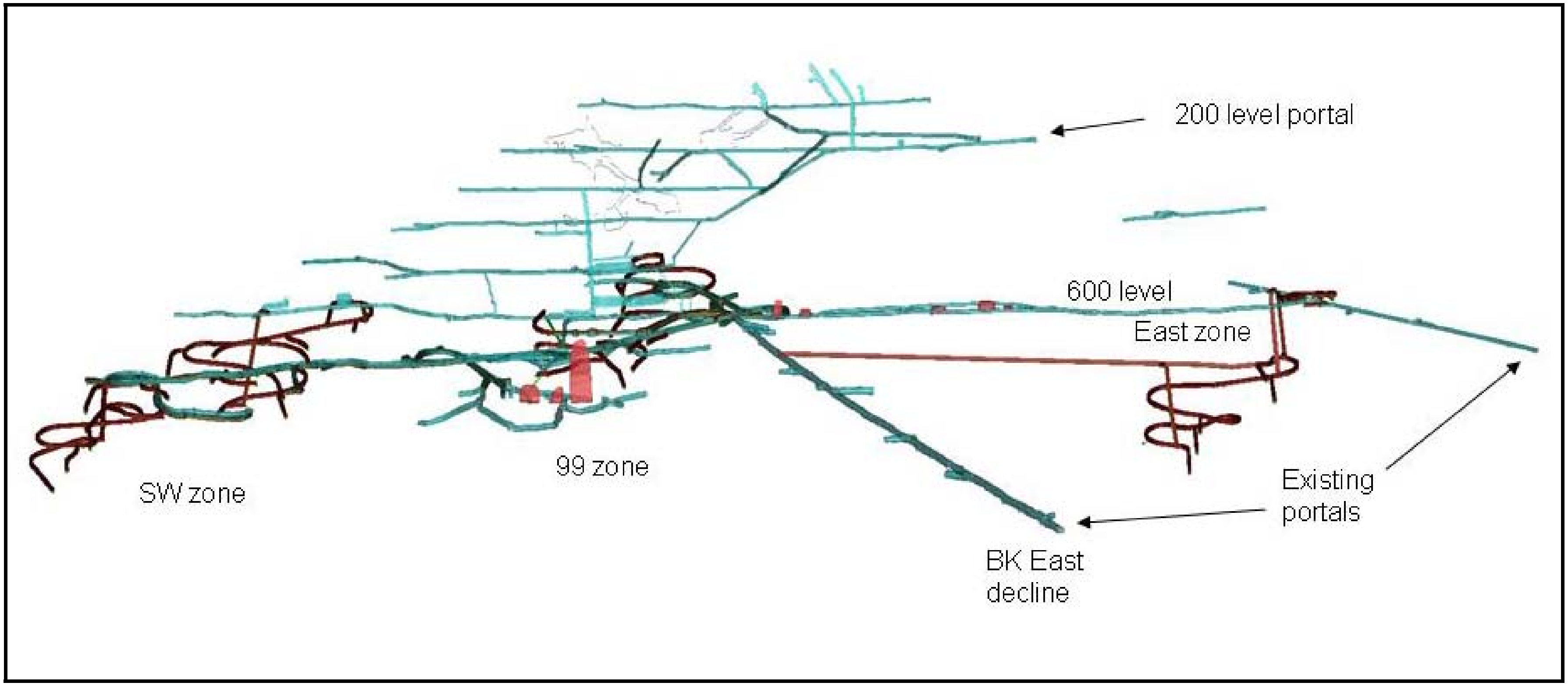


Figure 21: 3D View of Existing Mine (blue) showing Planned Development Areas (dark red)

4.4.3 Access for Stoping

The primary mining method planned is mechanized overhand cut and fill. Individual stopes will be accessed by cross cuts that branch off from the existing and planned extensions of the footwall ramps. The primary cross cuts are shown in Figure 14 and Figure 15.

In the SW Zone, cross cuts will access stopes with lengths of nominally 70m. Average primary cross cut length is in the range of 55 to 60m. The cross cuts are designed to approach the vein at a gradient of -15% such that back slashing will access up to 3 additional cut elevations. The cross cuts are planned at dimensions of 3.7 x 3.7m.

A remuck bay is planned at each intersection of a primary cross cut and the main access ramp. They are sized to allow a 6 tonne LHD load a small underground truck.

4.4.4 Life-of-Mine Lateral Development Requirements

The Gemcom mine model was used to estimate lateral and raising development requirements.

Life-of-mine (“LoM”) lateral development planned is shown in Table 14. Waste advance, which includes back slashing for cut & fill access, totals 5,887m. Waste rock tonnage from lateral development is estimated at 251 thousand tonnes, which will be partly consumed as backfill during the mine life.

Table 14: LoM Lateral Development Requirements

Main Access Ramps	Width (m)	Height (m)	Length (m)	Waste (t/m)	Waste (tonne)
SW Zone	4.0	4.0	1,072	43	45,586
99 Zone	4.0	4.0	99	43	4,204
East Zone	4.0	4.0	928	43	39,449
Sub-total ramps			**2,099**		**89,239**
Cross Cuts					
SW Zone	3.7	3.7	2,078	43	88,318
99 Zone	3.7	3.7	1,054	43	44,799
East Zone	3.7	3.7	571	43	24,255
Sub-total cross cuts			**3,702**		**157,371**
Infrastructure					
Tails Remuck	4.0	4.0	25	50	1,242
Other	4.0	4.0	60	50	2,981
Sub-total infrastructure			**85**		**4223**
Total lateral meters			**5,887**		**250,833**

Based on LoM production of 322,000 tonnes of vein material, the following key ratios have been estimated:

- 55 mined vein tonnes/meter waste development;
- 0.80 waste tonne/vein tonne (includes raise waste).

These are typical values for narrow vein mining projects.

The Bellekeno mine plan does not include any development on vein. The only advance on vein will be within the cut and fill stopes and it is reported as production in units of tonnes.

For the purpose of comparison only, the LoM cut and fill face advance on vein is estimated at 7,745m.

4.4.5 Life-of-Mine Raising Requirements

LoM raising requirements are shown in Table 15.

Table 15: LoM Raising Requirements

Raise Description	Width (m)	Height (m)	Length (m)	Waste (t/m)	Waste (tonne)
SW Vent Raise 800-600L	2.4	2.4	84	17.9	1,502
SW Vent Raise D block	2.4	2.4	45	17.9	805
East Mid Vent Raise	2.4	2.4	60	17.9	1,073
East Mid Vent Raise	2.4	2.4	69	17.9	1,234
East Upper - mining	2.4	2.4	70	vein	0
East Upper - Rock Pass	2.4	2.4	55	17.9	984
Local ventilation raises	2.4	2.4	100	17.9	1,788
Sub-total raises			**483**		**7,386**

The first four raises listed in Table 15 require manways. The line item “East Upper – mining” covers two timbered raises that will provide access to the shrinkage stope planned for East Upper mining block.

The local ventilation raises planned will create flow through connections between stope access cross cuts in key areas, reducing the dependence on auxiliary ventilation fans and ducting. These raises have not been designed, but they are included in the operating cost estimate.

4.5 Development and Production Schedule

4.5.1 Introduction

In order to create an accurate life-of-mine (“LoM”) production schedule for the Bellekeno mine, the period of ramp up to full planned production rate was first scheduled in detail. SRK developed a Gantt chart (schedule) representing the mine tasks that must be completed during the pre-production period. Refer to Figure 22. The pre-production period has been defined as:

Starts – When the independent project consultant (“IPC”) completes his review of the development plan and provides a positive report. This would trigger release of funding to start the capitalized work in the underground mine.
Ends – When the underground mine achieves “Initial Completion” which is 30 days of production mined and milled at a rate of 250tpd. The target for the 30 days is no later than June 2010.

For the purpose of the mine plan, SRK has assumed a 9 month pre-production period from Q4 2009 through Q2 2010.

In creating the Gantt chart (using Microsoft Project), it was assumed that:

- A positive report from the IPC would be received by Alexco by the end of October 2009. Work on pre-production tasks would then begin in the mine;

- Regarding the mill construction schedule, the latest version of the project master schedule indicates that mill commissioning will occur from mid-June to mid-July 2010;
- The planned lateral development advance rates generally do not exceed a line advance rate of 4m per day. Many development tasks are scheduled at lesser rates;
- Lateral development will be performed by a contractor;
- Initial mining will start in SW zone and 99 zone.

Objectives of this work:

- Schedule the key tasks to be completed in the mine such as rehabilitation, lateral development, ventilation raising, construction work, etc;
- Determine how many months will be required to complete pre-production tasks;
- Estimate the lateral development schedule, and the total meters per month to be achieved. Try to minimize and smoothen the development required per month;
- Determine the critical path activities;
- Determine capital requirements to complete the pre-production development work.

SRK prepared the LoM production schedule in Excel, covering the full mine life. Refer to Table 16. Its purpose is to define the sequencing of each stoping block, respecting the dependencies among them, and to schedule the tonnage and grade mined for the life-of-mine. The front end of this Excel schedule was defined by the pre-production Gantt chart.

Both schedules discussed above were based on continuous work, with two 11-hour shifts per day, seven days per week.

4.5.2 Contractor Involvement

Alexco will employ a mining contractor to perform the mining at Bellekeno. The contractor will provide labour and equipment and some supplies. Alexco will provide most of the mining supplies, and the technical services. Alexco personnel will also undertake surface trucking for the haulage of mined vein material to the mill, and development waste rock to the surface storage pile.

Limited in-fill diamond drilling and possibly some exploration drilling may be performed by a separate contractor.

This operating scenario impacts certain aspects of the schedule such as:

- Contractor mobilization time must be allowed for;
- Recruitment, build up, and training of Alexco’s own workforce is not required;
- Required productivities for development and mining will be achieved more quickly using a mining contractor;
- Lead times for purchasing key mining equipment are not an issue for Alexco.

ID	Task Name	Duration	Start	Finish
1	EXHAUST OUT 200L and 600 ADIT	0.03 mons?	Sep 1 '09	Sep 2 '09
2	**SLW Deal Key Milestones**	**8.63 mons**	**Oct 15 '09**	**Jul 1 '10**
3	Development Plan Completed	0 wks	Oct 15 '09	Oct 15 '09
4	Plan Review by IPC	2 wks	Oct 15 '09	Oct 29 '09
5	Positive Report from IPC	0 wks	Oct 29 '09	Oct 29 '09
6	Production at 250tpd for 30days	1 mon	Jun 1 '10	Jul 1 '10
7	Initial Completion	0 wks	Jul 1 '10	Jul 1 '10
8	**Mining Contractor**	**0.93 mons**	**Oct 29 '09**	**Nov 26 '09**
9	Award, Mobilize to Site	4 wks	Oct 29 '09	Nov 26 '09
10	**Mine Preparation Work**	**9.9 mons**	**Nov 29 '09**	**Sep 22 '10**
11	**Rehab Work**	**1.4 mons**	**Dec 20 '09**	**Jan 31 '10**
12	625 S. Rehab - Remove Timber	6 wks	Dec 20 '09	Jan 31 '10
13	**Down Ramp to SW 750L**	**9.2 mons**	**Dec 20 '09**	**Sep 22 '10**
14	Slash if needed, Pipe work, Road work	4 wks	Dec 20 '09	Jan 17 '10
15	99 Cave By-pass, 80m	3 wks	Sep 1 '10	Sep 22 '10
16	SW Zone By-pass, 36m	2 wks	Sep 1 '10	Sep 15 '10
17	**99 Cave Near Ramp**	**1.4 mons**	**Nov 29 '09**	**Jan 10 '10**
18	Vent Curtain in HW Drill Sta	3 days	Nov 29 '09	Dec 2 '09
19	99 Cave Test Holes	1 wk	Jan 3 '10	Jan 10 '10
20	**Lateral Development**	**47.7 mons**	**Nov 26 '09**	**Oct 27 '13**
21	**SW Zone**	**45.97 mons**	**Jan 17 '10**	**Oct 27 '13**
22	SW A-blk Ramp, 66m	2.4 wks	Jan 17 '10	Feb 3 '10
23	SW A-blk Ramp, 121m	12 wks	Apr 20 '10	Jul 13 '10
24	SW BC-blk Ramp 86m	3.1 wks	Apr 20 '10	May 12 '10
25	SW BC-blk Ramp, 58m	4.5 wks	Jun 11 '10	Jul 12 '10
26	SW BC-blk Ramp, 179m	13 wks	Jul 12 '10	Oct 11 '10
27	SW D-blk Ramp, 300m	22 wks	Jul 1 '10	Dec 2 '10
28	SW E-blk Ramp, 146m	12 mons	Nov 1 '12	Oct 27 '13
29	**99 Zone**	**9.92 mons**	**Nov 26 '09**	**Sep 20 '10**
30	99 FW Ramp Extn, 99m	7.1 wks	Aug 1 '10	Sep 20 '10
31	99 C-blk cross cut, 50m	4 wks	Dec 20 '09	Jan 17 '10
32	BT round at HW Drill Sta. Access	3 days	Nov 26 '09	Nov 29 '09
33	**East Zone Mid**	**7.47 mons**	**Mar 1 '12**	**Oct 11 '12**
34	East Ramp1, 449m	14 wks	Mar 1 '12	Jun 7 '12
35	East Ramp2, 95m	3 wks	Jun 14 '12	Jul 5 '12
36	East Ramp3, 384m	12 wks	Jul 19 '12	Oct 11 '12
37	East Upper Ore Pass, 55m	5 wks	Jul 5 '12	Aug 9 '12
38	**Ventilation Work**	**36.93 mons**	**Nov 12 '09**	**Nov 24 '12**
39	**Install Mine Air Heater**	**0.93 mons**	**Nov 12 '09**	**Dec 10 '09**
40	Propane Heater at BK East Decline	4 wks	Nov 12 '09	Dec 10 '09
41	**East Vent Circuit Thru 625 Adit**	**4.73 mons**	**Jul 5 '12**	**Nov 24 '12**
42	**EM1 Raise, 60m**	1 mon	Jul 5 '12	Aug 4 '12
43	**EM2 Raise, 69m**	1 mon	Oct 11 '12	Nov 10 '12
44	Install Fan 2A	2 wks	Aug 4 '12	Aug 18 '12
45	Install Fan 2B	2 wks	Nov 10 '12	Nov 24 '12
46	**SW Ventilation Circuit**	**11.94 mons**	**Jan 27 '10**	**Jan 20 '11**
47	Raise SW1 cross cut, 16m	1.5 wks	Jan 27 '10	Feb 6 '10
48	**SW1 Raise, 84m**	9 wks	Feb 6 '10	Apr 10 '10
49	Install SW Fan 5	10 days	Apr 10 '10	Apr 20 '10
50	**SW2 Raise, 45m**	7 wks	Nov 18 '10	Jan 6 '11
51	Install SW Fan 4	2 wks	Jan 6 '11	Jan 20 '11

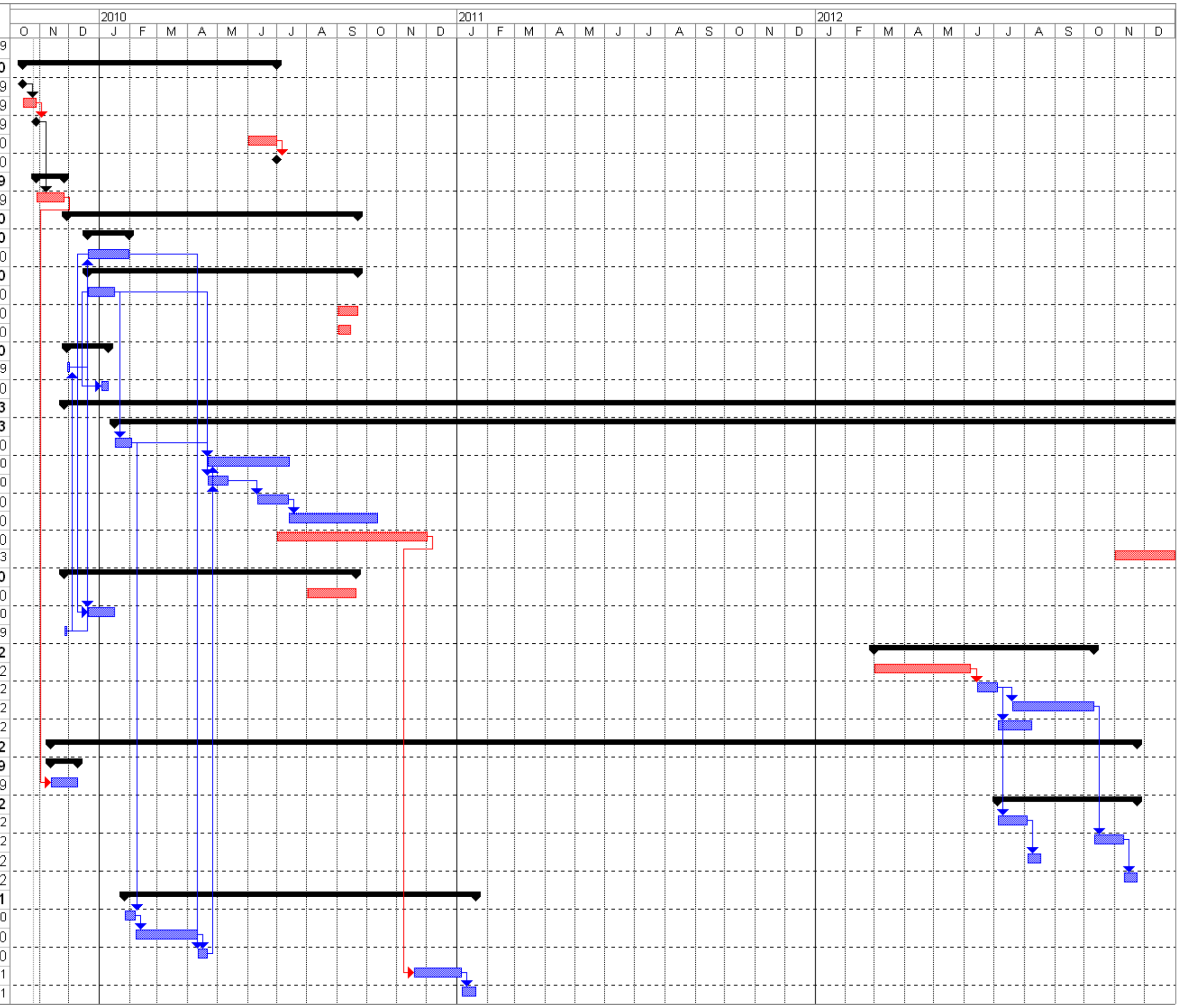


Figure 22: Pre-production Gantt Chart – Page 1 of 2

ID	Task Name	Duration	Start	Finish
52	**99 Zone FW Vent Circuit**	3.2 mons	Dec 17 '09	Mar 23 '10
53	Rehab 200L Cave	1.5 mons	Dec 26 '09	Feb 9 '10
54	Install Fan 3 (200L)	4 wks	Feb 23 '10	Mar 23 '10
55	Install Fan 1 and Ducting	10 days	Dec 17 '09	Dec 27 '09
56	**Definition Drill 12.5 x 12.5m**	2.64 mons	Apr 20 '10	Jul 8 '10
57	SW Defn Drill, 2 Drills, 1100m	11.3 wks	Apr 20 '10	Jul 8 '10
58	**Mine Infrastructure**	3.13 mons	Nov 26 '09	Feb 28 '10
59	Central Tails Storage, 25m	1 wk	Dec 10 '09	Dec 17 '09
60	Central Ore Remuck, 20m	1 wk	Dec 3 '09	Dec 10 '09
61	Central Waste Remuck, 15m	1 wk	Nov 26 '09	Dec 3 '09
62	Excavate Magazines, 25m	2 wks	Dec 27 '09	Jan 10 '10
63	Set Up Tool Crib	6 wks	Jan 17 '10	Feb 28 '10
64	**Process Plant Construction**	0.97 mons	Jun 20 '10	Jul 19 '10
65	Commissioning	29 days	Jun 20 '10	Jul 19 '10
66				
67	**BELLEKENO PRODUCTION**	41.83 mons	Mar 23 '10	Aug 29 '13
68	**SW -A Block**	13.4 mons	Apr 20 '10	May 27 '11
69	A Sill Xcut - 40m	2 wks	Apr 20 '10	May 4 '10
70	Vein Drainage -SWA	4 wks	May 4 '10	Jun 1 '10
71	A Production, 40tpd	12 mons	Jun 1 '10	May 27 '11
72	**SW - B Block**	33.37 mons	Apr 20 '10	Jan 15 '13
73	B Sill Xcut - 37m	2 wks	Apr 20 '10	May 4 '10
74	Vein Drainage - SWB	4 wks	May 4 '10	Jun 1 '10
75	B Production, 40tpd	137 wks	Jun 1 '10	Jan 15 '13
76	**SW - C2_U Block**	32.78 mons	May 12 '10	Jan 19 '13
77	C Sill Xcut - 50m	2.5 wks	May 12 '10	May 29 '10
78	Vein Drainage -SWC2_U	1 wk	May 29 '10	Jun 5 '10
79	C Production, 40tpd	137 wks	Jun 5 '10	Jan 19 '13
80	**SW - D Block**	33.37 mons	Dec 2 '10	Aug 29 '13
81	D Sill Xcut - 40m	2 wks	Dec 2 '10	Dec 16 '10
82	Vein Drainage -SW D	4 wks	Dec 16 '10	Jan 13 '11
83	D Production, 40tpd	137 wks	Jan 13 '11	Aug 29 '13
84	**99 - Block G**	13.28 mons	Apr 20 '10	May 24 '11
85	G Sill Xcut - 28m	1.5 wks	Apr 20 '10	May 1 '10
86	Vein Drainage - 99G	4 wks	May 1 '10	May 29 '10
87	99G Production, 40tpd	12 mons	May 29 '10	May 24 '11
88	**99 - F Block**	5.87 mons	Aug 18 '10	Feb 10 '11
89	Muck Out Waste, 2000t	3 wks	Aug 18 '10	Sep 8 '10
90	CRF Fill old stope above, 1000m3	2 wks	Sep 8 '10	Sep 22 '10
91	Fill curing time	3 wks	Sep 22 '10	Oct 13 '10
92	F Sill Xcut - 51m	2.5 wks	Sep 22 '10	Oct 9 '10
93	99F Production, 40tpd	4 mons	Oct 13 '10	Feb 10 '11
94	**99 - Block C**	5.4 mons	Mar 23 '10	Sep 1 '10
95	Cemented Rock Fill 600L, 600m3	2 wks	Mar 23 '10	Apr 6 '10
96	CRF the old stope	2 wks	Apr 6 '10	Apr 20 '10
97	99C Production, 40tpd	4 mons	Apr 20 '10	Aug 18 '10
98	Backfill C block CRF	2 wks	Aug 18 '10	Sep 1 '10

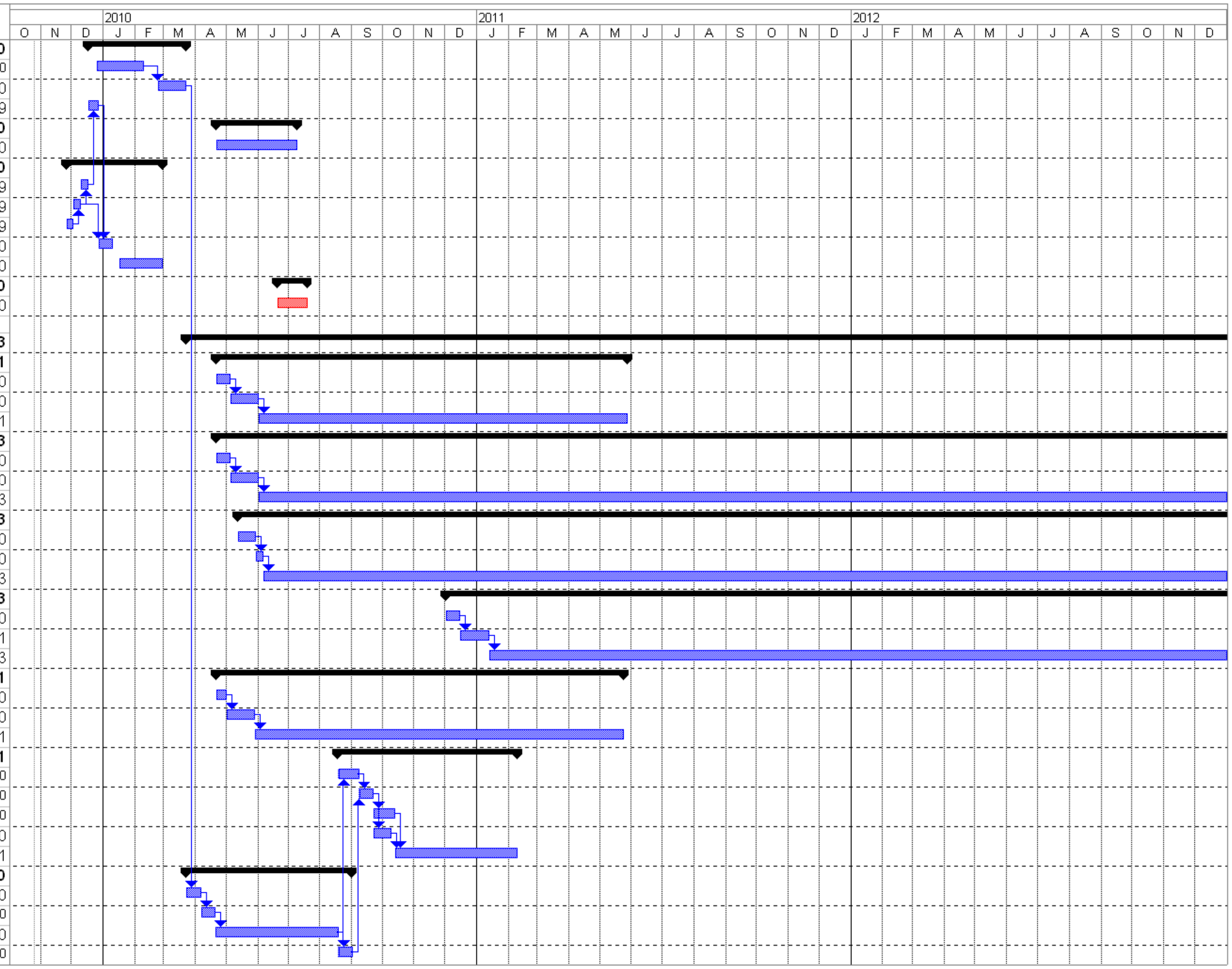


Figure 22: Pre-production Gantt Chart – Page 2 of 2

4.6 Pre-production Gantt Chart

4.6.1 Pre-production Schedule Results

The following list provides the key results from the pre-production Gantt chart:

- The pre-production Gantt chart indicates that the mine should be ready for production starting in June 2010;
- Lateral development required during the first 9 months totals 426m.

The critical path for mine activities is the following sequence of events. As noted above, mine pre-production tasks do not start until the IPC provides a positive report on the Development Plan. Mill tasks are excluded here:

Completed Date	
Oct.29.09	Positive report by IPC;
Nov.26.09	Contractor mobilization to site (4 weeks allowed);
Dec.17.09	Excavate the three central material storage cross cuts;
Dec.27.09	Install fan 1 and duct over and above the bottom of BK East decline;
Jan.17.10	Slash SW ramp for truck width, spot bolt, road and pipe work (4 weeks);
Feb.03.10	Drive 66m of SW A block ramp and 16m Alimak cross cut;
Apr.10.10	Drive 84m of Alimak for SW1 raise, install manway;
Apr.20.10	Install SW ventilation fan 5. Establish flow through ventilation for SW zone;
May.04.09	Complete SW A block cross cut, 40m;
Jun.01.10	Vein water drainage time allowance complete;
Beyond…	Production stoping can begin.

Note: There are many other important tasks happening in parallel, but this sequence has defined the critical path timing. Refer to the Gantt chart.

4.6.2 Ventilation Plans

The mine ventilation plan is closely linked to the pre-production mine tasks. General concepts that were considered to ventilate the mine included:

1. Exhaust air out through 625 adit (this is the current configuration);
2. Exhaust air through a new ventilation raise to surface;
3. Exhaust air through existing raises from historic mining. This would have exhaust air moving upwards through existing raises and onto upper levels, and out to surface on 200 level.

Note that all of these alternatives have in common fresh, heated air entering the mine through the BK East decline. Ventilation modelling and cost estimating support the current plan which is to combine (1) and (3).

Mining is scheduled to start first in SW and 99 zones and the initial start up ventilation plan is to move 30.7 cubic metres per second (“cms”) (65,000 cubic feet per minute (“cfm”) through the mine, with all exhaust out through the 600 level portal. This rate will continue until rehabilitation work on 200 level is completed.

Once 200 level is opened, total mine ventilation will increase to 33.0cms (70,000cfm) through the SW zone and 14.2cms (30,000cfm) through the upper 99 zone area, for a mine total of 47.2cms (100,000cfm). Intake will be through the BK E decline and exhaust will be split: 16.5cms (35,000cfm) out through 600 level portal and 30.7cms (65,000cfm) out on 200 level.

Once mining activity begins in the East zone, total mine ventilation will be increased to 63.7cms (135,000cfm).

Airway sizes and ventilation electrical power requirements are based on modelling.

4.6.3 Production and Development Schedules

The Bellekeno LoM production schedule is shown in Table 16. Although it is presented separately from the development schedule (refer to Table 17), the two schedules are linked together so that the production schedule is dependent on the timing of the development that accesses each mining area.

A maximum allowable production rate was estimated for each mining area (stope block) to ensure that it would not be scheduled at an unrealistically high production rate. These maximum rates were based on stope cycle times and the number of independent mining fronts within each stope block.

Total LoM production is planned at 321,000 tonnes over a 3.6 year mine life with plant feed metal grade averages of 0.42gpt Au, 870gpt Ag, 9.5% Pb, 5.6% Zn. The average NSR value is $506 per tonne milled.

Production is planned to begin in the SW zone with the high NSR value mining blocks and with no constraints from previous mining.

The development schedule shows a peak in requirements during mid-2012. This coincides with the development of the East Mid ramp system.

Table 16: Bellekeno Production Schedule

SW Zone	Mineable Tonnes	NSR diluted	2010 Q1	Q2	Q3	Q4	2011 Q1	Q2	Q3	Q4	2012 Q1	Q2	Q3	Q4	2013 Q1	Q2	Q3	Q4	TOTAL
A	29,454	$560		1600	4446	5000	3002	3000	3000	2000	3000	3000	1406						29500
B	71,223	$560		1900	5700	5330	5700	5700	5700	5700	5000	5000	5178	5700	4247	3844	3516	3008	71200
C_Upper	44,139	$618		1900	6700	6700	6700	6000	6700	6700	2739								44100
C_Lower	32,475	$396									4000	4000	4000	4100	4100	4100	4100	4075	32500
D	32,226	$388					1400	2800	2800	2800	2826	2800	2800	2800	2800	2800	2800	2800	32200
E	7,996	$475													1999	1999	1999	1999	8000
Sub-total SW	**217,512**	**$519**	0	5400	16846	17030	16802	17500	18200	17200	17564	14800	13384	12600	13146	12743	12415	11882	**218,000**
99 Zone																			
B	5,683	$377										1300	1300	1300	800	983			5700
C	4,627	$508			2776	1851													4600
D	1,364	$578									1364								1400
E	2,971	$466							1486	1486									3000
F	5,396	$854					2698	2698											5400
G	27,247	$675		2100	2878	3619	3000	2302	2815	2616	2373	2137	1200	2207					27200
H	6,128	$364										3064	3064						6100
J	4,795	$295								1199	1199	1199	1199						4800
Sub-total 99	**58,211**	**$572**	0	2100	5654	5470	5698	5000	4300	5300	4936	7700	6763	3507	800	983	0	0	**58,200**
East Zone																			
Upper 48	14,121	$454											2354	2354	2354	2354	2354	2354	14100
East_Mid_U	20,086	$345												4039	3,500	3,586	4,500	4,461	20100
East Mid_L	12,010	$271													2700	2834	3232	3245	12000
Sub-total East	**46,218**	**$359**	0	0	0	0	0	0	0	0	0	0	2354	6393	8554	8774	10085	10059	**46,200**
TOTAL PRODUCTION		**tonnes**	**0**	**7,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**22,500**	**21,900**	**322,000**
	Plant Feed:	**TPD**		250	250	250	250	250	250	250	250	250	250	250	250	250	250	244	
	Au	gpt	0	0.44	0.44	0.44	0.46	0.45	0.44	0.43	0.44	0.41	0.42	0.43	0.38	0.38	0.36	0.36	0.42
	Ag	gpt	0	1040	1000	1010	1060	1030	955	931	873	805	789	814	728	722	712	706	870
	Pb	%	0	12.0	11.6	11.8	11.9	11.6	11.5	11.1	10.1	8.6	7.7	7.5	7.2	7.0	7.0	7.0	9.5
	Zn	%	0	5.5	5.1	5.3	5.5	5.6	5.4	5.3	5.6	5.1	5.4	5.9	6.0	6.0	6.2	6.2	5.6
	NSR	$/t		**$607**	**$586**	**$592**	**$617**	**$601**	**$564**	**$549**	**$516**	**$469**	**$453**	**$460**	**$416**	**$412**	**$406**	**$402**	**$506**

Table 17: Bellekeno Lateral Development Schedule

RAMPS	2009	2010				2011				2012				2013				TOTAL
SW Ramps	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	**Q1**	**Q2**	**Q3**	**Q4**	
SW Blk_A Rp		66	31	50	40													187
SW Blk_D Rp				110	90	100												300
SW Blk_BC Rp			126	17.1	30	20	40	30	30	30								323
SW Blk_E Rp													36	36	36	36		146
99CaveByPass								80										80
99SWByPass				36														36
99 Ramps																		0
99_FW_Ramp								46	20	33								99
East Ramps																		0
East_Ramp1										179	269							449
East_Ramp2											76	19						95
East_Ramp3												346	38					384
Total Ramps	**0**	66.4	157	213	160	120	40	156	50	242	345	365	75	**36**	**36**	**36**	**0**	**2,099**
INFRASTRUCTURE																		
Central Cross cuts	60																	
Magazines (total)	25																	
Total Infrastructure	**85**	0	0															**85.0**
CROSS CUTS																		
SW cross cuts	0	0	40	123	127	131	152	156	144	176	160	143	131	152	150	148	144	2078
99 cross cuts	58	0	20	28	35	54	47	70	125	80	211	202	65	27	33	0	0	1054
East cross cuts	0	0	0	0	0	0	0	0	0	0	89	0	42	98	102	120	120	571
Total Cross Cuts	**58**	**0**	**60**	**150**	**162**	**185**	**199**	**225**	**269**	**256**	**460**	**345**	**239**	**277**	**285**	**268**	**265**	**3,702**
Total Lateral (meters)	**143**	**66**	**217**	**363**	**322**	**305**	**239**	**381**	**319**	**499**	**806**	**710**	**314**	**313**	**321**	**304**	**265**	**5,886**
Meters per day	1.6	0.7	2.4	4.0	3.6	3.4	2.7	4.2	3.5	5.5	9.0	7.9	3.5	3.5	3.6	3.4	2.9	

4.7 Equipment, Manpower and Services

4.7.1 Contractor Involvement

Alexco will utilize a mine contractor for all mining functions. Alexco personnel will undertake surface truck haulage, and will provide technical services. All mine development and production work will be performed by a mining contractor. Limited in-fill diamond drilling and any additional exploration drilling may be performed by separate contractor.

4.7.2 Mining Equipment

Table 18 shows the planned owner's equipment fleet under the operating scenario where a contractor performs the mining. Excluded from Table 18 are contractor diamond drills.

Table 18: Owner's Equipment Summary

Mine Mobile Equipment	Number
Tractor	2
Surface Truck - 28 tonne	2
Pickup truck	2
Total Units	**6**

The tractors are for Alexco supervision and technical service functions.

The surface trucks are planned as articulated six wheel drive units that will be able to safely negotiate the winter roads on surface. Chains will be used for traction as required.

Surface trucking includes the truck cycle time from the central underground remuck bays, up the BK East decline, and to the surface destinations. Average one way haulage distances are 1570m for waste rock and 5700m for haulage to the Flame & Moth mill site.

The pick up trucks will be used by Alexco's mine department employees.

Due to the relatively short mine life, no replacement equipment will be required.

4.7.3 Underground Rock Handling

Mine plans are based on using 10-tonne class underground trucks for handling vein material and waste. These smaller trucks will not be used on surface. Existing ramp cross sections are nominally sized at 3.0 to 3.2m in height with an average width of 3.7m. New ramps and cross cuts will be sized at 3.7 x 3.7m.

From the active stope face, LHDs will transport blasted vein material to remuck bays that are planned along the access ramps at each cross cut intersection. All vein material from cut and fill stoping must be transported to the planned central remuck bay where surface trucks will rehandle the material up the BK East decline to the processing plant.

All underground truck loading will be done by LHD.

Nominal one way haulage distances from ramp remuck bays near the active stopes to the centrally located vein material remuck bay are estimated as follows:

- SW zone – 960m;

- 99 zone – 310m;
- East zone – 600m.

All waste rock from development must be transported to the planned central waste rock remuck bay where surface trucks will rehandle the material up the BK East decline to the waste rock dump. Some of the waste rock will be placed by LHD into overhand cut and fill stopes as backfill.

A small percentage of the waste rock that may have the potential to become metal leaching ("PML") will be trucked to a special containment area, or used as preferred rock fill.

Surface trucking will utilize 28 tonne, six wheel drive articulated trucks to transport vein material and waste rock from the central underground remuck bays for delivery to the processing plant or waste rock pile.

Surface trucks hauling vein material to the mill will back-haul dry tailings from the mill, back down the BK East decline, to the central cross cut for tailings storage.

4.7.4 Grade Control

Alexco intends to establish a Grade Management System ("GMS") at the Bellekeno mine. Elements of the system recommended by SRK are described below.

Delineation. This task involves the generation of reliable vein data to enable the delineation of vein material above cut off. This will be achieved through a combination of detailed mapping and sampling of cut and fill headings and underground grade control drilling as needed. An on site assay lab will be utilized.

Database Update. This task will ensure that data such as underground face sampling and geological mapping is systematically and accurately transferred to an industry standard database in a format that can readily be exported to other software applications.

Grade Estimation. This task will ensure that the best unbiased grade estimate is obtained for each portion of vein to be mined. The 3D geological model will be updated by considering the latest mapping results. A 3D grade control block model will be constructed and regularly updated using grade control assay results.

Mining Block Definition. Stoping block outlines for each area of vein planned for mining will be established using the updated grade control block model. This will involve applying an appropriate cut off NSR and defining the mining limits. Areas of "low grade" vein material can also be identified. The outlines will represent practical mining shapes such that they can be extracted with minimum dilution.

Mining Extraction. This task focuses on mining quality, with the main objective being to minimize unplanned mining dilution being incorporated into the vein material. For mining extraction to occur effectively, close lines of communication between mine operators, geologists and mine engineers will be established. Mining quality will be a team effort that is the product of effective training and the adoption of well defined mining procedures and standards.

Stockpile Balance Sheet. The stockpile balance sheet enables "live" stockpile tonnage and grade status reports for the run-of-mine ("RoM") and low grade stockpiles. All stockpiles will be updated daily with tonnes added from mining sources, stockpile transfers and depletions due to crusher feed. The balance sheet is critical for predictions of daily grade as well as being an integral part of the resource to mine to plant reconciliation system, which will measure the overall mining process quality.

Reconciliations. Reconciliations will be conducted at various time intervals to evaluate different parts of the mining and material handling cycle. Reconciliations are valuable management tools to evaluate the quality of the mining process and to make management aware of any problems so that measures will be implemented to remedy the situation. SRK considers the following reconciliations as applicable:

- Predicted grade vs. plant head grade;
- Grade control model vs. as mined;
- Predicted grade vs. plant head grade averaged for a month.

Follow Up. Significant and consistent variances highlighted in the various reconciliation studies will be followed up and resolved. Reconciliation variances in the GMS indicate that the mining process is not being optimized, which if left unattended can impact negatively on the overall financial performance of the operation.

Reporting. Measurable outputs of the GMS will be reported monthly to management in a format that permits a clear understanding of the grade control and mining quality issues and allows informed decisions concerning remedial action if required.

4.7.5 Backfilling

Backfill will consist of a mix of development waste rock and dry (filtered) tailings from the process plant. Cemented backfill will be trammed into empty stopes by LHD and will be pushed up tight to the back using an LHD equipped with a "jammer" push plate. Refer to Figure 23.

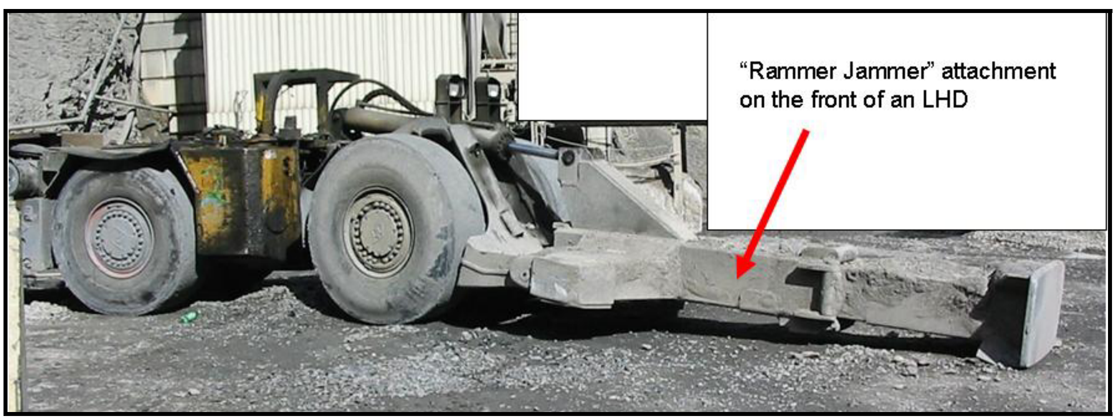


Figure 23: Typical Backfill Jammer Attachment

Cement addition to the backfill will be achieved by spraying a metered amount of cement slurry into the LHD bucket as it is delivering the backfill to the stope. The slurry will be prepared in a mobile underground mixing plant, fed with bulk (1 tonne) bags of cement. LoM cement addition to backfill is estimated at 2% by weight.

The dry tailings storage and waste rock storage quantities are designed to be accommodated in underground cross cuts to avoid winter freezing problems on surface.

Table 7, presented in a previous report section, shows the LoM quantities of development waste rock broken and the quantities of waste rock and dry tailings used in backfill. The LoM backfill supply will consist of 84.8 kilotonnes ("kt") waste rock (43%) and 114kt dry tailings (57%).

4.7.6 Mine Manpower

The owner's mine manpower estimate is shown in Table 19 organized by function. The planned work schedule is two 11-hour shifts per day, seven days per week. Employees will stay in camp on site, working a rotation of two weeks and two weeks out.

The mining contractor's manpower is not shown.

Table 19: Alexco Mine Manpower Estimate

Position	Alexco Mine Employees
Supervision	
Mine Captain	2
Mine Clerk	1
Surface O & W Trucking	
Volvo Truck Driver	3
Surface	
Dozer	0.5
Technical Services	
Senior Engineer	1
Mine Engineer	2
Mine Technician	1
Surveyors	2
Sr. Geologist	1
Geologist	1
Geological Technician	2
Total Alexco Mine Employees	**17**

4.7.7 Underground Support Services

Mine Ventilation

The initial air balance is shown in Table 20. All exhaust will be directed to surface out through the 600 level adit. Propane fired heaters are planned at the main air intake, BK East decline.

Table 21 shows how the air flows will be increased once 200 level rehabilitation is completed. The quantity of 47.2cms covers mining in SW and 99 zones. Once mining starts in the East zone, mine ventilation will be increased to a total of 63.7cms.

Table 20: Ventilation Balance - Initial

Planned Airway	Intake cms	Intake cfm
BK Main Decline splits into:	30.7	65,000

99 zone FW up ramp	9.4	20,000
600 level down ramp to SW zone	21.2	45,000
Planned Airway	**Exhaust cms**	**Exhaust cfm**
600 level back to central mine	21.2	45,000
99 zone exhaust down raises	9.4	20,000
these exhausts streams combine:		
600 level exhaust to 600 adit	30.7	65,000

Table 21: Ventilation Balance – 200 Level Open

Planned Airway	**Intake cms**	**Intake cfm**
BK Main Decline	47.2	100,000
splits into:		
99 zone FW up ramp	14.2	30,000
600 level down ramp to SW zone	33.0	70,000
Planned Airway	**Exhaust cms**	**Exhaust cfm**
600 level back to central mine	33.0	70,000
99 zone exhaust	14.2	30,000
these exhausts streams split:		
out to surface on 200 level	30.7	65,000
600 level exhaust to 600 adit	16.5	35,000

The underground mine ventilation system was modelled using commercial software to determine the operating points for the planned underground fans in terms of air flows and pressures. SRK calculated the power draw for each main fan to support the mine electrical power consumption estimate.

Mine Dewatering

The main sump was developed at the time that the BK East decline was driven. It is a double sump arrangement located centrally at the bottom of the BK East decline.

Clarified water from the main sump is pumped to the water treatment plant through piping installed along 600 level, to the 600 level adit. Pumping power requirements are minimal as there is almost no vertical head along the piping profile.

As the mine begins production and development opens new areas, small area sumps will be set up with 10 kilowatt (“kW”) submersible pumps, directing dirty water to the main sump.

Electrical Power Distribution

The estimated underground mine average power draw is 0.5 megawatts (“MW”). The larger loads are related to the main ventilation fans, auxiliary ventilation fans, surface air compressors and face jumbos.

The mine currently has an operating underground electrical power distribution system that will be expanded during mine production.

Power feed into the mine comes in through the 600 level adit comprising a 6900 volt (“V”) line that extends across 600 level to the central mine area where it feeds a 1500 kilovolt ampere (“kVA”) transformer.

Power will be distributed underground by 4160V cables to area transformers that will step the voltage down to 600V suitable for mining equipment. As the mine is developed, two new transformers will be installed in the SW zone, and one in the East Mid zone.

Compressed Air Distribution

Existing air compressors located at the 600 level adit will handle the planned air requirements. There are two existing JOY 750cfm, 93kW (125HP) electric compressors. Mine capital includes a $50,000 provision for upgrading these units. Compressed air is delivered into the mine through 150mm piping installed on 600 level.

The main uses of compressed air will be for stoppers and jacklegs installing ground support, Alimak raising, periodic shotcrete work, face pumps and anfo loaders, and limited use of the bench drill.

Average consumption during the production period is estimated at approximately 0.47cms (1,000cfm).

Mine Maintenance

Most of the mobile equipment maintenance will be performed in the existing surface shop, located at the BK East decline. The mine area is relatively small and it will not be difficult to bring underground equipment to the surface shop. An additional small maintenance shop will be set up underground to handle small repairs and routine servicing.

The maintenance department will have a fuel/service truck, a tractor, a scissor lift for electrical work, and a boom truck.

In addition to the mobile equipment, the mine maintenance department will be responsible for the stationary equipment consisting of air compressors, main ventilation fans and heaters, electrical distribution system, and main dewatering pumps.

5 References

Bellekeno Preliminary Economic Assessment Report, Keno Hill Mining District, Yukon, SRK Consulting (Vancouver)' June 2008

Draft Bellekeno Geotechnical Report, Ross Greenwood and Bruce Murphy, October 9, 2009

Bruce R. Otto (October 8, 2009) entitled "Structural and stratigraphic relationships at the Bellekeno Mine; their control on grade and thickness and their implications for exploration."