NI 43-101 Technical Report Sintoukola Potash Project Republic of Congo

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Summary (Item 1)

This Technical Report has been prepared by SRK Consulting (U.S.), Inc. (SRK), AMEC Americas Limited (AMEC), EGIS International (EGIS) and CSA Global Pty Ltd (CSA) (collectively the Consultants) on behalf of Elemental Minerals Ltd (ELM). The consultants were commissioned to prepare a Technical Report compliant with Canadian National Instrument 43-101 (NI 43-101) for a Prefeasibility Study (PFS) on the Sintoukola Potash Project (Sintoukola Project) located in the Republic of Congo (ROC). ELM is an international resources company listed on the Australian Stock Exchange (ASX) and Toronto Stock Exchange (TSX) under the symbol “ELM”. The Sintoukola Project is held by Sintoukola Potash S.A. (SPSA), a private company registered in the ROC in which ELM holds the majority equity interest of 93%.

Property Description and Location

The Sintoukola Project is situated in the Kouilou Province in the south west corner of the ROC. The Sintoukola Project comprises an exclusive exploration permit, for potash and associated salts, which covers an area of 1,436.5 square kilometres (km^2) along the northern part of the coastline in the west of the country. Access to the project area is via a 55 kilometre (km) dual-lane bitumen road from the port city of Pointe Noire, which terminates 9 km outside the permit’s southern boundary. From this point, access to the Kola deposit area is via a 35 km gravel track.

History and Ownership

Potash was discovered in the coastal Congolese Basin during oil prospecting in the region in the 1930’s. Subsequent exploration between the 1930’s and 1950’s revealed that the salt formations continued through ROC and into Gabon. In the 1960’s significant exploration for potash was completed in the Kouilou portion of the Congo Basin. The exploration discovered the Holle potash deposit, located to the south of the Sintoukola permit area, and identified sylvinite and carnallitite mineralisation at the Kola deposit and other areas within the current Sintoukola permit area.

There have been no historical estimates of Mineral Reserves, nor any production of potash or related products within the Sintoukola property. Underground mining of sylvinite seams has occurred near Holle, some 70 km south of the Kola deposit.

ELM’s majority held subsidiary SPSA was awarded the Sintoukola exploration permit on August 13, 2009. SPSA is a company registered in the ROC whose only asset is the Sintoukola exploration permit. ELM holds 93% of the equity in SPSA with Les Establissements Congolais MGM (MGM) and Tanaka Resources (Tanaka) holding equity positions of 5% and 2%, respectively. ELM holds a right of first refusal over the equity of SPSA by the minority shareholders.

ELM commenced exploration in 2010, focusing on the Kola deposit in the eastern part of the permit where high-grade sylvinite mineralisation was intersected in several historical drillholes. By February 2011 ELM had completed an initial 16 diamond drillhole program (Phase 1) for 6,577 m on the Kola target, as well as down hole geophysics, core sampling, core analyses and a two-dimensional seismic survey. This program was followed by a second drilling program (Phase 2a) of 20 drillholes for 6,429 m and a third drilling program (Phase 2b) of eight exploration holes for 2,745 m supported by a high resolution 2D seismic survey program.

In 2011, at the completion of ELM's first phase of exploration a maiden Mineral Resource estimate was reported for the Kola deposit by CSA (SRK, 2011). In May 2012 an update to the Mineral Resource was delivered (CSA, 2012) and a second Resource update was announced by ELM on August 21, 2012 that is reported within this technical report.

Geology

The evaporite sequence of the Congolese coastal basin consists of essentially flat-lying, but locally undulating salt layers of interbedded halite ($NaCl$), and other higher salts such as sylvite (KCl), carnallite ($KMgCl_3 \cdot 6H_2O$), bischofite ($MgCl_2 \cdot 6(H_2O)$), and minor anhydrite ($CaSO_4$) and dolomite ($CaMg(CO_3)_2$) beds that extend from the onshore Congo Basin north and south to sedimentary basins in adjacent West-African countries, as well as west into offshore regions.

The potash occurs within the Lower Cretaceous Loeme Evaporite Formation. At the Kola deposit, seven depositional cycles can be differentiated within the evaporite sections comprising alternations of the rock salt and carnallitite (carnallite and halite) converted locally to sylvinite (sylvite and halite).

Evaluation of drill core, assays, down hole geophysical logs, and seismic data suggest that potentially economic potash mineralisation occurs within the upper portion (cycles VI and VII) of the evaporite sequence.

The interpreted extent and structure of the potash beds is based on examination of new drill-hole data, regional historic drilling conducted by the previous explorers, and review of seismic data. Based on this data approximately 7% of the area surveyed by high resolution 2D seismic surveys contains geological anomalies (structural lows or disturbance areas). The structurally anomalous areas may impact the thickness, grade and potential mineability of the potash mineralisation and therefore further site-specific studies are required to assess these areas. For this reason, the areas identified as disturbance areas have been excluded from estimates of Mineral Resources.

Mineralisation

Four mineralised seams were interpreted and are named, starting with the uppermost, as the Hangingwall Seam (HWS), Upper Seam (US), Lower Seam (LS), and the Footwall Seam (FWS). The potash seams can be correlated across the deposit extent and are generally sub-horizontal with local undulations; modeled dips do not exceed 15 degrees.

The top of the salt sequence is marked by an unconformable contact. Depending on the position of the seams relative to this contact, they may be locally removed.

The mineralisation comprises sylvinite layers, carnallitite layers or less frequently both sylvinite and carnallitite. In the latter, the two mineralisation types are not inter-mixed; the sylvinite occurs above the carnallitite.

Sylvite is a considered to be a secondary mineral, forming by the leaching of magnesium chloride from primary carnallite. At the Kola deposit, it is envisaged that the replacement of carnallite by sylvite was controlled by the vertical and lateral movement of leaching brines originating from the top of the salt sequence.

The uppermost seam modelled to date is the HWS and is separated by approximately 60 m of halite from the top of US. The HWS is known to consist of high grade sylvinite with a thin layer of carnallitite towards the base of some intersections.

The US and LS are in close vertical proximity, separated by a barren halite interval with average thickness of 3.6 m. The US consists predominantly of sylvinite with an increasing proportion of halite towards the upper contact. The LS is either entirely of carnallitite or sylvinite, or comprised of sylvinite above carnallitite.

The FWS is the lowermost significant potash seam. It is separated by approximately 45 m of halite from the base of the LS. In places where the US and LS have been largely or entirely removed, it consists of sylvinite, elsewhere the FWS is known to consist of carnallitite and bischofite.

On the basis of potash mineralogy, sylvinite and carnallitite domains can be delineated within the US and LS. These are identified as Upper Seam Sylvinite (USS) and Upper Seam Carnallitite (USC) and the Lower Seam Sylvinite (LSS) and Lower Seam Carnallitite (LSC). Within the HWS and FWS, only domains of sylvinite mineralisation (HWSS and FWSS) were modelled for estimation of Mineral Resources. The FWS also has a carnallitic domain (often with significant bischofite) but as this mineralisation has not yet been systematically sampled, it has not been included in estimates of Mineral Resources. All mineralised domains have been constrained using “hard boundaries” from interpretation of geological logs, down hole geophysics, modelled seismic data and assay results.

Drilling and Sampling

The Sintoukola Project has been the subject of two periods of exploration i.e. historical exploration in the 1960’s and exploration by ELM during the last three years. The exploration has comprised drilling (mud rotary precollar to the top of evaporite section and then diamond cored tails to end of hole) and seismic surveys.

Of the 31 historical drillholes completed within and adjacent to the permit area, five drillholes within the Sintoukola Project area returned significant potash results and two (K-18 and K-6) were the initial target for ELM’s exploration program. Detailed well completion reports (in French) are available for all five holes. The geological and analytical data from these holes, combined with the historical seismic data, was used to create the initial geological model for exploration planning.

ELM’s 45 drillholes completed during three phases of exploration comprised two holes to validate the drill results from historical holes K-18 and K-6, a hydrogeological drillhole (EK_37) and 42 delineation holes to define extensions to the historically intersected potash mineralisation. Geological, geochemical and geophysical data from these holes were incorporated in interpreting geological models, but only data from ELM’s cored and assayed holes were used for the estimation of Mineral Resources.

Core recovery during diamond drilling, through the evaporites has been excellent. Minor dissolution of salts occurs from time to time but these are generally not in intervals of potash mineralisation and the problem is quickly overcome by adjusting the composition of the drilling mud. Four holes were abandoned due to drilling problems and another could not be geophysically logged due to hole collapse. These issues, and other factors that could impact the accuracy and reliability of the results, such as drill sample spacing, sampling handling and density, were considered and adequately managed.

Sample preparation and analysis were carried out by K-UTEC Salt Technologies in Sondershausen, Germany and Genalysis Laboratories, Perth, Australia. Both laboratories have been certified in accordance with ISO/IEC 17025. No aspect of laboratory sample preparation was conducted by an employee, officer, director or associate of ELM.

ELM has used a combination of duplicates, standards and blanks to ensure suitable quality control of their assay testing. Their procedures and quality assurance and control (QA/QC) management are consistent with industry practice and are deemed fit for purpose.

Mineral Resource Estimates

The geological model for the Kola deposit is based on the interpretation of geological boundaries from interpreting drill core, assays, down hole logging and seismic data. Three dimensional models were developed in Micromine software and then refined using the gridding process in MINEX software. The extents of the resource model were determined by availability of drillhole and seismic data and the limits of influence in accordance with selected Mineral Resource classification parameters.

Following compilation, review and interpretation of the latest geological, geophysical and analytical data an updated Mineral Resource was estimated for the main potash-bearing (sylvinite and carnallitite) horizons within the Kola deposit area. The geological model now comprises six mineralisation domains within four distinct seams (as described above).

The Mineral Resource estimates have been reported at a Cut off Grade (CoG) of 10% K_2O and with reference to Tables 1 and 2 below are presented as two scenarios:

- As a sylvinite estimation only, and
- As a combined sylvinite and carnallitite estimation

The Mineral Resources are reported in accordance with the Australasian Joint Ore Reserves Committee guidelines for reporting of Mineral Resources and Ore Reserves, (the JORC Code, 2004 edition) which is consistent with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition standards and hence complies with NI 43-101 requirements. The base-case Mineral Resource is of sylvinite only.

The Mineral Resource estimate for the modelled mineralised zones at the Kola deposit has been classified as Measured, Indicated and Inferred. This is based primarily on confidence in, and continuity of, the results from the drilling campaigns, and subsurface mapping of high density 2D seismic data. The results of the Mineral Resource estimate are shown in Table 1.

Table 1: Mineral Resource Estimate for Sylvinite Mineralisation only (Base case) at a 10% K_2O CoG, as announced August 21, 2012

	Measured			*Indicated*			*Inferred*		
	Tonnes (Mt)	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**
HWS							47	34.75	55.01
USS	171	22.45	35.54	159	22.04	34.89	96	21.78	34.48
LSS	93	19.22	30.42	150	19.06	30.17	107	19.14	30.30
FWS							225	17.63	27.92
Total	264	21.32	33.74	309	20.59	32.59	475	20.39	32.27

Notes:

1. *A bulk density of 2.07 g/cm^3 was applied for all sylvinite mineralisation and 1.70 g/cm^3 for carnallitite mineralisation.*
2. *Zones of geological uncertainty have been excluded.*
3. *Table entries are rounded to the second significant figure.*
4. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
5. *Insoluble contents for the resource were not estimated but insoluble content of the seam intersections are provided in Table 8.8.1.*

Table 2: Mineral Resource Estimate for Sylvinite and Carnallitite Mineralisation at a 10% K_2O CoG, as announced August 21, 2012 (Includes Resource in Table 1)

	Measured			*Indicated*			*Inferred*		
	Tonnes (Mt)	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**
HWS							47	34.75	55.01
USS and USC	245	19.53	30.92	310	17.76	28.11	278	16.33	25.84
LSS and LSC	313	13.26	20.99	448	13.74	21.75	398	13.12	20.77
FWS							225	17.63	27.92
Total	559	16.01	25.35	758	15.38	24.35	948	16.20	25.64

Notes:

1. *A bulk density of 2.07 g/cm^3 was applied for all sylvinite mineralisation and 1.70 g/cm^3 for carnallitite mineralisation.*
2. *Zones of geological uncertainty have been excluded.*
3. *Table entries are rounded to the second significant figure.*
4. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
5. *Insoluble contents for the resource were not estimated but insoluble content of the seam intersections are provided in Table 8.8.1.*

Mining and Mineral Reserves

A comprehensive hydrogeological and geotechnical survey program was completed to determine mine design parameters to be applied in the reserve conversion. The hydrogeological program concluded that although the anhydrite sequence can be considered an aquitard, it did not occur in some 10% of the holes drilled. For this reason, the salt back, which can be considered an aquitard, is relied upon for mine design purposes. The geotechnical parameters propose a conservative extraction ratio to protect the integrity of the salt back. All pillars are considered as permanent pillars, resulting in negligible subsidence.

The strength of potash material makes it amenable to a non-explosive mining method such as continuous miners (CM). The reserve estimate was exclusively determined from measured and indicated resources of sylvinite ore from the USS and LSS. Because of the sharp grade boundaries of the sylvinite seams and the fact that the economic CoG is below the resource CoG of 10% K_2O, all sylvinite in the Measured and Indicated Resource was considered for reserve conversion. The reserve estimation allowed for:

- Mining recovery based on geotechnical extraction ratios;
- Exclusion of disturbance zones and buffer pillars around these zones; and
- Allowing for pillars around existing drillholes.

Room and pillar mining will be used with a minimum mining thickness of 1.8 m and a maximum thickness of 8.2 m. A maximum mining height was not applied as multiple cuts can be taken with the CM. Material from the CM will feed onto a shuttle car and then through a feeder breaker onto a system of conveyors for movement to surface. A three main drive system is utilized where the center main houses the main conveyors and other two accesses are used for men/materials providing access to panels on either side of the mains.

Table 3 presents the Sintoukola Project underground mining ore reserve, based on the PFS resource model issued by CSA (August, 2012).

Table 3: Sintoukola Underground Ore Reserve

Proven			***Probable***			***Total***		
Tonnes (Mt)	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**
87.9	20.01	31.68	63.8	20.02	31.69	151.7	20.02	31.69

The following parameters were used in creating the production schedule:

- Process plant feed capacity of approximately 570 kt Run of Mine (RoM)/month;
- Mine one area completely prior to moving to another (southeast first, then northwest) to minimize ventilation and services requirements;
- Mine on retreat where possible;
- Mine USS prior to LSS mining in any one area;
- Five day panel change provision was included for each machine as it moved from one panel to another; and
- A rate of 2,600 tonnes per day (tpd) was used for each CM. For production ramp up 50% of this rate was used for the first 6 months, 75% next 6 months, and full production rate thereafter.

Table 4 shows a summarized annual production schedule for underground ore production.

Table 4: Yearly Production Schedule

Year	Mining Ore Tonnes (kt)	K_2O Grade (%)	Avg Mining Thickness (m)	Ore Tonnes to Process Plant (kt)	Process Plant MoP Tonnes (kt)
1	2,369	20.71	4.6	1,538	467
2	6,342	20.45	4.08	6,775	2,053
3	6,871	21.19	4.16	6,840	2,151
4	6,874	21.25	4.17	6,840	2,149
5	6,871	21.21	4.14	6,859	2,158
6	6,875	20.42	4.31	6,840	2,078
7	6,874	19.48	4.34	6,840	1,977
8	6,877	19.78	4.45	6,840	1,993
9	6,861	19.83	4.58	6,859	2,012
10	6,856	19.43	4.74	6,840	1,963
11	6,878	19.81	4.87	6,840	2,005
12	6,882	19.62	4.89	6,840	1,987
13	6,880	19.74	5.00	6,859	1,996
14	6,881	19.33	4.52	6,840	1,974
15	6,874	19.17	4.75	6,840	1,926
16	6,880	19.53	4.37	6,840	1,974
17	6,872	19.63	4.42	6,859	1,990
18	6,862	20.23	4.32	6,840	2,041
19	6,858	19.90	4.20	6,840	2,021
20	6,883	19.93	4.34	6,840	2,008
21	6,817	20.63	4.82	6,859	2,086
22	6,598	20.13	4.76	6,840	2,046
23	5,242	19.22	5.17	5,870	1,678
24	661	20.42	6.43	294	88

Since the depth to ore is considered to be shallow compared to traditional shafts, Alan Auld Engineering (AAE) was engaged to investigate alternative shaft sinking options, mainly focusing on civil type construction earthworks. The proposed alternative method uses civil engineering shaft sinking principles that involve less front end work and do not require the procurement of special hoisting equipment. Material is removed from the shaft and mine using a high angle conveyor (HAC) and industrial lifts are used for equipment access. A ground freeze will be utilized for shaft development.

Two large diameter shafts (12 m diameter) will be sunk to allow for efficient machine and personnel movement and effective ventilation flow. Two adjacent, auxiliary shafts (8 m diameter) will be sunk to a depth of 100 m and will be used to assist the shaft sinking and mucking process. Each auxiliary shaft will be connected to the main shaft on this sub-level. Upon completion, the auxiliary shafts will be equipped as plenums for the mine ventilation system.

Metallurgy and Process

Metallurgical testing was undertaken in 2011 to establish preliminary processing technology for the Sintoukola Project. Mineralogy analysis showed the sample was composed of 38% sylvite (24.2% K_2O). Insoluble content was less than 1% and consisted of anhydrite. The sample received was categorized as coarsely intergrown sylvinite. The low insoluble content and the ease of liberation size of the Sintoukola ore compare favourably with the best examples from Saskatchewan, Canada and other producing areas of the world. This allows for high recovery and competitive processing costs.

The results of the metallurgical test program indicate that the Sintoukola ore can be effectively processed using a conventional flowsheet consisting of rougher/ cleaner flotation followed by regrind flotation. Flotation recoveries of up to 94.8% were achieved in the laboratory. The mass balance was developed using METSIM (metallurgical process simulation software). A combination of the Saskatchewan Research Council (SRC) test results and AMEC's potash experience was used to determine the inputs into the model. This resulted in a process plant recovery of 91% at a feed grade of 39.6% KCl (25.0% K_20). At the LoM feedgrade of 20.02% K_2O, overall metallurgical recovery is projected at 89.5%.

Subsequent to the original metallurgical test program, testing was conducted on sylvinite samples for the lower grade portion of the Upper Seam 2 (US2) in May 2012. In addition, testing was conducted on material from the LSS in July 2012. The objective of these test programs was to determine if the insoluble material composition and liberation size of the US2 and LSS were different from the higher grade portion of US1.

The insoluble content and composition of the US2 sample was similar to the US1 sample. The insoluble content of the LSS sample was lower than the US1 sample (0.1% versus 0.4%) The composition of the insolubles in the LSS sample included anhydrite (similar to the US1) and trace amounts of quartz.

The liberation size of the US2 and LS samples was also similar, with the liberation size of the US2 sample slightly coarser than the US1 sample and the liberation size of the LSS is between that of the US1 and US2. The same crushing process used for US1 will be suitable for US2 and LSS material.

Based on these results the metallurgical performance of the US2 and LSS material is expected to be similar to that of the US1 material.

Marine

Bathymetric conditions support a transshipment solution that involves loading the potash from a jetty into barges, which transfer the product to Handimax and Panamax class vessels anchored approximately 6 nautical miles (11 km) from the coast. These barges are loaded from a 750 m long jetty, which supports the conveyor belt that delivers product from the process plant. The jetty will be protected by 250 m long breakwater structure.

Solid Residue and Brine Management

Two waste products, salt and insolubles, will be generated from the process and will be treated separately.

The insoluble residue will be pumped into a valley type impoundment in close proximity to the process plant, which will be raised in a downstream manner in approximately 3 year increments. The Residue Storage Facility (RSF) has been designed to contain a total of approximately 1.49 Mt of insoluble residue. This is based on a production rate of 7.74 tph insoluble residue and Life of Mine (LoM) of 22 years. The RSF can accommodate approximately 1.66 Mm^3 of insoluble residue. The RSF will be synthetically lined to prevent any seepage into the environment. Supernatant recovered from the RSF will be pumped to the process plant for use as makeup water, with any excess solution pumped to the brine distribution tank.

The salt brine, generated as a waste product in the process plant, will be dissolved and diluted with seawater before being discharged into the ocean. Ocean disposal of highly saline brines is an accepted practice for desalination plants and has also been approved for the proposed MagMinerals operation in the ROC based on compliance with International Finance Corporation (IFC) effluent guidelines. Ocean disposal relies on post-discharge dilution; this approach is considered generally acceptable because the brine is a concentrated version of the elements that are already present in seawater. Beyond the natural mixing zone the brine is undetectable. The salt brine will be diluted with sea water, resulting in a maximum salt content of 125 grams per litre (gpl) following dilution. The diluted sea brine will be pumped via a pipe attached to the jetty, and discharged into the ocean via diffusers installed in the seabed; this is a standard approach for desalination plants. Numerical modelling calculated that the salt concentration will be reduced to within ±2.4 gpl of the background ocean salt concentration within 250 m of the diffusers.

General Infrastructure

The Sintoukola Project's general infrastructure includes mine site facilities, haul road and road train, process site facilities, employee facilities, and infrastructure (which consists of power, natural gas, water supply and the access roads to the sites).

Mine site facilities include the preparation of the overall platform, ancillary buildings and utilities (power, water, lighting and, waste management). Ore will be transported from the mine site to the process plant located on the coast along a 36 km dedicated haul road, using a fleet of 21 road trains.

Process site facilities include the preparation of the overall platform, ancillary buildings and utilities (power, water, fuel, lighting and, waste management).

The employee facilities are located approximately 5 km from the process plant and will accommodate approximately 950 people.

Electrical power will be sourced from the ROC national grid. A 220 kV transmission line will be constructed from the Mongo Kamba II substation south of Pointe Noire to the process plant, employee facilities and mine site distance of 92 km. The power demand is estimated to be 24 MVA at the mine site and 32 MVA at the coastal site. The natural gas needed for product drying will be supplied by an 81 km long pipeline from the gas treatment plant at Cote-Mateve, 10 km south of Pointe Noire.

Fresh water will be supplied from wells located at each site. The seawater required for process makeup water and for dissolution as part of the salt brine management will be supplied from seawater intake installed approximately 400 m from the shoreline.

Social and Environment Impact Assessment

A comprehensive Social and Environmental Impact Assessment (SEIA) that meets national and international requirements is being undertaken. The SEIA is at an advanced stage, with social and biophysical baseline field studies nearly completed and associated reports currently being drafted. A comprehensive understanding of baseline conditions has been developed through the field studies and analysis of data; this will form the basis for the subsequent impact assessment and development of mitigation measures.

As the project is located in a sensitive biophysical and social environment, opportunities to minimize negative impacts have been identified and integrated with project design throughout the prefeasibility phase. At the current stage in the SEIA process (pre-impact assessment) it appears that the majority of potential environmental impacts identified can be readily managed through the implementation of standard environmental management plans. However, several material negative risks have been identified, including some related to social and community issues, which will warrant specific management measures in order to avoid project delays and reputational damage.

The material risks relate to the environmental permitting process; meeting the expectations of conservation non governmental organizations (NGOs) and other stakeholders with respect to protecting biodiversity in the project area; delays in land acquisition, resettlement and compensation (all of which are government-led processes); the need to build trust and constructive relationships with key stakeholders (in particular local communities and indigenous peoples); engagement and partnering with government authorities to manage influx of people to the project area and effectively managing road safety on the upgraded service road and pedestrian access to the dedicated haul road.

ELM is aware of these material risks and is developing a range of approaches to address them and manage the potential impacts on the project. While the material negative risks are significant, with the implementation of appropriate mitigation measures and proactive management by ELM, they should not represent fatal flaws for the project. Effective stakeholder engagement must remain a core element of ELM's mitigation and management plans throughout the feasibility study and the subsequent project lifecycle (from construction to closure).

The SEIA should be completed as originally programmed, with submission of the national SEIA report in December 2012 and the international report in February 2013. The impact assessment process and national and international SEIA reports should be updated on completion of the feasibility study; the cost of updating will depend on the nature and extent of any significant changes to the project description.

Marketing and Economics

ELM utilized market research from CRU International Limited (CRU) and Fertecon Limited (Fertecon) to develop its potash marketing strategy. Both companies are highly respected independent potash commodity research analysts, utilized by potash industry participants. The reports covered all aspects of potash supply, demand, marketing, potash logistics and pricing.

Both companies foresee a strong growth of both demand and supply in potash in the next decade.

Five target markets were considered in detail by CRU (Brazil, China, India, South-East Asia and South Africa). The research indicates that ELM should focus exports on Brazil and South Africa, mainly due to highly competitive freight rates and expected ease of entry into these markets. India, China and South-East Asia, however, also offer promising options, with SPSA being more competitive than most other major producers.

For the purpose of the PFS, Brazil is therefore considered to be SPSA's target market. Projected growth in Brazilian potash demand is sufficient to absorb all Sintoukola production.

The preferred product in the Brazilian market is granular material, which will form the bulk of the production from the Sintoukola Project, at a targeted MoP grade of 60.5% K_2O. Fertecon provided

CFR price projections for the Brazil market through 2020. Thereafter, the 2020 price was kept constant.

Capital costs were developed for each scope area based in US dollars (US$). These estimates allowed for direct costs only and SRK included an allowance for indirect costs in the economic modelling. For the purpose of the Sintoukola PFS, initial capital is defined as any capital spent during construction and the ramp up period, while any capital spent after reaching nameplate capacity (2.0 Mtpa MoP), referred to as production start, is considered sustaining capital, unless otherwise specified.

Capital costs for all disciplines are summarized in Table 5.

Table 5: Capital Cost Summary

Description	Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Mining	352,569	178,852	531,420
Haul Road & Road Trains	115,664	35,694	151,358
Processing	535,825	154,621	690,446
Marine	124,639	45,525	170,164
Subtotal Waste & Brine	42,435	15,777	58,212
Employee Facilities	47,383	12,637	60,021
General Infrastructure	97,679	70,459	168,138
Owner's Costs	59,236	24,143	83,336
Subtotal Capital Costs	1,375,430	537,709	1,913,139
Contingency	252,464	97,615	350,079
Subtotal Capital + Contingency	1,627,894	635,324	2,263,218
EPCM	203,543	-	203,543
Insurance	19,540	-	19,540
Capital Expenditures - 2Q 2012	1,850,977	635,324	2,486,301

The operating costs presented in Table 6 are extracted from the economic model, which is based on the mine schedule. LoM operating costs are estimated at US$79.71 / tonne of product free on board (FOB).

Table 6: Life of Mine Operating Costs by Discipline

Item	LoM (US$ 000s) (Q2, 2012)	Cost - (US$ / tonne MoP) (Q2, 2012)
Mining	1,243,410	27.67
Hauling and Road Trains	590,230	13.14
Processing	1,057,014	23.53
Marine	75,830	1.69
Solid Residue and Brine Disposal	27,671	0.62
Employee Facilities	228,834	5.09
General Infrastructure	89,929	2.00
Environmental	268,621	0.29
Owner Costs	3,581,539	5.69
Total Operating Cost	1,243,410	**79.71**

The post tax economic analysis shown on Table 20.4.1 indicates the following:

- Net present value for the project at a 10% discount rate (NPV_{10}) of US$2,971M;
- Internal rate of return (IRR) of 29.3%; and

- A pay back period from construction startup of 6 years.

Implementation

ELM intends to continue directly to the feasibility study (FS) upon completion of the PFS. ELM will strengthen the Owner's Team during the FS and early works. The Engineering, Procurement, and Construction Management (EPCM) approach to be followed by ELM will be addressed during the FS and achieves the following key milestones:

- Complete FS: Q3, 2013;
- Full construction implementation commences: Q4 2013;
- Mining commences: Q1, 2016;
- First product shipped: Q3, 2016; and
- Achieve nameplate capacity (2 Mtpa MoP): Q1, 2017.

Conclusions and Recommendations

Geology and Resources

The August 21, 2012 Mineral Resource estimate represents a steady advancement in both the understanding of the geological setting and the delineation and estimation of Mineral Resources. The updated Mineral Resource estimate is based upon high quality analytical, stratigraphic-structural interpretation of drillhole data and high-density 2D seismic data. The geological framework as interpreted from historic drillholes, was refined following the 2011 drilling and has been reconfirmed and further differentiated by subsequent exploration programs. The combination of grid drilling (1 to 2 km hole spacing) and detailed 2D seismic data allowed for confident interpretation of the subsurface environment and the geometry of the host stratigraphy.

The result of this work is a substantial increase in the Mineral Resource estimates and raised confidence in the geological model which allowed an upgrade in the classification of the Mineral Resource estimates to Measured, Indicated and Inferred as listed in Tables 1 and 2.

In CSA's opinion, the work completed by ELM over the past three years has substantially advanced the understanding of the Kola deposit resulting in improved confidence in the development potential of this significant potash resource.

During the Phase 3 exploration program it is recommended that ELM complete the following:

- Improve the processing of the seismic data to enhance the imaging of the reflectors and areas of discontinuity;
- Acquire and process new 2D reflection seismic data within the areas currently defined as Indicated and Inferred Resources and new exploration targets;
- Complete a trial 3D seismic survey to support the existing 2D interpretation;
- Continue exploration to further investigate the HWS potential;
- Conduct testwork to enhance understanding of density, petrography, mineralogy and geochemistry; and
- Undertake umpire analyses using classical analytical techniques.

Mining and Mineral Reserves

The Sintoukola PFS underground mine design and production schedule presents an estimate of the ore that can be economically and safely extracted from the geologic model. Only Ore Reserves determined from Measured and Indicated Resources are evaluated in the design.

The room and pillar mine design that is applied to the orebody is a widely used method in the industry and has been used successfully in other similar deposits. CMs are a proven technology and have been operating potash mines worldwide for many years. Civil engineering shaft sinking methods will be utilized, involving less front end work and procurement of special hoisting equipment. A ground freeze will be utilized for shaft development. Material will be removed from the shaft and mine using a HAC and industrial lifts will be used for equipment access.

The mine production schedule achieves a consistent production rate within a reasonable ramp up period. The schedule also delivers a tonnage and grade suitable to the process and material balance can be smoothed by using the surface stockpile when necessary.

The following recommendations are made regarding future work:

- Proceed with a more detailed design as part of the FS, including:
    - Shaft design,
    - Panel layout planning,
    - Ventilation and cooling design,
    - Conveyor design and optimization, and
    - Ground support and pillar design.
- Shaft sinking contractor should be involved in this phase to optimize shaft sinking and design;
- A geotechnical and hydrogeological testwork program should be completed at the shaft site prior to shaft detail design;
- Long lead time capital equipment should be ordered in a timely manner; and
- Key underground technical and management staff should be recruited to optimize the detailed design phase.

Metallurgy and Process

AMEC has designed a conventional flotation plant for the Sintoukola Project based on test work, projected mining grade and process simulation (mass balance). Due to the low insoluble content of the ore the design incorporates a simpler insoluble removal process than a typical Canadian flotation plant. The process plant is designed to produce 2 Mtpa of MoP at a grade of 60.5% K_2O. The process simulation resulted in a recovery of 89.5%, based on a mining grade of 19.8% K_2O (31.3% KCl).

It is recommended that the following metallurgical testwork be conducted:

- Complete phase 2 X-Ray Diffraction (XRD) analysis and insoluble removal and phase 3 (rougher and regrind flotation) of the proposed test program for the LSS, FWS and HWS;
- Conduct a test program on a composite sample consisting of material from the US, LS, HWS and FWS once the mine plan has been developed in further detail. The program should include liberation testing, XRD analysis and insoluble removal and rougher and regrind flotation testing; and

- Conduct a test program to investigate the feasibility of filtering the insoluble.

Solid Residue and Brine Management

Specific key recommendation for the insoluble and brine management includes:

- Dewatering of the slimes should be investigated for the FS stage of design; and
- Develop a 3D model of the brine dispersion, including all trace elements contained in the salt brine to check compliance with the IFC effluent discharge guidelines.

Infrastructure

The infrastructure for the Sintoukoloa Project has been analysed, defined and sized to satisfy all direct or indirect requirements for the production of 2.0 Mtpa MoP.

The infrastructure design aligns with the process plant design that has been developed by AMEC and the mine production plan that has been developed by SRK.

Specific key recommendations for infrastructure include the following tasks:

- Launch near shore geotechnical investigations to improve and refine the design of the jetty foundations;
- Conduct navigational and ship-mooring simulations to verify if navigational safety requirements are achieved with the proposed attached breakwater layout;
- Carry out shoreline evolution modelling to confirm the influence of the breakwater on coastal sediment dynamics;
- Perform a hydrogeological investigation at the coastal site to confirm the potential of the ground water sources; and
- Perform further investigations to determine the salinity levels of the surface raw water sources if the hydrological survey does not provide satisfactory results.

Marketing and Economics

It is recommended that a detailed product marketing strategy be developed by ELM to evaluate the optimal method to penetrate the various product markets. Sales and marketing costs will also be evaluated in more detail.

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Appendices

1 Introduction (Item 2)

1.1 Terms of Reference and Purpose of the Report

Elemental Minerals Ltd (ELM) commissioned SRK Consulting (U.S.), Inc. (SRK), AMEC Americas Limited (AMEC), EGIS International (EGIS) and CSA Global Pty Ltd (CSA) (collectively the Consultants) to prepare a Technical Report compliant with Canadian National Instrument 43-101 (NI 43-101) for a Prefeasibility Study (PFS) of the Sintoukola Potash Project (Sintoukola Project) located in the Republic of Congo (ROC). The quality of information, conclusions, and estimates contained herein are consistent with the level of effort involved in the Consultant's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by ELM subject to the terms and conditions of its contract with the Consultant's and relevant securities legislation. The contract permits ELM to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with ELM. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

This report provides Mineral Resource and Mineral Reserve estimates, and a classification of resources and reserves in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum Standards on Mineral Resources and Reserves: Definitions and Guidelines, November 27, 2010 (CIM).

1.2 Qualifications of Consultants

The Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, geotechnical engineering, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in ELM. The Consultants are not insiders, associates, or affiliates of ELM. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between ELM and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

The following individuals, by virtue of their education, experience and professional association, are considered Qualified Persons (QP) as defined in the NI 43-101 standard, for this report, and are members in good standing of appropriate professional institutions. The QP's are responsible for specific sections as follows:

- Dr Simon Dorling (Principal Consultant Geologist with CSA Global Pty Ltd in Perth, Western Australia) is the QP responsible for property description, history and ownership, geology and mineralisation, drilling and sampling, in the Executive Summary, Sections 2.1, 2.2, 2.3, 2.3.1, 3.1, 4, 5 (excluding 5.10), 6, 7, 8 (excluding 8.3.7), 9, 10, 21, 23.1, 24.1, and 25.1;

- Dr. Andrew Scogings (Associate Geologist of CSA Global Pty Ltd in Perth, Western Australia) is the QP responsible for the mineral resource estimate in the Executive Summary and Section 12;
- Paul O'Hara (Manager, Process, AMEC Americas Limited, Saskatoon Office) is the QP responsible for metallurgy and process in the Executive Summary, Sections 11, 15, 19.1.3, 19.2.3, 23.4, 24.4 and 25.4);
- Jean Hector, Senior Geologist (EGIS) is the QP responsible for marine, brine management and general Infrastructure in the Executive Summary, Sections 3.2, 3.5, 16.1, 16.2, 16.3, 16.4, 16.5, 16.6 (excluding 16.6.4), 16.7.2, 19.1.2, 19.1.4, 19.1.5.2, 19.1.6, 19.1.7, 19.2.2, 19.2.4, 19.2.5.2, 19.2.6, 19.2.7, 23.3, 23.5, 23.6.2, 23.7, 23.8, 24.3, 24.5, 24.6.2, 24.7, 24.8, 25.3, 25.5, 25.6.2, 25.7, and 25.8;
- Johan Boshoff (Principal Geotechnical Engineer, SRK Consulting Ltd, Australasia) is the QP responsible for the solid residue in the Executive Summary, Sections 16.7.3, 19.1.5.1, 19.2.5.1 23.6.1, 24.6.1 and 25.6.1;
- Jane Joughin (Principal Environmental Scientist, SRK Consulting (UK) Limited) is the QP responsible for environmental and social impact assessment in the Executive Summary, Sections 3.3, 3.4, 16.7.1, 18, 23.9, and 25.9; and
- Dr. Neal Rigby (Corporate Consultant, Mining, SRK Consulting (U.S.), Inc. in Denver, Colorado), is the QP responsible for mining and Mineral Reserves, and marketing and economics in the Executive Summary, Sections 1, 2.4, 2.5, 2.6, 2.7, 5.10, 8.3.7, 13, 14, 16.6, 16.8, 17, 19.1, 9.1.1, 19.1.8, 19.1.9, 19.1.10, 19.2, 19.2.1, 19.2.8, 19.2.9, 20, 22, 22.1, 22.2, 23.2, 23.10, 24.2, 24.10, 24.11, 25.2, 25.10, 25.11, 26 and 27.

The Sintoukola Project team organization structure is presented in Figure 1-1. Certificates of Authors are presented in Appendix A.

1.3 Sources of Information

SRK relied on others for the following information presented in this report:

- AMEC: Metallurgy and process;
- CSA: Property description, history and ownership, geology and mineralisation, drilling and sampling and mineral resource estimate;
- EGIS: marine, brine management and general infrastructure; and
- ELM.

1.4 Reliance on Other Experts (Item 3)

The Consultant's opinion contained herein is based on information and data provided by ELM, information in the public domain, observations and data obtained during the site visits, as well as documents referenced in Section 19. The Consultants have relied upon the work of other consultants in the project areas in support of this Technical Report.

Descriptions of the Sintoukola Project tenure were provided to Consultants by the issuer. ELM has warranted to the Consultants that the information provided for preparation of this report correctly represents all material information relevant to the Sintoukola Project. ELM has taken reasonable measures to ensure that title to its properties are in good standing, including obtaining a legal title opinion with respect to validity of the relevant Sintoukola Project licenses and agreements (PWC, 2011). No attempt to independently verify the land tenure information was made by SRK.

The Consultants used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information, which required subsequent calculations to derive subtotals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the Consultants do not consider them to be material.

1.5 Site Visits

The following QP's conducted site visits to the Sintoukola Project on the dates indicated:

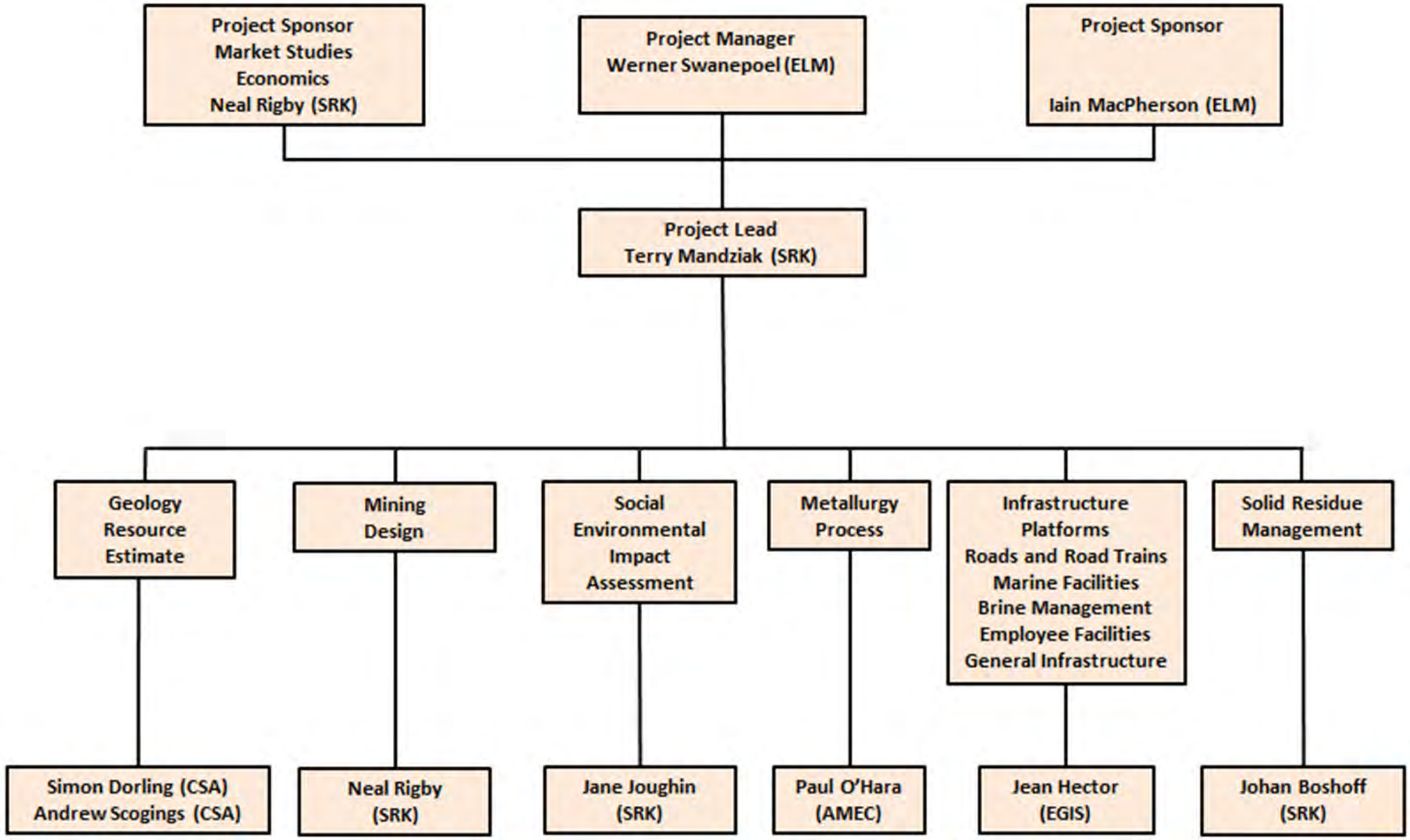
- Dr Simon Dorling completed four site visits to the Sintoukola Project during the past three years. The initial site visit was made in August, 2010 and then additional visits in May 2011 and April 2012. The most recent visit was undertaken in September 2012. The initial site visits was to observe the commencement of the first phase of exploration (Phase 1) at the Sintoukola Project and ensure adherence to exploration protocols, to assess results, collect information and discuss the ongoing activities with site personnel. During subsequent visits the focus shifted on to technical aspects of the program and verification of operational procedures;
- Jean Hector completed two site visits to the Sintoukola Project area. The first site visit was made in December, 2011 and the second site visit was in March 2012. The first site visit was to view the general location of the property and site conditions. The second site visit was to investigate the possibility to open an aggregate quarry for construction and to obtain quotes from local aggregate suppliers;
- Paul O'Hara visited the site on September, 2011 to view the general location of the property and site conditions; and
- Johan Boshoff of SRK undertook a site visit in March 2012 to inspect the five potential RSF sites and to identify any potential fatal flaws associated with these locations. All the potential RSF locations were inspected; with the exception of one site which was located across a floodplain thus making access impossible. In addition to identification and visual inspection of potential RSF sites, the site visit was used to confirm, modify and add to the site ranking criteria.

1.6 Effective Date

The effective date of this report is September 17, 2012.

1.7 Units of Measure

The metric system has been used throughout this report. Tonnes are metric of 1,000 kilograms (kg), or 2,204.6 lb. All currency is in U.S. dollars (US$) unless otherwise stated.



srk consulting

elemental minerals limited

Sintoukola Potash Project
Republic of Congo

Source: SRK, 2012

Figure 1-1

Project Organizational Chart

2 Property Description and Location (Item 4)

The Sintoukola Project includes the Kola deposit which is a potash project situated in the Kouilou Province in the south west corner of the ROC. The Kola deposit is a shallow, high-grade, near-term sylvinite deposit currently in the PFS phase of evaluation.

2.1 Property Location

At its nearest point, the Sintoukola permit area is approximately 50 km to the north of the major West-African port of Pointe Noire and extends along the coast north west towards the border with Gabon and inland for some 40 km as illustrated by Figure 2-1. Access to the project area is via a 55 km dual-lane bitumen road from the port city of Pointe Noire, which terminates 9 km outside the permit's southern boundary. From this point, access to the Kola deposit area is via a 35 km gravel track. The main focus of exploration has been on the Kola deposit in the eastern part of the project area between the regional villages of Dougou and Kola.

The project comprises an exclusive exploration permit, for potash and associated salts. The permit covers an area of 1,436.5 km^2 bounded by the corner coordinates provided in Table 2.1.1 and the coastline (see Figure 2-1).

Table 2.1.1: Exploration License Co-ordinates

Point	Longitude	Latitude
A	11^o 11' 53" E	4^o 00' 00" S
B	11^o 41' 37" E	4^o 00' 00" S
C	11^o 48' 00" E	4^o 20' 00" S
D	11^o 36' 10" E	4^o 22' 13" S

2.2 Republic of Congo

The ROC is situated on the western edge of the African Continent and borders the countries of Angola (Cabinda Province), The Democratic Republic of the Congo, The Central African Republic and Gabon. It has a total land area of 342,000 km^2 and a population of approximately 4.4 million people of which 45.6% are less than 14 years of age and an estimated 84% are literate (CIA World Fact Book, 2012).

The ROC is a former French colony which was granted independence in 1960. From 1960 to 1990 the country was effectively Marxist led, though a democratically elected government took office in 1992. In the early 1990's the country was subject to a civil war which finally ended with the signing of a peace accord between the government and the southern based rebels in 2003.

Oil is the predominant export of the ROC with oil production of 302,200 in 2010, of which over 211,000 are exported. Oil accounts for more than 2/3 of the ROC's Gross Domestic Product (GDP) (US$12 billion in 2010), 85% of its fiscal revenues and 87% of its total exports. The ROC Government wants to diversify away from reliance of oil and has recently agreed to investments of US$3 billion by ENI (Italian Petroleum Company) in biofuels and US$1.2 billion by Mag Industries (MAA - TSX) to develop the Mengo Project.

2.3 Mining Code and Tenure in the ROC

The ROC Mining code is based on the ROC Constitution dated January 20, 2002 and is enacted through the issuing of law 23/92 on July 7, 1982 and it's decree 86/814 issued on June 11, 1986 which implemented the mining code and the subsequent law 4/2005 on April 11, 2005 which enacted the mining code. These laws stipulate that the state owns all mineral resources contained within its borders, but exploration and exploitation of these resources is allowable by private companies through the issuing of mining permits which are granted by decree.

Mining tenure in the ROC is obtained in three stages:

- Authorisation to Prospect *(L'authorisation de prospection)* which is non-exclusive, allows for non-invasive prospecting for a period of 12 months which can be extended for a further 12 months by application. During the authorisation to prospect the applicant can apply for further minerals to be added to any exploration permit applied for;
- Exploration Permit *(Permis de recherché minieres)* gives the holder exclusive rights to explore for minerals to an unlimited depth over a defined area. The Exploration Permit is valid for a period of three years which can be extended twice, each for a period of two years. The permit can be cancelled by the Minister of Mines if no exploration activities are commenced within a period of nine months from the granting of the permit; and
- Exploitation Permit *(Permis d exploitation)* gives the holder the exclusive right to exploit mineral substances to unlimited depth as defined by the Permit. The permit is valid for a maximum period of 25 years which can be extended for a further maximum period of 15 years. In the event that the permit holder has not commenced development of the project within a period of 12 months after the granting of the Exploitation Permit the Council of Ministers can decide to withdraw such a permit.

2.3.1 Nature and Extent of Issuer's Interest

The exclusive mineral exploration permit, "Sintoukola", was awarded to Sintoukola Potash S.A. (SPSA) through Presidential Decree No 2009-237 dated August 13, 2009 which was published in the Congolese Official Gazette No 35 on September 27, 2009. The Sintoukola exploration permit has been awarded for an initial period of three years and can be renewed twice for two extension periods of two years. ELM has informed CSA that it has submitted a permit renewal application on May 7, 2012. Receipt of application was confirmed by the government agency on the same day. In its application, ELM proposed a tenement reduction totalling approximately 15,760 ha (157.6 km^2). ELM was formally advised on August 22, 2012 that the renewal has been delivered to the Cabinet for ratification.

SPSA is a company registered in the ROC whose only asset is the Sintoukola exploration permit. SPSA is a private company whose shareholders are:

- ELM (ABN 31108066422) of 14 Emerald Terrace, West Perth, Western Australia (ELM);
- Les Establissements Congolais MGM of 42 Rue Bouzala, Moungali, Brazzaville, Republic of Congo (MGM); and
- Tanaka Resources (Proprietary) Ltd. (Reg No. 2002/008115/07) of number 114 11th Street, Parkmore, Sandton City, 2146 in the Republic of South Africa (Tanaka).

ELM holds 93% of the equity in SPSA with MGM and Tanaka holding equity positions of 5% and 2%, respectively. ELM holds a right of first refusal over the equity held in SPSA by the minority shareholders.

2.4 Royalties, Agreements and Encumbrances

The Sintoukola Project is held under an Exploration Permit and is operated in accordance with an Exploration Agreement which does not stipulate any royalty. However a 3% mining royalty based on Earnings Before Interest Tax Depreciation Amortization (EBITDA) is due during the exploitation phase upon the granting of the mining license.

ELM has not yet submitted an application for an Exploitation Permit. SRK has been advised that ELM intends to submit an application for an Exploitation Permit upon completion of the Congolese Social and Environmental Impact Assessment (SEIA) (as discussed in Section 18.4.2) and this PFS report.

SRK has been informed by ELM that there are no back-in rights or encumbrances material to the Sintoukola Project.

2.5 Environmental Liabilities

ELM and its subsidiary, SPSA, comply with the requirements permitting access to the surface without restriction assuming adherence to the ROC Mining Code (Code Minier, Law No.4-2005 of April 11, 2005) and the environmental impact assessment procedure decree (Decree No. 2009-415 of November 20, 2009).

Since granting of the permit, neither the Mining Code nor the laws controlling access to the permit area have changed. ELM is in compliance with the provisions of Law No.003/91 of April 23, 1991 on the Protection of the Environment.

ELM has engaged various environmental consultants to manage the preparation of Environmental Impact Assessments (EIA) and compliance with environmental aspects of the Exploration Agreements for the activities at the Sintoukola Project. Current environmental liabilities are limited to cut lines for drilling and seismic access, drill pad clearings, mud sumps and various temporary infrastructure.

In terms of the approved environmental management plans in the exploration EIA, exploration activities at the Sintoukola Project will comply with the environmental provisions of the Mining Code, viz:

- The rehabilitation of the surface soil or other areas adjacent to the mine or deposit in accordance with a rehabilitation plan or land use; concurrently or with other work required in case of closure or cessation of work (S. 128);
- The reinstatement of forests or other areas whose integrity has been impaired as a result of mining activities (S.129); and
- The work of exploration or exploitation of a mine or quarry will be in compliance with the obligations relating to (S.132):
    - safety and health of personnel and the population;
    - protection of the environment;
    - preservation of the mine;

- o conservation of buildings, ground safety and soundness of dwellings; and
- o maintaining open channels of communication.

2.6 Required Permits and Status

SPSA has received the following permits as part of the ongoing exploration program:

- Authorisation from the Ministry of Development, Economic Forestry and the Environment (No 000317/MDDEFE/CAB-CEDD, dated February 25, 2010) to access and commence exploration activities;
- Authorisation from the Ministry of Energy and Water to drill a water drillhole and extract 10,000L per day (No 070/MEH-DGH dated June 16, 2010); and
- Certificate of Conformation No 001435/MDDEFE/CAB/DGE/DPPN approving the ELM's EIA for the exploration phase.

SPSA has, in collaboration with PricewaterhouseCoopers (PWC) in the ROC, identified the most probable rights and permits that will need to be obtained in order to construct, implement and operate the project. The permits are listed in Table 2.6.1, supported by the applicable legislation, the responsible government department and the current status. SPSA is evaluating the permitting needs on a continuous basis in close collaboration with PWC and Herbert Smith (an international law firm specializing in African natural resource legal issues), as well as the ROC Government as part of the ongoing permitting process.

Table 2.6.1: Permits to be Obtained

Permit ***[Permit Requirements]***	**Legislation**	**Issuer**	**Status**
Mining License *SEIA approval* *PFS approval*	Mining Code (Law No. 4-2005 of April 11, 2005 Decree No. 2007-274 dated May 21, 2007 providing requirements for prospection, research and exploitation of mineral substances and administrative surveys Environmental Code (Law No. 003/91 of April 23, 1991)	Ministry of Mines and Geology Ministry for Sustainable Development, Forestry and the Environment	Application to be submitted
Permit to import material and equipment customs duty & tax free *Signing Investment Agreement*	Mining Code (Law No. 4-2005 of April 11th, 2005 Investment Charter (Law No. 6-2003 of January 18th, 2003) Decree No. 99-167 dated August 21, 1999 Minute (Note Circulaire) No. 088/MFBPP-CAB dated December 30, 2010 & Memo No. 293/MEFB/DGDDI dated July 17, 2006	Ministry of Finance, Budget and Public Portfolio Ministry of Mines and Geology Departmental Direction of customs / COTECNA Ministry of Commerce	Application to be submitted
Permit to export Muriate of Potash customs duty and tax free	Order No. 7660 dated September 10, 2009 relating to the control of minerals exportation out of the	Ministry of Finance Ministry of Mines and Geology	Application to be submitted

Permit ***[Permit Requirements]***	**Legislation**	**Issuer**	**Status**
Signing Investment Agreement	ROC	Departmental Direction of customs Ministry of Trade and Supply Chamber of Commerce General Direction of Money & Credit (DGMC) Bureau Veritas	
Prefectural Order to access, occupy and use land, offshore, sea shore for construction and exploitation: *Issuing Decree of Public Utility Declaration (DUP)* *Public enquiry* *Plot enquiry* *DUP Commission* *Signing Investment Agreement*	Mining Code 2005 (Law No. 4-2005 of April 11, 2005) Article 104 to 117 Circular Letter No. 295/MDIMEA/CAB dated October 29, 2002, prescribing the requirements to implement an Industry or carry out Industry activity in the ROC Land Code (Law No. 9-2004 dated March 26, 2004) articles: 49-52-53-64 Law No. 9-2004 dated March 26, 2004 relating to land affairs general principles Law No. 11-2004 and Decree 2005-516 relating to expropriation due to public utility Law No. 24/2008 dated September 22, 2008 on land regime in urban areas Law No. 28-11 dated 3 June 2011 relating to the office and monitoring body of cadastral works Environmental Code (Law No. 003/91 of April 23, 1991)	Kouilou Prefecture Ministry of Mines and Geology Ministry of Industry General Direction of Industry Ministry of Land Affairs and of Public Domain Ministry for Sustainable Development, Forestry and the Environment Ministry of Construction, urbanism and habitation	Identification of land in progress
Permit to use of air transports and fly over project areas *Signing Investment Agreement*	Congolese Aeronautic Regulations (RAC)	Ministry of Civil Aviation National agency of Civil Aviation	Application to be submitted
Permit to construct and exploite communication system infrastructure *Signing Investment Agreement*	Law No. 14-97 dated 26 May 1997 regulating the electronic communications in the ROC Decree No. 99-188 of 29 October 1999 relating to the conditions of licensing, establishment and operation networks and telecommunications services;	Ministry of Post, telecommunicationsand new technologies of communication Electronic Communication Regulatory Agency (ARPCE)	Application to be submitted

Permit *[Permit Requirements]*	**Legislation**	**Issuer**	**Status**
	Law No. 9-2009 dated on November 25th, 2009, regulating the electronic communications in the ROC Law No. 20 – 2010, dated December 2010, relating to the 2011 Finance Law providing fees, taxes and royalties to be paid in the Post and electronic communications sector Ministerial order referenced No. 11279 /MPTNTC/MEFB fixing the duties, taxes, charges and fees relating to the establishment and use of networks and telecommunication services	Directorate General Headquarters Posts and Telecommunications;	
Working permit (ATE)	Labour Code (law No. 45-75 of March 15, 1975) Law No. 022/88 of September 17, 1988 (ONEMO) *Article 25 & subsequent. Law No. 23-96 of June 6, 1996 regarding Expatriates conditions of work in ROC	Ministry of Labour and Social Welfare Employment & Manpower National Agency (ONEMO)	Application to be submitted
Permit to construct and use an electrical power line *Signing of Investment Agreement* *SEIA & EMP approval* *Detailed design approval* *Transfer of ownership of the power grid to the Public Company of Electricity (SNE)*	Mining Code 2005 (Law 4-2005 of April 11, 2005)* Article 104 & subsequent Act No. 6-67 of June 15, 1967 establishing the Public Company of Energy (S.N.E) Electrical Code (Law No. 14-2003 of 10 April 2003) Law No. 15 -2003 April 10, 2003 establishing the public agency of rural electrification Law No. 16-2003 of April 10, 2003 establishing the regulation agency of the electricity sector Law No. 17-2003 of April10, 2003 establishing the development fund of the electricity sector Decree No. 84-403 of April 23, 1984 approving the statutes of the public energy company (S.N.E.) Decree No. 2007-290 of May 31, 2007 approving status of the regulatory agency of the electricity sector	Ministry of Mines and Geology Ministry of Land Affairs and Public Domain Ministry of Construction, urbanism and habitation Ministry of Energy & Water Ministry for Sustainable Development, Forestry and the Environment Minister for Infrastructure and Public Works Public Company of Electricity (SNE) Minister of Economy, Planning, Spatial Planning and the Integration Minister of Finance, Budget and Public Portfolio	Application to be submitted

Permit ***[Permit Requirements]***	**Legislation**	**Issuer**	**Status**
	Decree No. 2007-291 of May 31, 2007 approving Statutes of the Public Agency for Rural Electrification Decree No. 2008-560 of November 28, 2008 approving the Statute of the development fund of the electricity sector Decree No. 2010-123 of February 19, 2010 on the Minister of Energy and Water's functions Decree No. 2010-241 of March 16, 2010 on the organization of Ministry of Energy and Water Decree No. 2003-156 of August 4, 2003 on the organization and functions the General Directorate for Energy Decree No. 8215 of December 16, 2005 on the establishment, functions,organization and functioning of the monitoring committees of projects Decree No. 8216 of December 16, 2005 on the establishment, functions, organization and operation of steering committees projects		
Permit to construct and exploit of a gas pipeline *Signing of Investment Agreement* *SEIA & EMP approval* *Detailed design approval*	Mining Code 2005 (Law No. 4-2005 of April 11, 2005)* Article 104 & subsequent Environmental Code (Law No. 003/91 of April 23, 1991) Land Code (Law No. 9-2004 dated March 26, 2004) Hydrocarbon Code	Ministry of Mines and Geology Ministry for Sustainable Development, Forestry and the Environment Ministry of Land Affairs and Public Domain Ministry of Construction, urbanism and habitation Ministry of Infrastructure and Public Works Ministry of hydrocarbon Minister of Finance, Budget and Public Portfolio	Application to be submitted
Permit to construct and use the haulage and service roads	Mining Code 2005 (Law 4-2005 of April 11, 2005)* Article 104 & subsequent Land Code (Law No. 9-2004 of	Ministry of Mines and Geology Ministry of Land Affairs and Public Domain	Application to be submitted

Permit *[Permit Requirements]*	Legislation	Issuer	Status
Signing of Investment Agreement *SEIA & EMP approval* *Detailed design approval*	March 26, 2004)	Ministry of Construction, urbanism and habitation Minister for Infrastructure and Public Works Ministry for Sustainable Development, Forestry and the Environment Minister of Finance, Budget and Public Portfolio	
Permit to construct and exploit marine facilities including the breakwater (dredging type and capacity), the jetty (barges and vessels route and mooring, export terminal) and sea pipelines (seawater intake & brine disposal pipelines capacity) *Signing of Investment Agreement* *SEIA & EMP approval* *Detailed design approval*	Mining Code 2005 (Law 4-2005 of April 11, 2005)* Article 104 & subsequent Land Code (Law No. 9-2004 of March 26, 2004) Environmental Code Law No. 003/91 of April 23, 1991)	Ministry of Mines and Geology Ministry of Land Affairs and Public Domain Ministry of Marine Marchant Minister of Fisheries and Aquaculture Ministry for Sustainable Development, Forestry and the Environment Minister of Finance, Budget and Public Portfolio	Application to be submitted
Permit to construct and exploitIndustrial facilities including the process plant, the residue storage facilities, the service and office buildings. *Signing of Investment Agreement* *SEIA & EMP approval* *Detailed design approval*	Mining Code 2005 (Law 4-2005 of April 11, 2005)* Article 104 & subsequent Circular Letter N° 295/MDIMEA/CAB dated October 29, 2002, prescribing the requirements to implement an Industry or carry out Industry activity in the Republic of Congo Environmental Code (Law No. 003/91 of April 23, 1991) Circular 12/MTE/DGE dated February 15, 2008 relating to the classified installations nomenclature	Ministry of Mines and Geology Kouilou Prefecture Ministry of Industry General Direction of Industry Ministry of Land Affairs and Public Domain Ministry for Sustainable Development, Forestry and the Environment Minister of Finance, Budget and Public Portfolio Ministry of Construction, urbanism and habitation Minister of Industrial Development and Private Sector Promotion	Application to be submitted
Permit to construct and exploit the employee facilities including the catering services and shopping services	Mining Code (Law 4-2005 of April 11, 2005.) Environmental Code (Law No.	Ministry of Mines and Geology Ministry for Sustainable Development, Forestry and the Environment	Application to be submitted

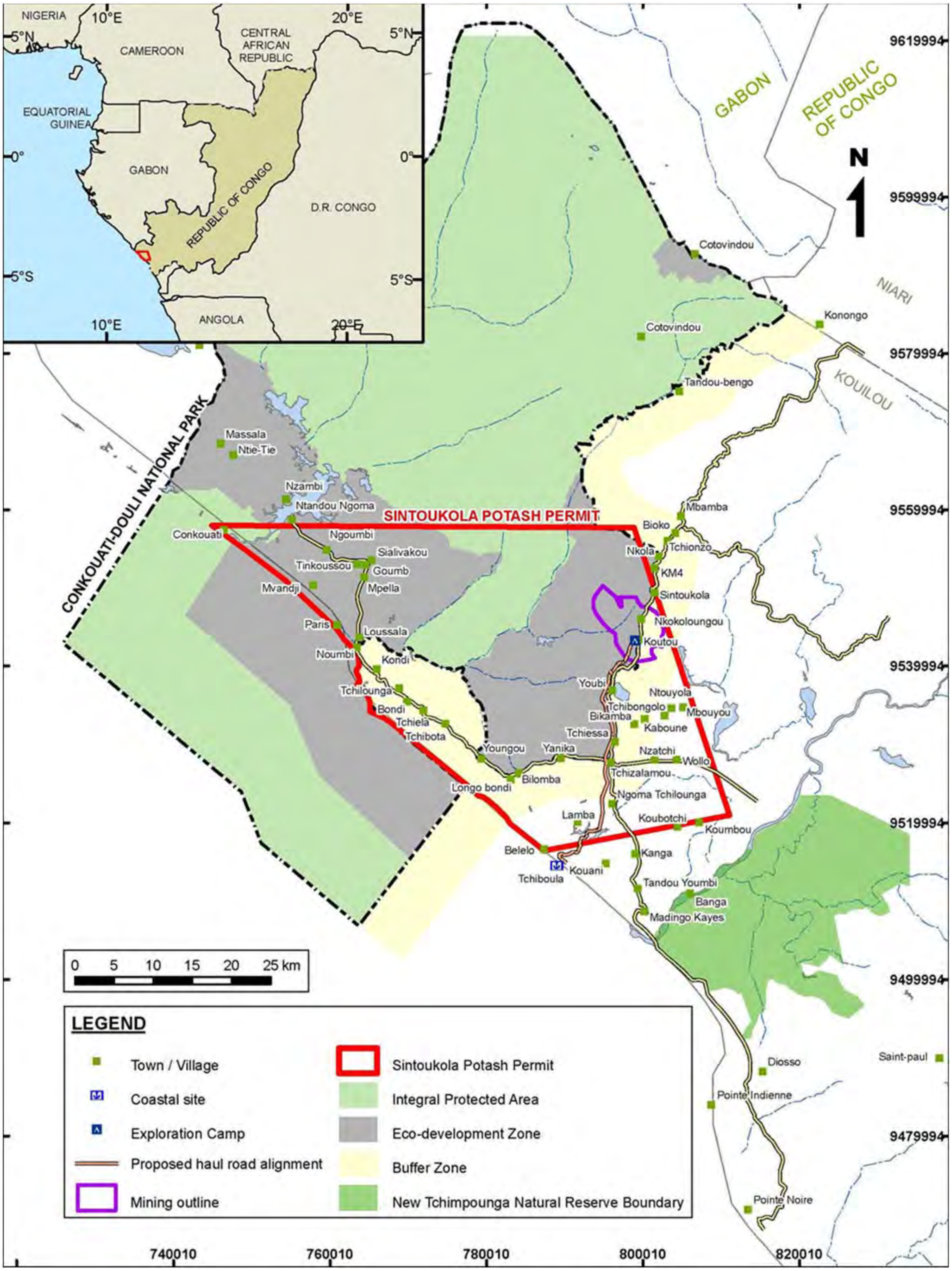
Permit *[Permit Requirements]*	Legislation	Issuer	Status
Signing of Investment Agreement *SEIA & EMP approval* *Detailed design approval*	003/91 of April 23, 1991)	Ministry of Construction, urbanism and habitation Minister of Health and Population	

2.7 Other Significant Factors and Risks

ELM has identified the following risks that may affect access, title or right or ability to perform work at the Sintoukola Project:

- Although SPSA has diligently investigated and believes it has taken reasonable measures to ensure that title to its properties are in good standing, including obtaining a legal title opinion (PWC, 2011), with respect to validity of the relevant Sintoukola Project licenses and agreements, there is no guarantee that title to its properties will not be challenged or impaired by third parties, or that such rights and title interests will not be revoked or significantly altered to the detriment of SPSA;
- Governmental approvals, licenses and permits are current, and may in the future, be required in connection with the Sintoukola Project. To the extent such approvals, licenses and permits are not obtained, SPSA may be curtailed or prohibited from proceeding with planned exploration, development or operation of the Sintoukola Project;
- Although SPSA believes that its development activities are currently being carried out in accordance with all applicable rules and regulations, no assurance can be given that new rules and regulations will not be enacted or that existing rules and regulations will not be applied or amended in a manner that could have a material and adverse effect on the business, financial condition and results of operations of SPSA;
- Delays may be experienced in gaining access to privately owned freehold, communal, state or leasehold land. Delays may be caused by weather, deference to and holders' activities such as cropping, harvesting, calving and musteringand other factors;
- SPSA's operations are conducted in the ROC, and as such, are exposed to various levels of political, economic and other natural and man-made risks and uncertainties over which SPSA may have limited control; and
- Opposition from international or locally based NGOs or other bodies may impact on the ability of SPSA to secure the environmental permits necessary for construction and operation and may also impact on the ability of SPSA to finance the future development of the Sintoukola Project.

ELM and SPSA have a risk management process in place to address these items.



srk consulting

Sintoukola Potash Project, Republic of Congo

elemental minerals limited

Source: ELM, 2012

Figure 2-1

Location of the Sintoukola Project

3 Accessibility, Climate, Local Resources, Infrastructure and Physiography (Item 5)

The Conkouati-Douli National Park (CDNP), which covers an area of 504,950 ha, has a variety of diversified environments that include open savannahs, wetlands, forests and coastlines.

The CDNP covers approximately 55% (790 km^2) of the permit area (1,436.5 km^2). The CDNP is subdivided into two principal zones with varying degrees of environmental protection:

- Integral protected zone – no new resource extraction, agriculture or logging are permitted in this zone; and
- Eco-development zone – sustainable use of natural resources by local communities is allowed, along with logging, agriculture, mining and oil exploration or exploitation (subject to appropriate environmental and social studies and associated permits).

Additionally, a 5 km buffer zone is defined around the borders of the park. Mining and exploration are not permitted in this zone without the prior approval of the relevant authorities.

Thirteen percent (182 km^2) of the Sintoukola permit is contained within the integral protected zone and no activity by SPSA has taken place or is planned in this zone.

Forty two percent (608 km^2) of the Sintoukola permit is contained within the eco-development and buffer zones, where the majority of SPSA's activities have taken place with full authorization from the Minister of Environment and in consultation with the NGO and government representative responsible for managing the CDNP.

The priority exploration targets within the Sintoukola Project are all located in the eco-development or buffer zones.

3.1 Topography, Elevation and Vegetation

The Sintoukola permit area consists of flat to rolling forests and grasslands between the ocean and the inland plateau, which rises to the east of the Sintoukola Project. The Kola deposit lies at an average elevation of 40 m above mean sea level (amsl). The topography is incised by numerous drainage channels, some of which flow into lakes. No major rivers transect the footprint of the Sintoukola infrastructure.

3.2 Access to the Property

Access to the Sintoukola Project mine site (near the village of Koutou) is via National Highway 5 (RN5) from Pointe Noire to the village Tchizalamou (70 km), within the Sintoukola permit's southern boundary (Figure 2-1) and then via sand tracks to the mine site (20 km).

Access to the Sintoukola Project process plant site (near the village of Tchiboula) is via the RN5 from Pointe Noire to the village of Tandou Youmbi (50 km) and then via sand tracks to the process plant site (12 km).

The RN5 is a dual lane bitumen road approximately 7 m in width with hard shoulders and 50 km from Pointe Noire to the village of Tandou Youmbi. The remaining section of the RN5 is unpaved. The road crosses the Kouilou River with a pre-stressed concrete bridge built in the 1980's. The bridge is currently under repair, and heavy materials currently use a temporary ferry to make the river crossing.

3.3 Proximity to Population Centre

Pointe Noire is a major hub for the oil industry in the ROC and is connected to Libreville in Gabon, Johannesburg in South Africa, Paris in France and Frankfurt in Germany by flights operated by Gabon Airways, South African Airways, Air France and Lufthansa, respectively. There are also several daily flights to the capital, Brazzaville, operated by internal carriers, which allow connection to other carriers flying to Europe, South Africa and Ethiopia.

The Kola deposit is in close proximity to fourteen villages and three blocks (sub-settlements within a village); the three closest villages to the mine site have populations ranging from approximately 400 to 800 people.

3.4 Climate and Length of Operating Season

The mean average temperature for the Sintoukola Project area is 25°C with variations of 2°C from the mean during the wet and dry seasons. There is a distinct wet season (October to May) and dry season (June to September). Rainfall is largely restricted to the wet season but declines slightly in December and January and this period is referred to locally as the ‘small dry season’. Annual rainfall in the project area is 1,242 mm.

3.5 Infrastructure Availability and Sources

The current infrastructure within the Sintoukola Project area is sufficient to support ongoing exploration activities and the facilities available at the exploration camp are described as follows:

- Fully self-contained exploration camp for up to 140 employees. The camp is fully catered by Sodexo with 250 meals supplied per day serving 24 hour shifts. Refuse disposal and recycling sites have been developed;
- Office, workshop and warehouse facilities have been constructed and a good communication system (satellite broad band) is available;
- A primary health care and trauma stabilisation unit is in operation with emergency evacuation capabilities;
- Water for washing is supplied from an established bore field; and
- Electricity is generated on-site in a diesel power generation plant. Water and electricity supplies are adequate for the intended exploration programme and ancillary operations.

The status of regional infrastructure is as follows:

- No surface or groundwater infrastructure is available;
- There is currently a surplus of generation capacity of 87 MW (dry season) in the country and a rehabilitated and extended transmission grid. The closest connection to this grid is at the Mango Kamba II substation, 57 km away from the process site;
- Commercial natural gas production is limited as most gas produced is a by-product of oilfields and is currently flared. Currently, natural gas is commercially available from one source at Cote-Mateve 10 km south of Pointe Noire. This natural gas is produced by Eni S.p.A. (ENI) Congo and transported from the M’boundi onshore gas field to the gas

treatment facilities located at Cote-Mateve. These facilities which feed the Centrale électrique à gaz du Congo (CEC) power plant are owned by ENI Congo. The Marine XII off shore gas field is under development and will supplement the gas coming from M'Boundi within the next years;

- A dual lane bitumen road (RN5) runs from Pointe Noire to just north of Madingo-Kayes, 36 km from the existing exploration camp. From this point, access is via a sandy track to the exploration camp;
- The port of Pointe Noire is one of the major deep water ports in West Africa, with imports and exports exceeding 12 Mt in 2008. Discussions with the port authorities have indicated that a new mineral export port is proposed to the north of the existing port adjacent to the oil refinery, it is expected that this port will be financed by other companies developing mineral projects in the ROC; and
- Though limited fixed line telephone networks operate in Pointe Noire, several mobile operators including the South African major MTN operate cellular networks in the area which are capable of data and voice transmission.

4 History (Item 6)

4.1 Prior Ownership and Ownership Changes

The Sintoukola permit is a new permit that was issued to SPSA in 2009. Although there has been prior exploration in the general area (discussed below), there has been no prior ownership of the Sintoukola permit.

4.2 Previous Exploration and Development Results

Potash was discovered in the coastal Congolese Basin during oil prospecting in the region in 1935. Additional work carried out in the late 1930's and 1950's discovered that the salt basin existing in Gabon was only a portion of a vast littoral basin which extended south through the ROC into Cabinda, Angola. The basin ranges from 5 to 40 km in width and is over 200 km in length.

In 1960 a company, Syndicat de Recherches de Potasse au Congo, was established to explore for potash in the Kouilou portion of the Congo Basin. The company was initially granted an exploration permit covering the entire coastal plain (8,450 km^2) including the area currently held by ELM. After the reinterpretation of seismic data and the completion of some 44 exploration drillholes totalling 34,648 m, the focus of exploration activities concentrated around the Holle project situated in the southern section of the province and external to the Sintoukola permit.

Approximately 31 historic drillholes occur within and adjacent to the Sintoukola permit area (at least 17 of which were within the area now known as the Sintoukola Project). Details of the historical drilling relevant to the Sintoukola Project are discussed further in Sections 9 and 10.

Since ELM's involvement in the Sintoukola Project, ELM has completed two phases of exploration (Phase 1 and Phase 2, the latter comprised of 2a and 2b). Details of the work completed are provided in the following sections.

4.3 Previous Mineral Resource and Reserve Estimates

A maiden Mineral Resource estimate was reported for the Kola deposit by CSA (SRK, 2011).

The geological model for the maiden Mineral Resource estimate was based on the interpretation of geological boundaries from observations of drill core, assays, down hole logging and seismic data available on March 25, 2011. The extents of the Kola deposit were determined by drillhole and seismic data.

The Mineral Resource estimates listed in Tables 4.3.1 and 4.3.2 are based on two mineralization domains (seams) which are described as Lower Seam (LS) and Upper Seam (US). The seams are sub-horizontal in attitude and are separated by a barren, on average 3.6 m thick halite zone, ranging from 0 to 6.5 m. The US consists primarily of sylvinite with an increasing proportion of halite towards the upper contact, while the LS is only partially converted to sylvinite and includes carnallitite and halite.

Table 4.3.1: Mineral Resource Estimate at a 15% K_2O CoG as reported April 4, 2011

Seam Name	Resource Classification	Tonnes (M)	K_2O%	KCl%	Avg. thickness (m)	Density
USS	Indicated	229	21.33	33.78	4.94	2.01
USS	Inferred	289	21.41	33.92	5.08	2.01
LSS+LSC	Indicated	133	16.51	25.81	5.60	1.85
LSS+LSC	Inferred	153	16.10	25.51	5.60	1.85
Both Seams	Indicated	362	19.48	30.83	5.18	1.95
Both Seams	Inferred	442	19.57	30.98	5.22	1.95

Notes:
1. The above table presents the Mineral Resource, above a CoG of **15% K_2O** for both the mixed sylvinite and carnallitite zone in the LS and the sylvinite zone of the US
2. Fault losses have been applied.
3. Mineral Resources are reported in accordance with The JORC Code, which is consistent with the CIM Definition Standards.
4. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.
5. Table entries are rounded to the second significant figure. Differences may occur due to rounding errors.

The US was further sub-divided, based on mineralogy and location into a lower high-grade sylvinite zone (US1) and an upper lower-grade zone (US2). The same resource parameters have been used to estimate the mineral content within the US1 but using a 20% K_2O lower CoG for reporting purposes (refer to Table 4.3.2).

Table 4.3.2: Mineral Resource Estimate at a 20% K_2O CoG for High-grade Domain Within the USS as reported April 4, 2011

Seam Name	Resource Classification	Tonnes (M)	K_2O%	KCl%	Seam thickness (m)	Density
US1	Indicated	151	25.08	39.72	3.27	2.01
US1	Inferred	186	25.24	39.97	3.27	2.01

Notes:
1. The above table presents the Mineral Resource, above a CoG of **20% K_2O** for the lower high-grade domain within the sylvinite zone of the US.
2. Fault losses have been applied.
3. Mineral Resources are reported in accordance with The JORC Code, which is consistent with the CIM guidelines.
4. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.
5. Table entries are rounded to the second significant figure. Differences may occur due to rounding errors.

By May 2012, ELM had completed a second phase of exploration work, including 2D seismic and drilling, which enabled, due to the larger number of drillholes and some targeted drillholes, a revision of the geological and mineralisation models and consequently a new Mineral Resource estimate.

The Mineral Resource estimate was reported based on a "three seam" model and domaining of the seams by potash mineral type. A lower CoG of 10% K_2O for combined sylvinite and carnallitite mineralisation and sylvinite only mineralisation as listed in Table 4.3.3 and Table 4.3.4.

The Mineral Resource estimate for the modeled mineralized zones for the Kola deposit was classified as Measured, Indicated and Inferred. This is based primarily on confidence in, and continuity of, the results from the drilling campaigns, and subsurface mapping of high density 2D seismic data. The results of the May 8, 2012 Mineral Resource estimate are presented below.

Table 4.3.3: Mineral Resource Estimate for Sylvinite and Carnallitite Mineralisation at a 10% K_2O CoG, as reported May 8. 2012

Item	*Measured*			*Indicated*			*Inferred*		
	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl
USS	238	18.56	29.38	194	17.62	27.89	163	16.83	26.64
LSS	280	12.55	19.87	247	12.71	20.12	227	12.63	19.99
FWS	-	-	-	-	-	-	123	18.34	29.03
Total	518	15.31	24.24	441	14.87	23.54	513	20.39	24.24

Table 4.3.4: Mineral Resource Estimate for Sylvinite Mineralisation only at a 10% K_2O CoG, as reported May 8. 2012 (Includes Resource in Table 4.3.3)

Item	*Measured*			*Indicated*			*Inferred*		
	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl
USS	161	21.47	33.99	115	21.29	33.70	78	21.09	33.39
LSS	69	18.45	29.21	68	18.27	28.92	60	18.33	29.02
FWS	-	-	-	-	-	-	123	18.34	29.03
Total	230	20.56	32.55	183	20.17	31.93	261	19.16	30.33

Notes:

1. Table 4.3.3 above presents the Mineral Resources, above a CoG of 10% K_2O for both the mixed sylvinite and carnallitite zone in the US and LS as well as the sylvinite zone of the FWS).
2. Table 4.3.4 above presents the Mineral Resources, above a CoG of 10% K_2O for the sylvinite mineralisation only in the US, LS and the FWS.
3. Zones of geological uncertainty have been excluded (resulting in approximately 7% volume losses).
4. Table entries are rounded to the second significant figure. Differences may occur due to rounding errors.
5. Mineral Resources are reported in accordance with the JORC Code, which is consistent with CIM guidelines.
6. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.

There have been no other historical estimates of Mineral Resources or Mineral Reserves for the Kola deposit.

4.4 Historic Production

There has been no production of potash or related products from the Sintoukola permit area.

Underground mining of sylvinite seams has occurred at the historic Holle Potash Mine, some 70 km south of the Kola deposit, discussed further in Section 23 (Depege, V., 1967).

5 Geological Setting and Mineralization (Item 7)

Detailed discussions on the geology and resources are presented in in Volume II (CSA, 2012).

5.1 Regional Geology

The Sintoukola Project area covers the Tertiary and Quaternary coastal plain area of the ROC, which formed along the base of the escarpment of the Mayombe Ridge. The basin is bounded by deformed and metamorphosed basement rocks of the Congo Craton (Figure 5-1) to the northeast and extends to the west below the ocean floor and above oceanic crust of the South Atlantic Ocean (Figure 5-1 and Figure 5-2).

The development of the evaporite basin began during the Middle/Upper Jurassic and peaked during the Lower Cretaceous (Figure 5-3). The basin strata of the western continental margin of Africa have their origin in the breakup of Gondwana and the subsequent separation of the South American and African continents. Evaporite and salt-bearing formations occur onshore in the Douala basin (Cameroon), the Gabon basin, the Congo basin (ROC) and the Cuanza basin (Angola) with time equivalent deposits also present along the eastern margin of South America (de Ruiter, 1979).

A regional profile based on offshore seismic data shows that the clastic and saline units extend subsurface below the present-day eastern margin of the equatorial South Atlantic Ocean (Figure 5-2).

The evaporites separate an underlying continental sequence from an overlying marine section (Figure 5-3). The pre-salt continental clastics were deposited prior to, and during, the rifting phase of the Afro-American continent. The post-salt shallow marine and deltaic facies units accumulated during the subsidence of the continental margin of Africa.

Pre-rift continental clastic sediments of Permian to Middle Jurassic, are found in outcrop in the northeastern part of the Gabon sedimentary basin. These deposits can be considered to represent the remains of a pre-rift sequence laid down on a stable shield then covering parts of Africa and South America (Chouteau et al, 1997). Within the Congolese basin, this period is marked by non-deposition highlighting its erosional position. The oldest known sediments are of upper Jurassic age.

Rift sediments loosely called the "Cocobeach Group" and lateral equivalents of the Djeno/Lucina Sandstones and the Pointe Noire Marls are continental clastics of varying depositional environment (Figure 5-3). Fault-controlled thickness differences and syn-sedimentary tilting is characteristic. Their fresh-water environment is indicated by the ostracod faunas. The "Lower Cocobeach" (Upper Jurassic-Neocomian) consists largely of relatively deep-water, lacustrine shales with occasionally turbiditic sands. It is, in places, underlain by a sandy unit that overlies basement rock.

The post-rift sedimentary sequence was concluded by a phase of faulting, tilting and subsequent peneplanation. A new cycle of deposition commenced during the Aptian. Its development was governed by two dominating factors: the gradual appearance of salt water by the newly established connection with the oceans and subsidence along the continental margin. A salt formation (Loeme Evaporite Formation) together with the underlying transgressive Chela Formation makes up the basal part of the post-rift sequence. The salt is overlain by an increasingly marine sequence of Albian-Cenomanian carbonates (Senji Carbonate Formation) and Cenomanian to recent clastics. A major regional break is recorded at the base of the Miocene (Figure 5-3).

5.2 Local Geology

Results from recent and historic exploration for petroleum in the onshore and offshore parts of the Congo Basin have provided a reasonably well understood geological framework. In general, the thicknesses of the sedimentary strata decrease from west to east towards the Mayombe basement. The following rock unit descriptions are based on drill logs from exploration wells drilled within the project area. A summary geological log for the project area is shown in Figure 5-4 and described below.

From the top (younger strata) to the bottom (older strata), the following lithological units are differentiated:

- The project area is overlain by a Pliocene to Pleistocene layer of up to 50 m of unconsolidated and alternating clay and sand beds *(sable de surface)*. The clay and sand units are variably coated by iron oxides. If the phosphate horizons are missing in the underlying weathered sandstones, it is not always possible to distinguish between the clays and sands and the underlying ferruginous sandstones;
- The clays and sand overlie deeply weathered and partly unconsolidated medium to coarse-grained ferruginous sandstone. The ferruginous Sandstone Formation (Sables Gressiers Ferrugineux) consists of layers of relatively coarse-grained sandstone, locally with iron concretions and separated by layers of plastic red clay strata;
- Below the sandstones occurs an interbedded unit of mudstone and dolomitic siltstone referred to as the Grey-Blue Claystone Formation (Serie Argilo-Greso-Dolomitique). It consists of grey-blue to grey-green claystone to siltstone and fine-grained sandstone with dolomitic cement and some layers/lenses of limestone. The contact to the overlying iron-bearing sandstone is an unconformity;
- The dolomitic sandy limestone rests on a narrow (5 to 16 m) thick stratified gypsum/anhydrite and black organic rich marl/shale unit (Anhydrite Formation, “Serie Anhydritique”) which caps the top of a thick salt sequence. Gypsum, the hydrated original precipitate, can lose up to 40% of its molecular water content during diagenesis and compaction when dewatered to become anhydrite. Petrographic data shows that the gypsum possibly underwent dewatering and re-hydration since its deposition. This contraction and expansion led, in places, to partial destruction and dissolution of the original sediments and layering and to plastic movement of the constituting sediments. As a result, the anhydrite / gypsum unit is locally completely removed and the dolomite unit can rest on a narrow clay layer upon rock salt. The Anhydrite Formation consists of an upper portion that is 4 to 8 m thick and is made up of massive white gypsum. The underlying, alternating anhydrite and marl and carbonaceous clay unit is up to 8 m thick. The Anhydrite Formation is variable in appearance ranging from nodular, brecciated, or laminated to massive; and
- The Salt Sequence (Serie Salifere) is described to contain, depending on local development and preservation, up to 11 depositional cycles. Within the project area, seven cycles are identified and differentiated (Figure 5-5). Each of these cycles consist of several alternations of the sequence: shale, rock salt, carnallitite and sometimes with layers rich in bischofite ($MgCl_2 \cdot 6\ H_2O$) as shown in Figure 5-6.

The structure of each cycle is very clearly recognizable on the geophysical down hole logs (Figure 5-5). The halite beds can be identified from the logs by their low gamma-ray response, low neutron porosity, and the 2.10 g/cm^3 density. The carnallitite beds are characterized by relatively high radioactivity, very high neutron porosity, reflecting the high percentage of molecular water and a lower density of approximately 1.60 g/cm^3.

Detailed geological work has shown that within the uppermost salt cycle there are several minor lithologies and textural changes that serve as marker horizons. The most prominent of these is an interval of 4 to 5 m consisting of close-spaced, very thin (<2 mm) organic shale/clay beds (up to 20 per m). Locally, there may be a shale/clay bed up to 15 cm thick. This zone occurs between two sets of thin potash beds (Triplets and Dublets shown in Figure 5-5) in the halite between the US and HWS. A second marker horizon suitable for lateral correlation is a halite facies transition mappable approximately 10 to 12 m below the base of the LS. The gradual transition occurs from light to creamy coloured, rhythmically thin bedded, and calcite/anhydrite-capped halite to dark massive and coarse crystalline halite. These markers are preserved in the majority of drillholes and were used for correlation.

The evaporite sequence of the Congolese coastal onshore basin is known to measure between 300 to 900 m with an average of 600 m. It is thinnest near the crystalline basement in the east and thickens to the west (Figure 5-7). Within the project area the evaporite thickness ranges from 340 to 400 m. Contour lines of the depth to the top of the evaporite sequence, based on regional onshore and offshore drillholes, indicate a relatively stable and gradual deepening of the formation towards the offshore continental margin. Offshore seismic shows a marked change of style of deformation with pronounced diapirs occurring above oceanic crust.

The evaporite sequence unconformably overlies reduced facies sandstones of the Cocobeach Formation (Serie du Cocobeach). This contact marks the transition from a rift environment to a marine hypersaline environment.

The Cocobeach Formation consists of a series of grey-green sandstone with interbedded pyrite bearing green claystone probably deposited under continental conditions in fluvial and lacustrine environments. This unit tends to form the underlying stratigraphy across the project area. The unit is reported to measure more than 2,000 m thick locally. It rests unconformably on top of the crystalline basement.

5.3 Deformation

The project scale assessment of deformation relies on the interpretation of 2D seismic data. The accuracy with which this can be conducted depends on the density of seismic data available for interpretation.

Following acquisition of additional seismic data in 2011, which reduced the line and data spacing from 1 km to a range of 130 to 150 m, the structural model for the project area was revised.

Based on higher density of data it was concluded that the seismic “disturbance areas” identified in 2011 are not indications of regional continuous structural trends, rather they represent locally developed depressions due to salt removal by dissolution akin to salt collapse structures in the Devonian potash basin of Saskatchewan. A disturbance area is an area that has the potential to affect the laterally continuous character or preserved thickness of the potash-bearing beds and may represent a zone that is potentially unsuitable for underground mining.

The disturbance areas were mapped in section view first and then interpreted in plan-view. Depth-migrated seismic sections were individually interpreted and assessed for areas of reflector disturbances (Figure 5-8). The following criteria were applied in identifying the disturbance areas:

- Lowered position of the base anhydrite reflector;
- Poor lateral seismic reflector continuity;
- Cone or pipe-like shape of area of reflector loss;
- Opposing and abruptly inclined reflectors on the edge of the disturbance zone; and
- Evidence of refraction multiples below the base of the gypsum (anhydrite) / clay horizon.

If the above features were not recognizable on adjacent sections a radius of the diameter of the observed features was drawn as a circular polygon (in plan-view) and this was applied to the seam model using a cookie cutter method. The interpreted spatial location of the disturbance zones is based largely on the high density seismic data which is restricted to the central part of the Kola deposit (see Figure 5-9). In the areas that lack high-density seismic data the spatial location of areas of uncertainty could not be confidently interpreted.

The current interpretation is that over geological time the position of the anhydrite-salt contact has shifted due to infiltrating ground water and salt removal by dissolution. This “dissolution surface” developed to variable depth across the project area.

Figure 5-10 illustrates the geometry of the potash horizons below the base of anhydrite and the main types of structural anomaly that may affect the mineralized seams. The long-section shows that the geometry of the beds below and above the anhydrite-salt contact may differ. While each domain above and below this surface shows internal parallel layer patterns this is not the case across the contact. This interpretation gives rise to the consideration of two fundamental conceptual processes: a) salt undergoing folding and erosion prior to the deposition of the Anhydrite Formation; b) salt becoming unstable to the sediment loading and begins to rise or form salt cushions which are counteracted by dissolution at the anhydrite salt contact.

Conceptual process b) is the favoured structural process considered for the sub-surface geometry of the salt beds at the Kola deposit. This contrasts with the interpretation of similar geometries observed at the historic St. Paul potash mine near Holle where the interpretation is based on the assumption that the deposition of the Anhydrite Formation was preceded by an event of folding and peneplanation (de Ruiter,1979, Feuga et al., 2005). This concept does not explain the variation in relative depth below sea level of the base of anhydrite and the consistent thickness of the cover sequence units. Secondly, it can be expected that a folding event would have resulted in a more symmetrical fold pattern in the salt below the anhydrite and laterally extensive residual breccias or incised valleys. It is therefore thought that a process of contemporaneous salt migration (upwelling) and dissolution at the anhydrite–salt interface explains the current and historical geological situation more adequately.

There are no brittle structures observed in drill core that can be linked with an offset of the anhydrite-salt contact. The revised structural interpretation represents a major change to the previous structural geological model. It is possible that extensional faulting related to the accelerated opening of the Atlantic Ocean in the Late Cretaceous triggered block faulting and block rotation in the basement rocks below the salt basin and initiated salt migration and an initial phase of pillowing of very low amplitude. This type of deformation is seen on seismic lines outside the project area and would explain the flexure/plateau-type geometry of the anhydrite/salt reflector seen on seismic data.

In addition, textural down hole imagery generated from the wireline tool provided structural data of the evaporite beds. The apparent dip angles measured are broadly in line with the orientation of reflectors indicated on 2D seismic sections. The data confirm that the tilt of the salt varies in place from that of the anhydrite and cover sequence. In some places the salt layering may be flat and the cover sequence sub-horizontal and vice versa.

5.4 Mineralization

The potash mineralisation encountered within the permit area occurs as discrete stratiform layers within an overall much thicker sequence of other salts. The potash horizons are described in this section in terms of their mineralogical composition and stratigraphic position (Figures 5-10 and 5-11).

5.5 Potash Geology

The salt formations of the coastal Congo Basin are extensive and at a regional scale are largely undisturbed, although locally the original horizontal layering may be disturbed. The salt formation occurs both on land and under part of the continental shelf. Data from Belmonte et al. (1965), Lambert (1967) and project data indicate that the chemical composition of the salt is exceptional by the virtual absence of sulphates and carbonates and the high proportion (15%) of carnallite. Moreover, in several wells the extremely soluble bischofite which lies beneath the FWS, was found in thicknesses of several tens of metres.

De Ruiter (1979) refers to three normal salt cycles that can be easily recognized within the evaporite sequence and can be correlated between wide-spaced wells of the Congo Basin. These cycles tend to show a complete mineralogical sequence and great thicknesses indicating prolonged and balanced depositional conditions. The lower and upper parts of the evaporite section show a different cyclical development, with incomplete and thin cycles indicating probably less stable depositional conditions. A complete salt cycle can measure up to 100 m but is commonly less than 50 m thick.

Lambert (1967) noticed that the salt of the Congo Basin is very rich in carnallite; the in situ potash and magnesium quantities are estimated to be several billions of tons. However, for reasons of practicality (de Ruiter, 1979) and possibly economics, the only salt that has been mined is the sylvinite contained in sylvinite layers. This sylvinite is interpreted to be a secondary mineral derived from carnallitite and laterally a sylvinite bed may be of unknown extent.

Drilling around the Holle mine area confirmed these sylvite layers to be of semi-regional extent. Its composition and micro-facies are variable. In some places, it is made up of imbricated sylvite and halite crystals, elsewhere it consists of sylvite laminated with halite. In the area of Holle four sylvinite layers were found in the second highest salt cycle (IX). Only rarely do all four layers occur in one well. Two layers, #3 and #7/8, of cycle IX were selected for exploitation.

At Holle, a major factor in the geometry and distribution of sylvinite at the mine was the presence of folds. The anticlines played a part in the localization of the areas where the carnallitite has been converted to sylvinite. The conversion process being controlled by the possibility of brine circulation, it was interpreted in the Holle mine that the sylvinite zones are located on the flanks of the salt-bearing anticlines of which the summits were truncated by erosion preceding the transgression and deposition of the anhydrite (Feuga et al., 2005).

A comparison of the historic data from the Holle mine with mineralization geometries interpreted from the Kola deposit area suggests that despite significant similarities there are also marked differences. Details of the shape of the mineralization are given below.

5.6 Description of the Potash Seams

The potash mineralization of the Kola deposit occurs within the uppermost cycle VII of the Loeme Evaporite Formation. The sylvinite seams within this cycle occur as thick (3 to 8 m) laterally extensive beds of sylvinite mineralization separated vertically by halite intervals which commence at an average depth of 264 m below surface (Figure 5-4).

The initial target for exploration was the US and LS and these comprise the bulk of the current Mineral Resource. More recently two additional seams (the HWS and FWS) have also been intersected in the project area for which Mineral Resources have been estimated. The HWS occurs above the US and the FWS below the LS (Figure 5-10 and 5-11).

Up to five narrow (0.3 m to 0.7 m thick) potash seams (the Triplets and Doublets) occur in the halite interval between the US and the HWS, and one is known to occur in the halite below the LS (Figures 5-10 and 5-11). However these are not included in the mine planning or economic evaluation.

The HWS, US, LS and FWS were defined on the basis of stratigraphic position and mineralogical composition. No CoG values have been applied in the determination of the boundaries. Further parameters used to determine the upper and lower boundaries of the seams are detailed below (described in stratigraphic order).

5.6.1 Hangingwall Seam (HWS)

The HWS is a significant potash seam which occurs approximately 60 m above the roof of the US and is intersected between 5 and 25 m below the base of the Anhydrite Formation. The HWS was first intersected during the Phase 1 exploration programme. Further evaluation of the HWS was not possible until recent drilling (during the Phase 2b programme) provided several additional intercepts of this particular potash horizon which allowed it to be modelled and included into the current Mineral Resource estimate.

To date, all HWS intercepts at the Kola deposit consist of sylvinite mineralisation and range from 2.5 to 3.8 m in length. The HWS seam is particularly enriched in potash and exceeds 30% K_2O in all drillhole intercepts. The seam has been intersected in adjacent drillholes in the western part of the Kola deposit where the salt sequence appears to be gradually dipping to the west leading to the preservation of a more complete salt stratigraphy.

5.6.2 Upper Seam (US)

Based on the drillhole intercepts and contact relationships between the Anhydrite Formation and the salt it can be assumed that in excess of 88 m of halite and other salt beds were deposited, in the project area, above the carnallitic protolith of the US. The US rests, and is separated from the LS by an interval of halite, averaging 3.6 m thick. The immediate upper portion of the US is a transitional zone marked by interbedded potash and halite layers and occurs over an interval of 1 to 2 m (locally this can occur over a shorter interval). For geological modelling purposes the upper contact of the US is taken as the first recognizable potash horizon which is identified by the change from coarse, light- to dark-grey coloured cubic euhedral halite to light-red coloured, fine crystalline sylvite with thin bands of halite.

As discussed in Section 4.2, the US had been subdivided into US1and US2, based on mineralogy and relative mineral abundance reflected in the grade of potassium chloride (KCl) (CSA, 2011). This subdivision coincided empirically and graphically with the shoulder of the US gamma response and the upper boundary of an interval which showed a lack of halite horizons. Furthermore, geochemical testing shows that the distinction between the US1 and the US2 is simply a function of a lower average amount of sylvite versus halite in the sampling interval and is not a geological surface.

Recent drilling has shown that contrary to the previous interpretation the US is not always converted to sylvinite and has remained carnallitic in places along the seam. Of the 29 drillholes that intersected the US (and included in the Mineral Resource estimate), four intersected carnallitite-only mineralisation and four drillholes intersected an upper sylvinite portion and a lower carnallitite portion in the US. Based on additional data and revised interpretation the US has now been subdivided along mineralogical boundaries into zones consisting of potash in the form of sylvinite (USS) and potash in the form of carnallitite (USC), see Figure 5-11.

The sylvinite mineralisation is almost entirely free of magnesium-bearing minerals (e.g. carnallite) and predominantly consists of sylvite and halite. The amount of insoluble minerals, carbonates and anhydrite is typically below 0.5%. Sylvinite occurs as 1 to 5 mm wide euhedral interlocking crystal grains that form layers or as disseminations intergrown with halite. Visually the change in potassium chloride grade is, in places, associated with a change in colour from light to darker red. The basal contact is indicated by a very sharp change to a grey coloured and banded halite.

5.6.3 Lower Seam (LS)

The LS occurs below the US and immediately below the interburden halite. Potash mineralisation within the LS commences abruptly as it terminated in the US and is associated with a marked colour change. The colour of the LS gradually becomes dark red and coincides with the appearance of commonly granular textured carnallite crystals in a matrix of halite. Towards the base of the LS the number of interlayered halite bands increases together with their thickness.

The LS is less frequently, in its entirety, converted to sylvinite and consists more commonly as carnallitite. The transition from sylvinite occurs abruptly but may vary in depth from drillhole to drillhole. The sylvinite interval is always at the top of a seam. The base of the interburden halite marks the upper part of the LS.

The conversion of carnallitite to sylvinite in the LS is laterally more complex. In some drillholes it has remained entirely composed of carnallitite and halite, in ten out of 31 drillholes it is sylvinitic and in seven holes it is sylvinitic at the top and carnallitic at the bottom. Based on drillhole data, inside any single seam the sylvinite always occurs above the carnallitite. Petrographic and X-Ray diffraction (XRD) work suggest that the conversion takes place across a very sharp boundary and results in potash monomineralic assemblages.

The LS was previously subdivided into an upper Lower Seam (LS_1) and a lower Lower Seam (LS_2) based on grade and relative concentration of KCl and Mg. This subdivision is not applied any longer.

Rather, the greater drillhole density and geological data has facilitated, as in the case for the US, a subdivision in line with potash mineral composition. Where the LS is composed of sylvinite the seam is mapped as LSS, where it consists of carnallitite, it has been mapped as LSC, (see Figure 5-11).

5.6.4 Footwall Seam (FWS)

The FWS occurs at the top of cycle VI (see Figure 5-5) and is separated by approximately 45 m of halite from the base of the LS. The FWS mineralisation is bound at the top by an interval of halite-bearing dolomite, carbonaceous clay and gypsum or anhydrite approximately 0.6 to 1.1 m thick. This interval marks the base of cycle VII and represents genetically and geologically a period of basin flooding and a sudden sea water inflow and the start of a new cycle of evaporation and condensation. The base of the FWS is marked by a gradual transition to carnallitic bischofite.

The stratigraphic position of the FWS has been intersected in several drillholes previously but was found to commonly consist of carnallitite and bischofite. It was only during ELM's second phase of exploration drilling that sylvinite was encountered in this stratigraphic position (Figures 5-10 and 5-11). Where sylvinite was intersected in the FWS, the stratigraphically higher potash seams are generally absent. When converted to sylvinite, the composition of the FWS is almost exclusively sylvite and halite with less than 2% of other salts and insoluble minerals. The average thickness of the FWS is 6.7 m.

5.7 Potash Grades

The grade of potash mineralisation is a function of mineralogy and is generally higher in sylvite beds then it is in carnallitite beds. The richest mineralisation is where the entire US and LS have been converted from carnallitite to sylvite.

Where the US or LS are sylvinitic the grade of potash (KCl) is symmetrically distributed around the central barren and separating halite zone. In these cases the grade peaks adjacent to the halite interval and gradually tails off away from the halite towards the respective seam boundaries. The separating halite interval has sharp boundaries with the overlying and underlying potash seams. This symmetrical distribution pattern of potash about the central halite layer is developed to a variable degree in all drillholes that intersected mineralization.

Within the US the grade decreases towards the top of the seam and in the LS it decreases towards the base of the seam. Grade variation occurs in a non-linear distribution and is possibly a function of diagenetic enrichment. The changes in grade on a centimetre to metre-scale within each seam contrasts with the relative stability of potash content per seam as defined by geological boundaries and irrespective of the mineral facies.

The range of average K_2O grades for mineral domains within a seam is as follows:

- HWS ranges between 30.58% K_2O to 37.65% K_2O with a mean of 34.00% K_2O;
- USS ranges between 15.04% K_2O to 26.24% K_2O with a mean of 22.14% K_2O (the low minimum reflects a partially preserved USS);
- USC ranges between 11.03% K_2O to 16.46% K_2O with a mean of 13.42% K_2O;
- LSS ranges between 11.47% K_2O to 24.08% K_2O with a mean of 19.01% K_2O;
- LSC ranges between 8.11% K_2O to 13.42% K_2O with a mean of 10.79% K_2O; and
- FWS ranges between 13.09% K_2O to 22.27% K_2O with a mean of 17.72% K_2O.

Modelled grade and thickness distribution within the Kola deposit area are presented in Section 14 of this report. The average grade compared by mineralisation type in the US and LS fall within a narrower grade range across the entire deposit area. In the sylvinite mineralisation they commonly are above 20% K_2O, and in the carnallitite mineralisation they are commonly above 11% K_2O.

The sylvinite mineralisation in the HWS is of particularly high grade. All intercepts of this mineralisation are above 30% K_2O and occur over a thickness that is considered potentially minable. These grades are not exceptional for the Congolese potash basin. The historic St Paul potash mine in the Holle area initially exploited a similar high grade seam (Feuga et al., 2005).

Sylvinite mineralisation in the FWS is correlated with the absence and removal of the US and LS. At present, it is known to occur in seven locations where it is intersected by drillholes. Given the small number of drillhole intercepts the lateral extent of the sylvinite mineralisation in the FWS is more difficult to quantify than the US or LS.

5.8 Thickness of Potash Seams

The thickness of the US and LS interesections are shown in Table 8.8.1. Thickness contours are plotted on top of coloured grade contours presented in Section 14.

The variation in thickness observed in drillholes is a function of partial conversion of carnallitite to sylvinite. The thickness of an entirely converted potash seam tends to fall between 4 m and 6 m with a mean of 4.4 m. The carnallititic seams range between 6 m and 14 m with a mean of 9 m. These values represent the limits of the geological boundaries in the system.

Based on core observations, CSA's experience from other potash systems, published literature and mineralogical processes, the variations in thickness and grade based on geological boundaries are considered natural to such systems. The observation made with regard to grade and thickness within the project area suggest that it is likely that the system extends in the current form and shape beyond the currently defined boundaries.

5.9 Verification of Geological Interpretation

Geological drillhole logs and mineralogical composition were compared with the geophysical logs of natural gamma, density and porosity. A good correlation is observed between bulk mineralogical composition and down hole geophysics. When reviewing the natural gamma against the geological drillhole trace in log normal display the boundaries appear sharper and therefore can be better defined.

Twelve samples from both seams were selected for semi-quantitative mineralogical analysis by XRD in conjunction with energy dispersive spectroscopy (EDS) using a scanning electron microscope. The results of mineralogical work are summarized in (Table 5.9.1) and support the assay data.

In addition, 39 samples were collected and underwent petrographic analysis and point specific XRD analysis. Table 5.9.2 provides a summary of the principal mineral composition (composition >90%) of 39 petrographic samples analysed. Petrographic work shows that the potash seams are essentially non-porous. In the case of carnallite, the mineral forms euhedral and interlocking textures with halite whereas sylvinite is seen to have irregular boundaries and to surround halite crystals.

The review of the semi-quantitative mineralogical data further supports the conclusions drawn from core observations, down hole geophysics and drill assays.

Table 5.9.1: Summary of the Results from Semi-Quantitative XRD and SEM/EDS

	Sample-ID	From	To	Seam	Halite	Sylvite	Anhy-drite	Calcium Hydrogen Phosphate Hydrate	Carnallite	Quartz	Calcite	K_2O% XRD	K% XRD	Mg% XRD	Na% XRD	K % EDS	Mg % EDS
EK_06	KO_DH0307	276.5	278	1	65.0	34.0	0.0	0.0	0.0	0.0	0.0	27.45	22.8	0.02	21.5	16	0
EK_06	KO_DH0314	287.3	288.8	2	51.0	0.3	0.0	0.0	49.0	0.0	0.0	14.8	12.3	7.82	5.6	13	7.4
EK_09	KO_DH0424	248.2	249.7	1	65.0	35.0	0.2	0.0	0.0	0.0	0.0	25.64	21.3	0.01	21.2	18.4	0
EK_09	KO_DH0433	260	261.5	2	40.0	0.0	0.0	0.0	60.0	0.0	0.0	13.12	10.9	6.96	8.4	10.7	7.7
EK_07	KO_DH0460	249.85	251.1	2	74.0	18.0	0.0	0.0	8.0	0.0	0.0	16.97	14.1	2.65	18.9	18.1	3
EK_10	KO_DH0557	276.25	276.99	1	63.0	37.0	0.1	0.0	0.0	0.0	0.0	26.24	21.8	0.02	21.9	21.1	0
EK_10	KO_DH0559	278	279.25	1	67.0	33.0	0.1	0.0	0.0	0.0	0.0	27.93	23.2	0.02	20.9	19.6	0
EK_10	KO_DH0566	283.64	284.96	2	74.0	25.0	0.3	0.0	0.6	0.0	0.0	23.59	19.6	0.15	22.9	17.9	1.5
EK_11	KO_DH0598	295.49	296.8	1	72.0	27.0	0.4	0.0	0.0	0.2	0.1	20.7	17.2	0.03	23.8	14.8	0
EK_11	KO_DH0603	300.66	302.07	2	79.0	20.0	0.3	0.0	0.0	0.1	0.0	18.54	15.4	0.05	24.9	19	0
EK_12	KO_DH0657	257.55	258.54	2	73.0	27.0	0.0	0.0	0.0	0.0	0.0	15.53	12.9	0.04	30.3	11.4	0
EK_13	KO_DH0725	258.74	259.63	2	59.0	40.0	0.2	0.0	0.8	0.0	0.0	33.95	28.2	0.12	16.4	26.2	0

Table 5.9.2: Summary of Petrographic Results

Sample	Depth (m}	Main Minerals (> 10%)	Minor Components (< 10%)
EK 01a	346.6	Bi	Can Hal, Syl
EK 01b	362.55	Bi	Car, Hal, Syl
EK 05a	214	Gyp	Dol
EK 05b	277.5	Syl, Hal	-
EK 06a	264.05	Hal	-
EK 06b	280.2	Syl, Hal	-
EK 07a	242.3	Hal, Syl	An
EK 08a	259.3	Hal, Syl	An, Qal, Hem
EK 09a	248.4	Hal, Syl	-
EK 10a	229	Gyp	Dol, An
EK 10b	427.8	Hal. Car	An, Syl, Hem
EK 12a	252.65	Hal	An
EK 13a	258.8	Hal, Syl	Hem
EK 13b	269.3	Hal	An
EK 15a	265.6	Hal, Svl	-
EK 16a	540.7	Car	Hal, An, Svl
EK 16b	586.3	Q, Dol	Chi, Mus. Pyr, Al, Mic, Hal
EK 16c	584.6	Q, Dol	Oil, Mus, 8a, Anor. Hal
EK 16d	585.8	Q. Mus, Chl, Anor	Dol, Hal, Sph, 8a
EK 20a	208.2	Gyp	Hal, An
EK 20b	249.9	Hal, Car	Syl
EK 20c	310.75	Car, Hal	Syl
EK 23a	259	An	Dol, Hal, Q
EK 23b	360.05	Bi, Hal	Car, Magh, Mus, Dol, Q, An
EK 24a	341.75	Bi	Car, Hal, Syl
EK 25a	264.85	Bi, Hal	Car
EK 25b	270.85	Bi, Hal	Car
EK 27a	318.15	Hal, Svl	-
EK 28a	261.75	Hal, Svl	-
EK 28b	323.85	Hal, Q	Anor, Chi, An
EK 28c	334	Bi	Car, Hal, Syl
EK 29a	292.25	Hal	Syl, An
EK 29b	360	Dol, Hal, An	Q
EK 31a	214.25	An	Dol, Hal
EK 31b	343.5	Bi	Car, Hal, Syl
EK 32a	301.1	Car. Hal	Syl. Gyp
EK 35a	275.15	Bi	Hal, Car, Syl
EK 36a	213.55	Gyp	-
EK 36b	284.35	Hal, Svl	-

Note: Main Minerals and Minor Components are defined as follows:

Al	Albite	$NaAlSi_3O_8$,
An	Anhydrite	$CaSO_4$
Anor	Anorthite	$CaAl_2Si_2O_8$
Ba	Barite	$BaSO_4$
Bi	Bischofite	$MgCl*6H_20$
Car	Carnallite	$KMgCl_3*6H_20$
Chi	Chlorite	$(Mg, Al)_6(Si,Al)_4O_{10}(OH)_8$
Dol	Dolomite	$CaMg(CO_3)2$
Gyp	Gypsum	CaS04 *2H20
Hal	Halite	NaCl
Hem	Hematite	Fe_20_3
Magh	Maghemite	Fe_20_3
Mic	Microcline	$KAlSi_3O_8$
Mus	Muscovite	$KAl_2(SiAl)O_{10}(OH,F)_2$
Pyr	Pyrite	FeS_2
Q	Quartz	SiO_2
Sph	Sphalerite	ZnS
Syl	Sylvite	KCl

5.10 Hydrogeology

A detailed hydrogeological study was carried out as part of the PFS for the Kola deposit which included:

- Detailed hydrogeological and groundwater quality characterization/testing in multi-level piezometer sets installed at two sites (Sites A and B);
- Hydrogeological characterization/testing in an area of geological disturbance;
- Installation of and hydrogeological testing in shallow groundwater monitoring wells distributed across the potential mining area; and
- Chemical analysis of groundwater from deep piezometers installed at Sites A and B and the shallow groundwater monitoring wells shown in Figure 5-12.

The data from hydrogeological characterization of the groundwater system were used to develop a conceptual model of the hydrogeology of the Kola deposit, assess potential hydrogeological risks related to mining and develop recommendations for future work at the FS level. The results of completed hydrogeological investigation are presented in Volume V (SRK, 2012a).

The hydrogeology of the Kola deposit consists of a single overburden (above-salt) groundwater system that is subdivided into (from top to bottom) surficial sands, laterite/ferruginous sandstone, dolomitic siltstone, and dolomitic limestone units, which follow the geologic units discussed in Section 5.2 of this report. The surficial sands and poorly consolidated laterite and sandstone are approximately 190 m in thickness; the siltstones and commonly fractured limestones together are approximately 45 m thick. The overburden stratigraphy is summarized in Table 5.10.1.

Detailed hydrogeological testing and groundwater quality sampling was carried out in ten shallow piezometers, and in two multi piezometer sets. Each set has one pumping well within dolomitic limestone and six piezometers installed in each hydrogeological unit. Five-day pumping tests were completed at each site additionally to short-term airlift testing from each individual piezometer. Piezometer installation and testing locations are shown in Figure 5-12. Hydrogeological test results were analyzed using a combination of analytical methods and site specific pump-test numerical groundwater models. Results of the hydrogeological testing are summarized in Table 5.10.1, which indicate that the overburden groundwater system is relatively permeable and that hydrogeological units within this groundwater system are hydraulically connected vertically. The total combined transmissivity of all units within the overburden groundwater system ranges from 112 to 243 square metres per day (m^2/d) in the two locations where it was tested. Based on results from the 10 shallow piezometers, the transmissivity of the laterite/ferruginous sandstone unit ranges from 21 to 60 m^2/day. Additionally, there is a strong hydrogeochemical gradient within the overburden groundwater system with Total Dissolved Solids (TDS) increasing with depth from 0.04 grams per litre (gpl) within the upper part of the laterite to up to 180 gpl at the bottom of the dolomitic limestone.

Infiltration from precipitation recharges the groundwater system. Large seasonal variations in water levels are not seen due to the relatively permeable nature of the overburden groundwater system. Lateral discharge is mainly to streams and other surface water bodies in the Kola deposit. The measured depth to the water table varies between 4 and 26 m below ground surface. Vertical gradients measured in the Kola deposit indicate that in some locations vertical flow occurs and that in some areas shallow water recharges to the dolomitic siltstone and dolomitic limestone occur.

The overburden groundwater system lies on top of the anhydrite sequence which is not present everywhere. Based on measured hydraulic parameters the anhydrite sequence could be considered an aquitard, where it is present. However, because drilling has found it to be missing or anomalously thin in approximately 10% of holes, it is considered to be unreliable as an impermeable seal between the mine and the overlying aquifer.

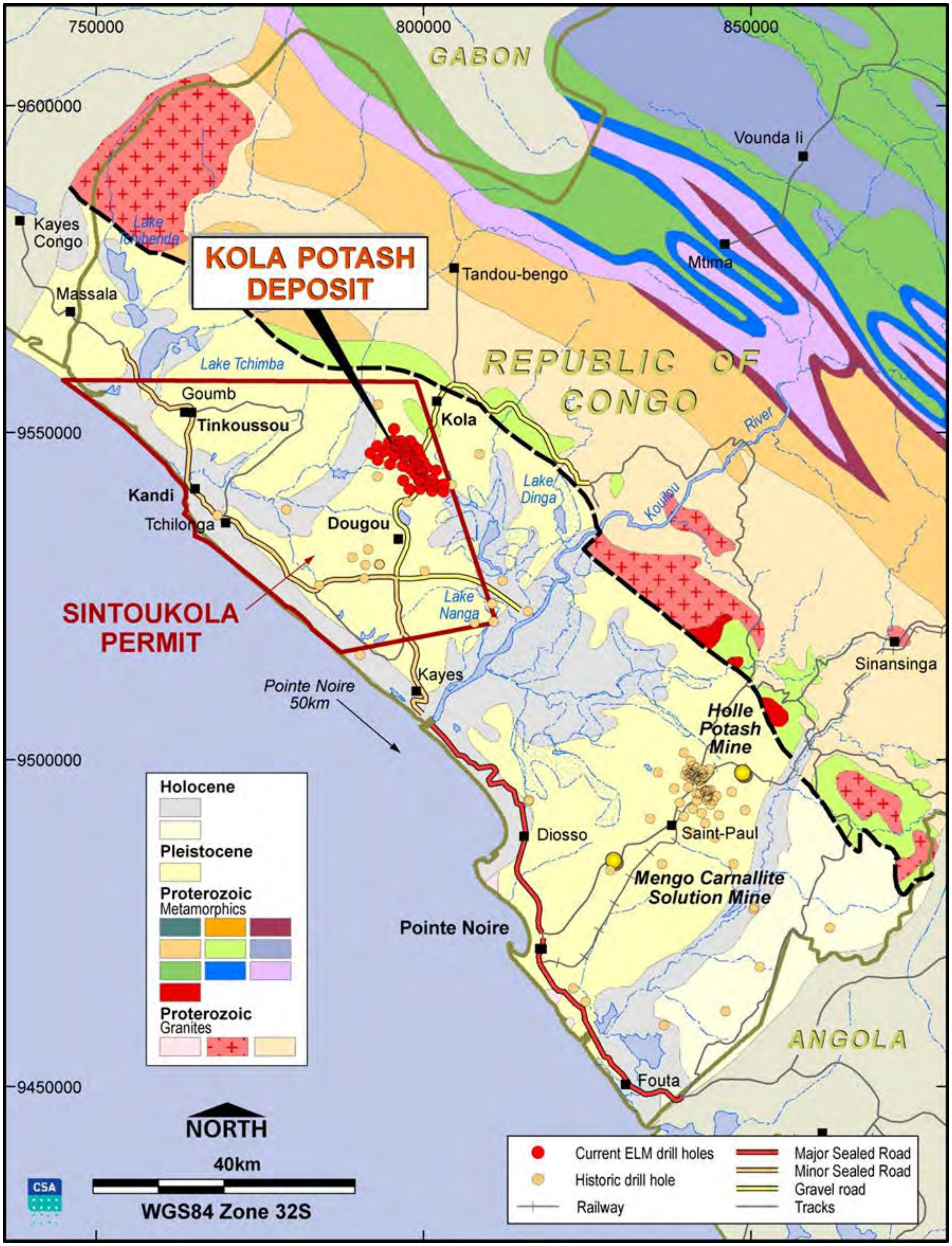
Halite, between the anhydrite sequence and the sylvinite mining horizon, varies in thickness from 10 m to 100 m. Based on hydrogeologic test results, the halite sequence, which is found beneath the anhydrite sequence may be considered to be an aquitard.

Disturbance areas previously described within this report are present in the regional geologic system (extent shown in Figure 5-9). In places, the disturbance areas breach the anhydrite sequence and bring clastic sediments down into contact with the upper evaporite rocks. Where tested, the transmissivity of the disturbance areas is higher than in the two undisturbed test locations.

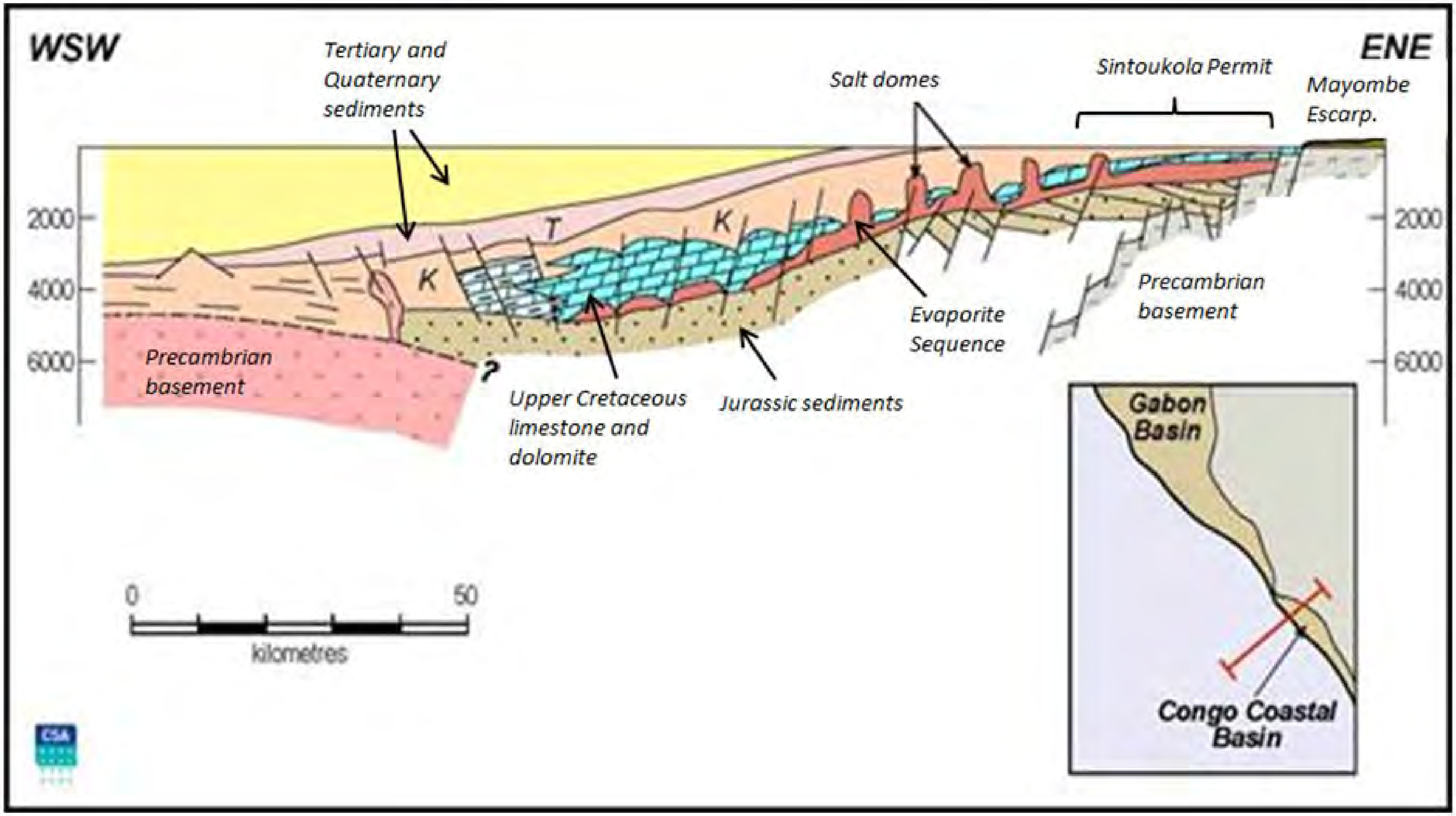
Table 5.10.1: Summary of Overburden Hydrostratigraphy and Hydrogeological Test Results

Formation		Lithology	Thick-ness (m)	Hydro-geological Role	Estimated Horizontal Hydraulic Conductivity (K_h) (m/d)	Estimated Vertical Hydraulic Conductivity (K_v) (m/d)[1]	TDS (gpl)	Comments
Unconsolidated Sediments	Neogene Sediments (Sables de Surface)	Unconsolidated clayey sand	6-20	Aquifer	Likely similar to laterite/ ferruginous sandstone	Likely similar to laterite/ ferruginous sandstone	Not sampled	Drained in some areas
Unconsolidated/ consolidated	Laterites and Iron Bearing Sandstone Formation (Sables Grossiers Ferrugineux)	Unconsolidated / consolidated ferruginous clays, sands, and sandstone	80-180	Aquifer	From 0.88 to 2.32	0.088 to 0.232	0.04 to 0.933	Tested at Sites A and B
Hard Rock Aquifers With Some Unconsolidated Layers	Grey-Blue Claystone Formation (Série Argilo-Gréso-Dolomitique)	Dolomitic siltstone	10-90	Aquifer	From 0.25 to 0.35	From 0.0025 to 0.0035	2.6 to 5.5	Tested at Sites A and B and DH-21 disturbance area.
	Dolomitic Limestone Formation (Série Argilo-Gréso-Dolomitique)	Fractured Dolomitic Limestone	30-33	Aquifer	From 1.25 to 2.6	From 1.25 to 2.6	6 To 180 (at base)	Tested at Sites A and B.
Aquitard	Anhydrite Formation (Série Anhydritique)	Anhydrite sequence	0-14.6	Aquitard where tested	From $5*10^{-3}$ to below $5*10^{-5}$	From $5*10^{-3}$ to below $5*10^{-5}$	Not sampled	Tested at Sites A, B.
	Salt Formation (Série Salifère)	Cyclical bedded evaporite sequence	400-900	Aquitard	Below $5*10^{-5}$	Below $5*10^{-5}$	Not sampled	Upper portion tested at Sites A and B

Note: Based on results of site-specific pumping test analyses conducted by numerical groundwater flow model.



srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	Figure 5-1 Geological Map of the Near Coastal Region of the Republic of Congo (Sintoukola permit red polygon)



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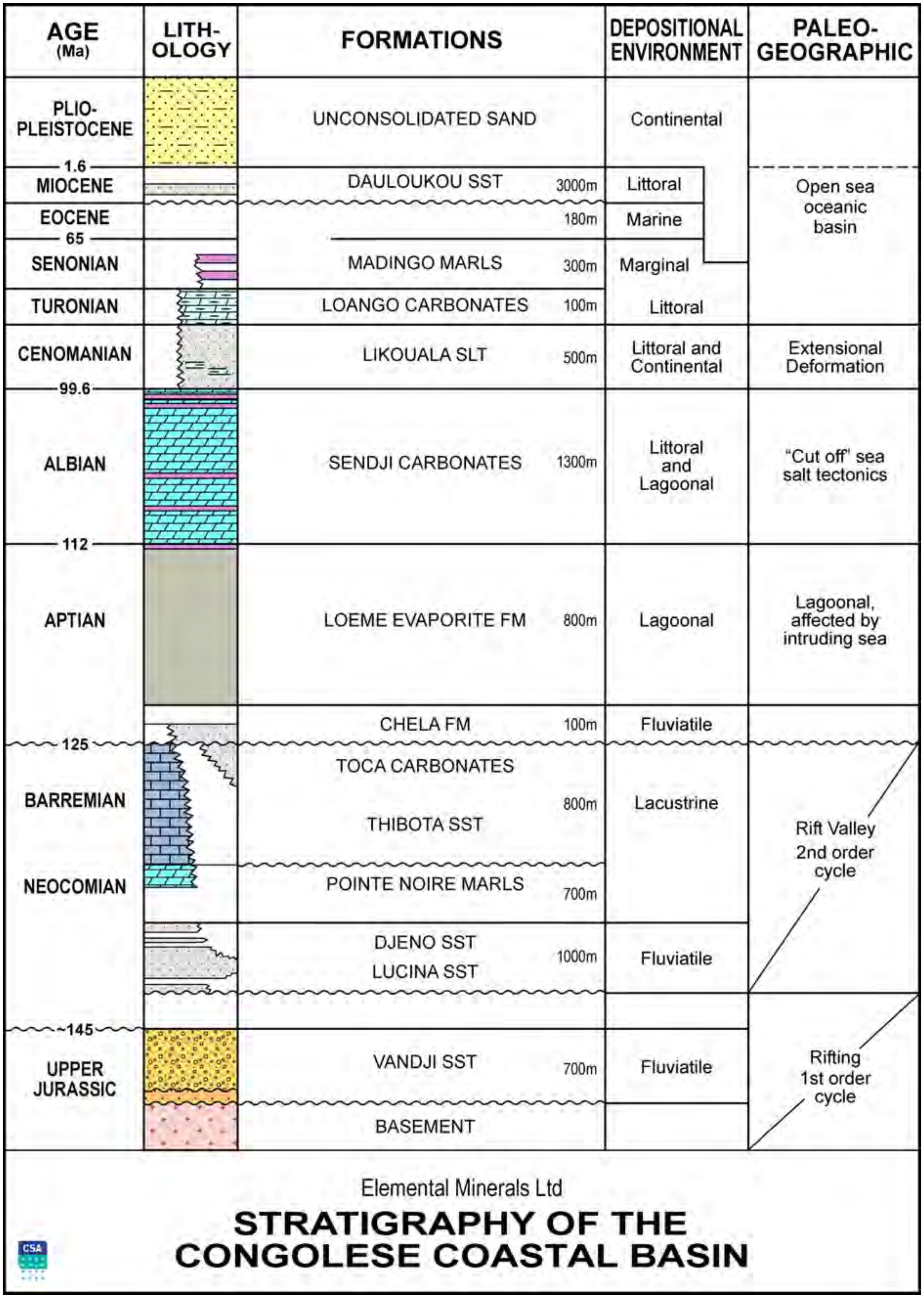
Source: CSA, 2012

Figure 5-2

Central West-African Continental Margin (after de Ruiter, 1979)



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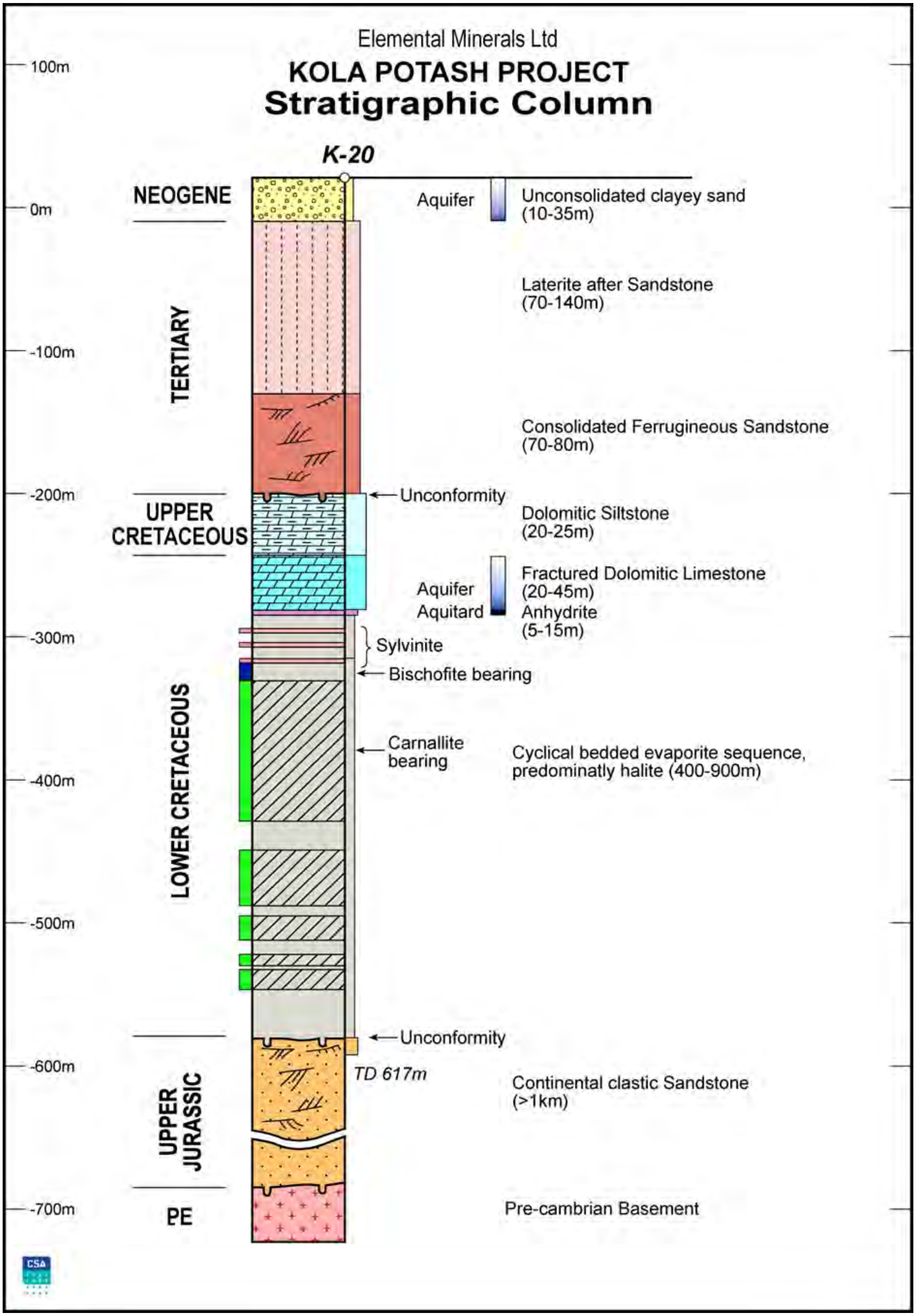
Source: CSA, 2012

Figure 5-3

Stratigraphic Column of the Onshore Congo Basin



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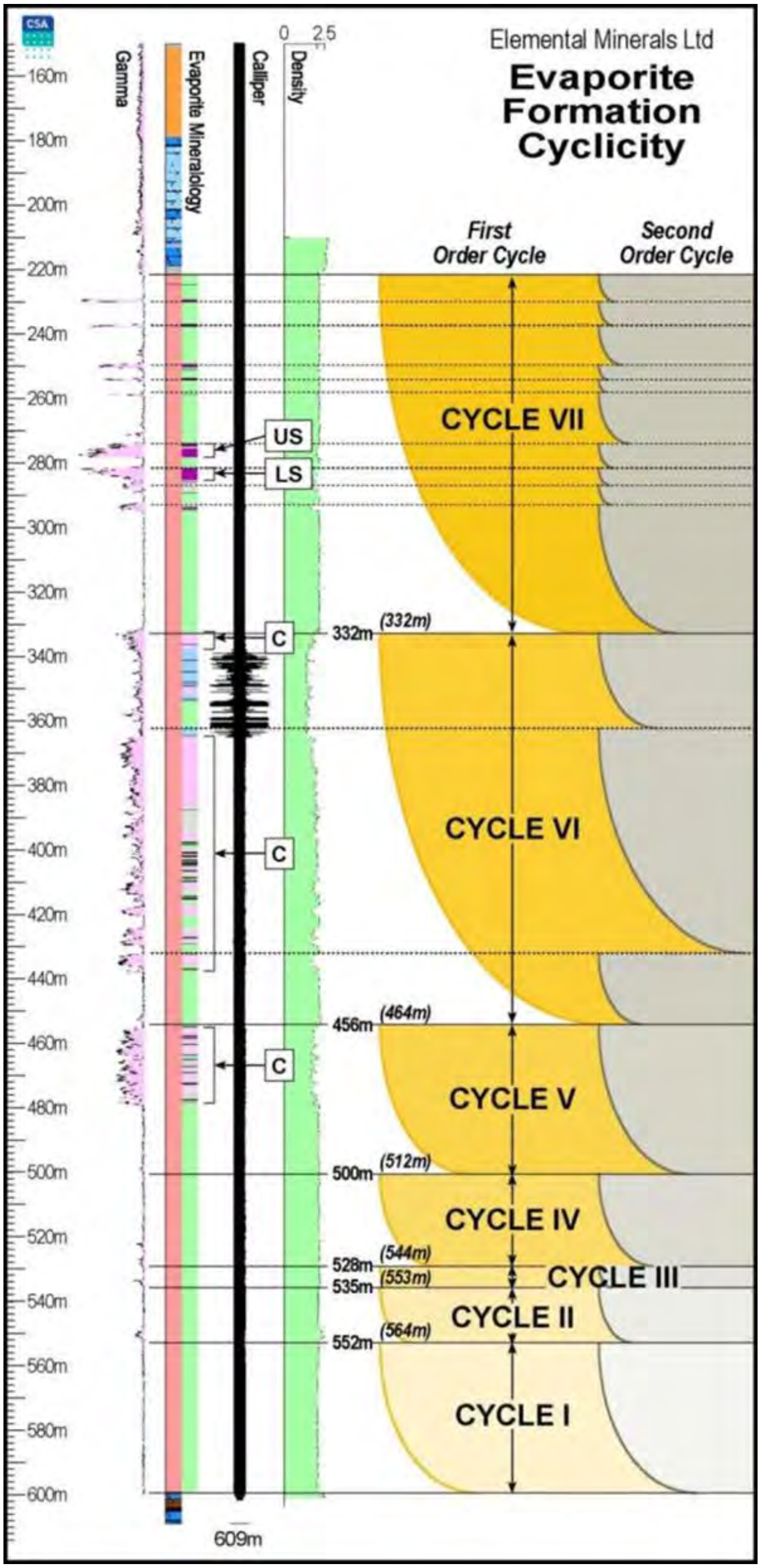
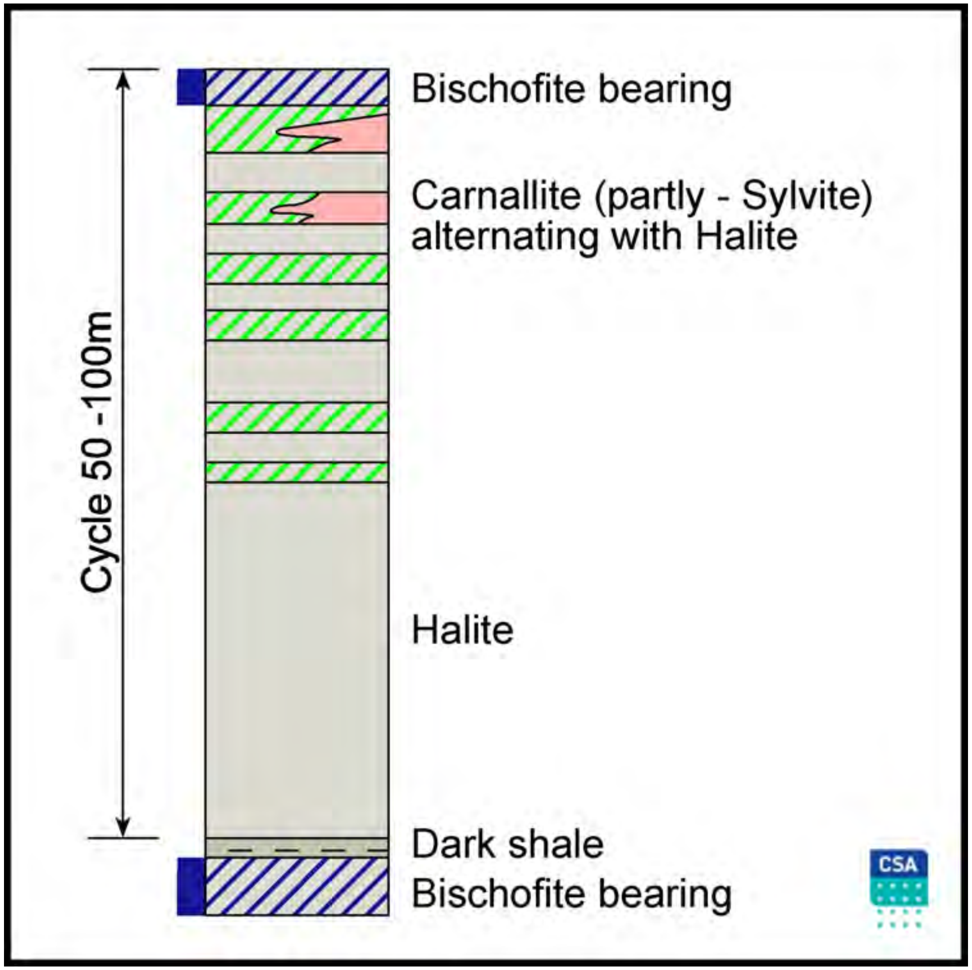
Source: CSA, 2012

Figure 5-4

Generalized Stratigraphic Column for the Kola Deposit Area



srk consulting	**Sintoukola Potash Project, Republic of Congo** elemental minerals limited **Source: CSA, 2012**	**Figure 5-5** **Internal Sequential Cyclicity of the Evaporite Formation**



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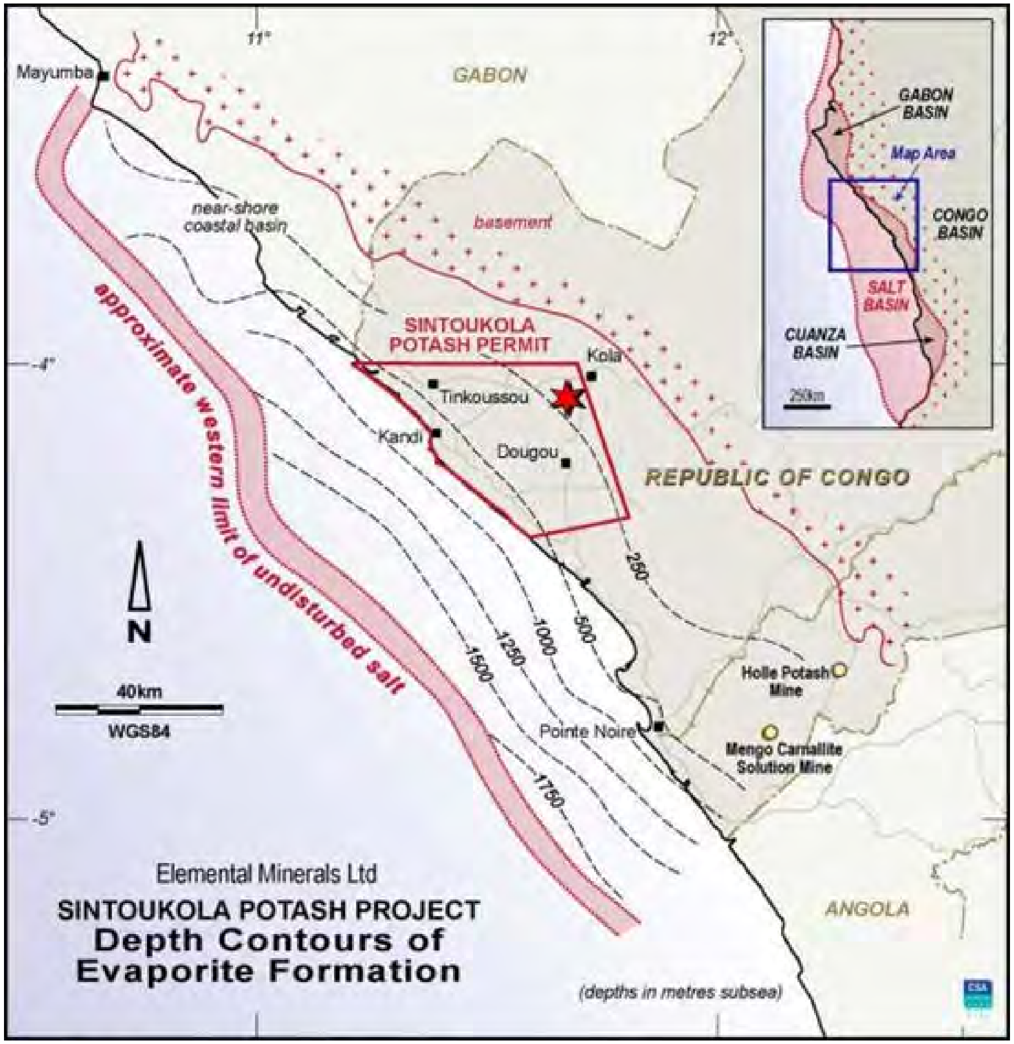
Source: CSA, 2012

Figure 5-6

Internal Sequential Arrangement of Evaporite Minerals Within Individual Depositional Cycles



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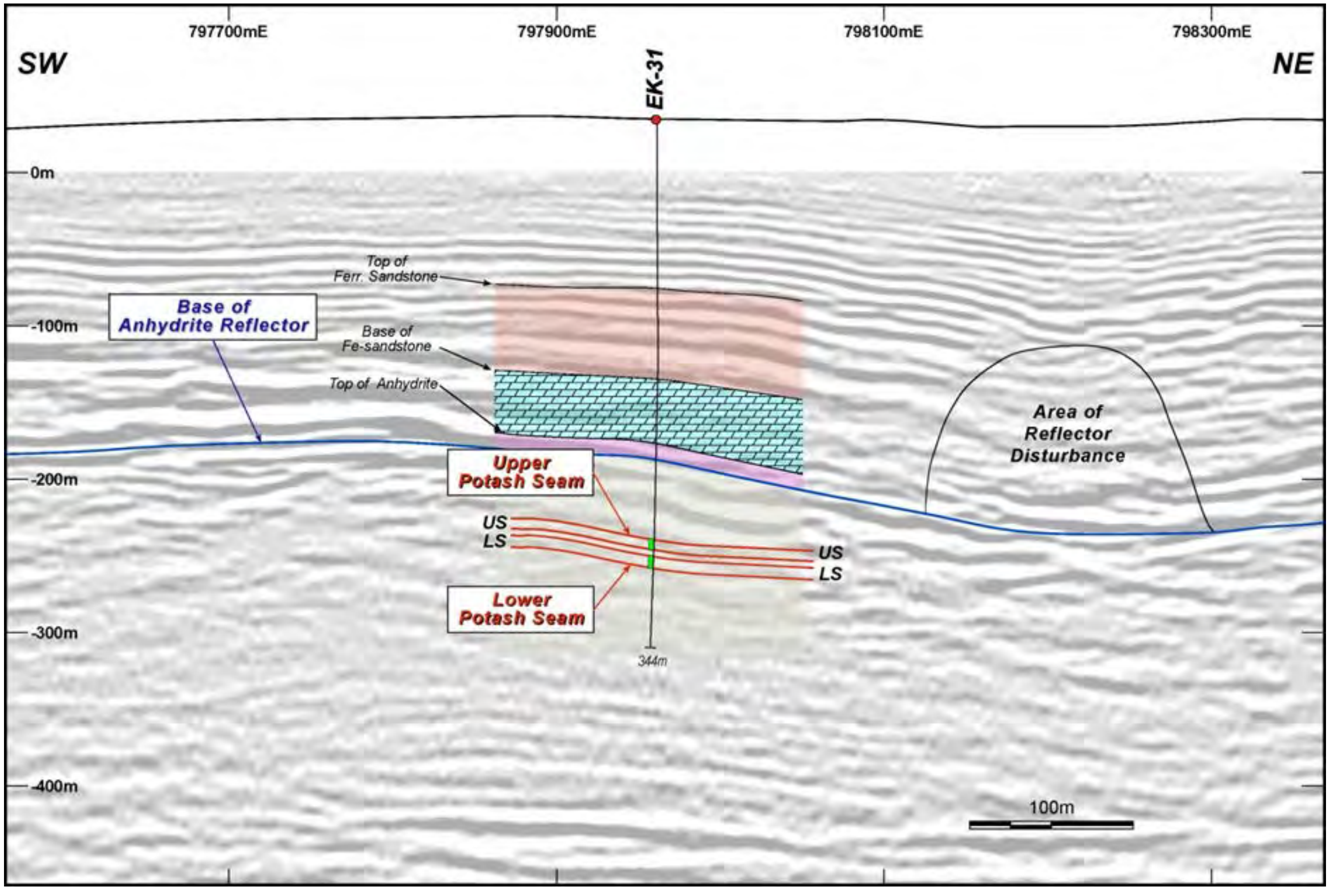
Source: CSA, 2012

Figure 5-7

Contoured Depth to Evaporite Formation (modified after de Ruiter, 1979)



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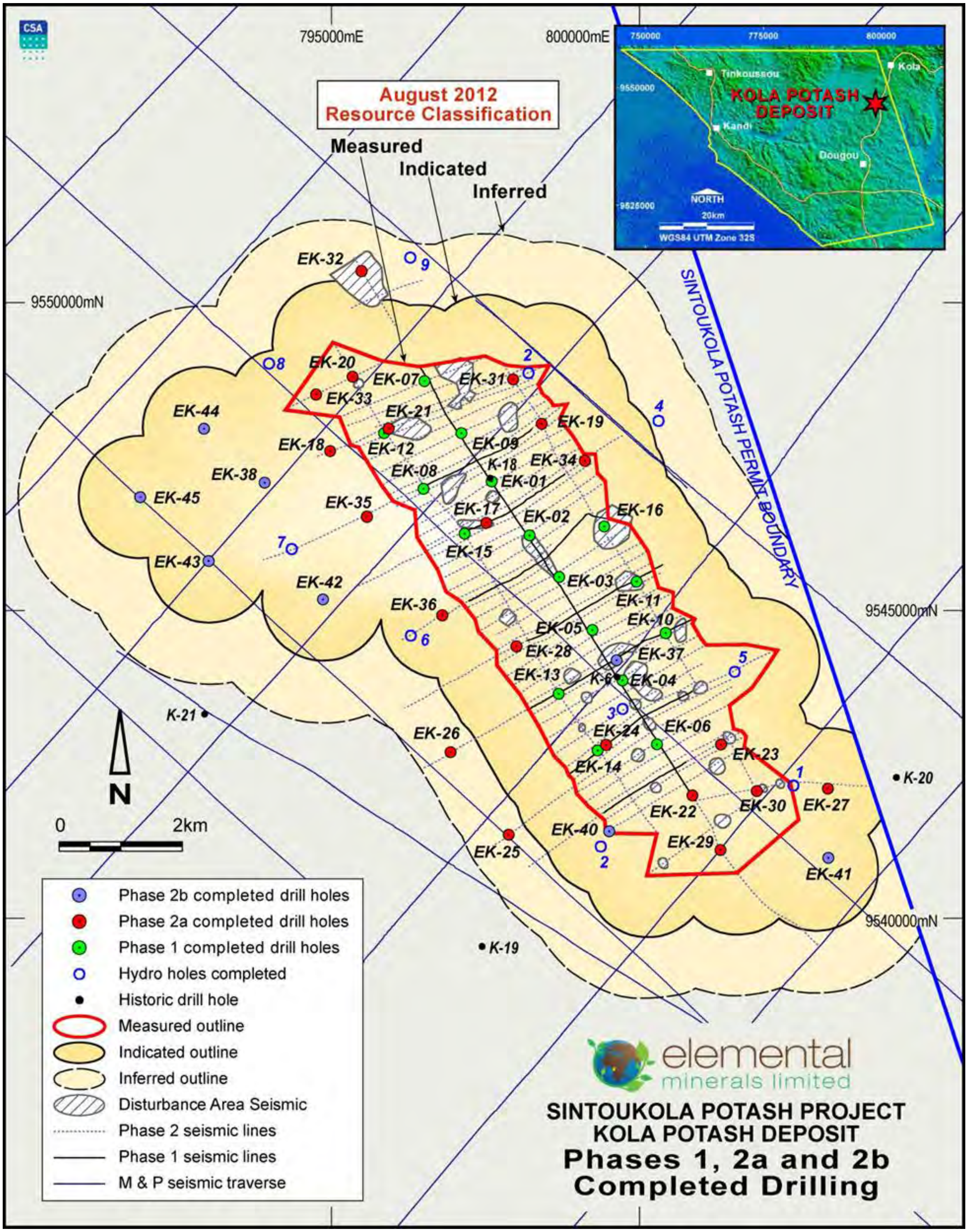
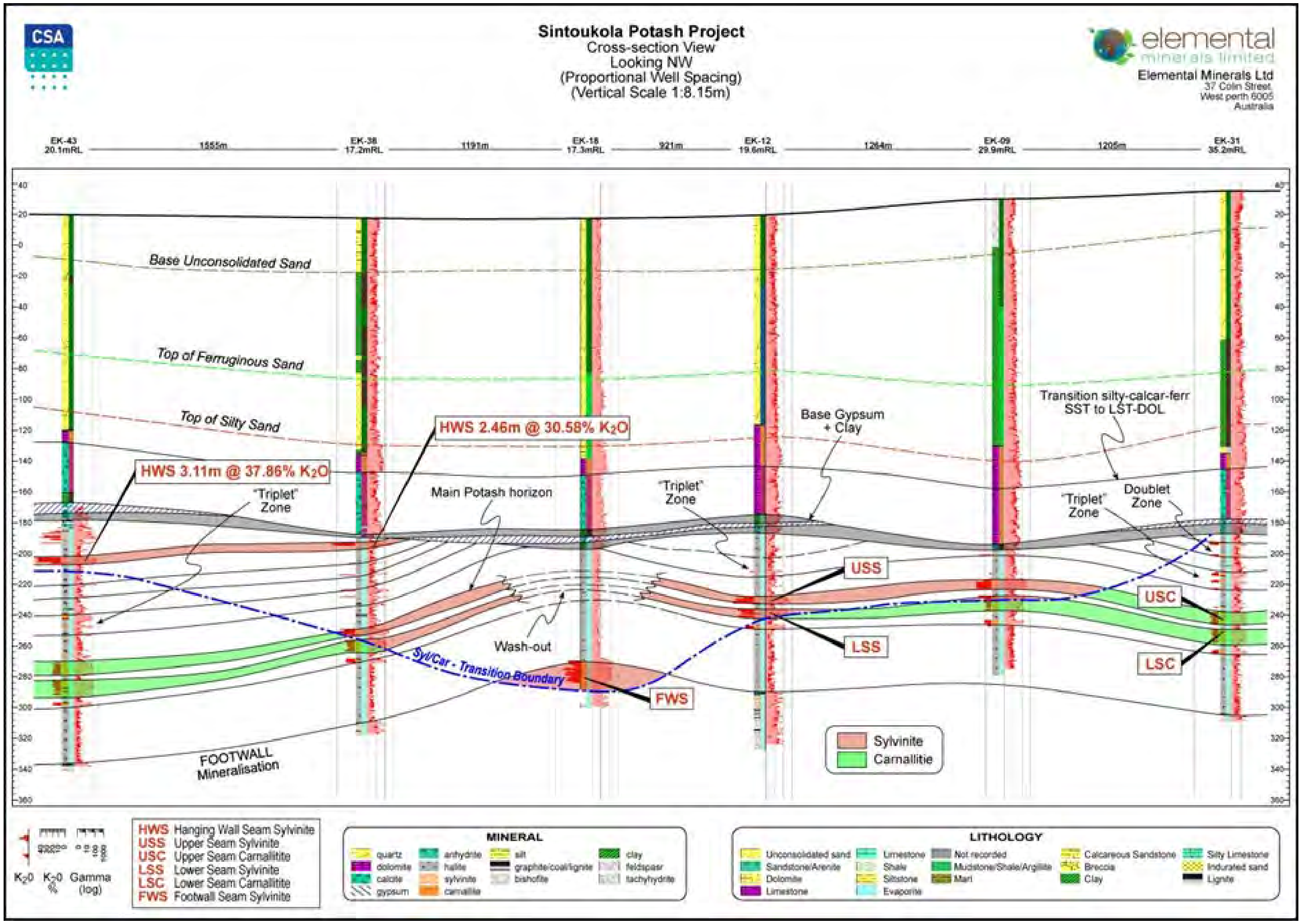
Source: CSA, 2012

Figure 5-8

Interpreted Geological Cross-Section Based on Seismic and Borehole Correlation



srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	**Figure 5-9** Disturbance Areas Mapped from Close-Spaced 2D Seismic inside the Mineral Resource classification areas for the US and LS



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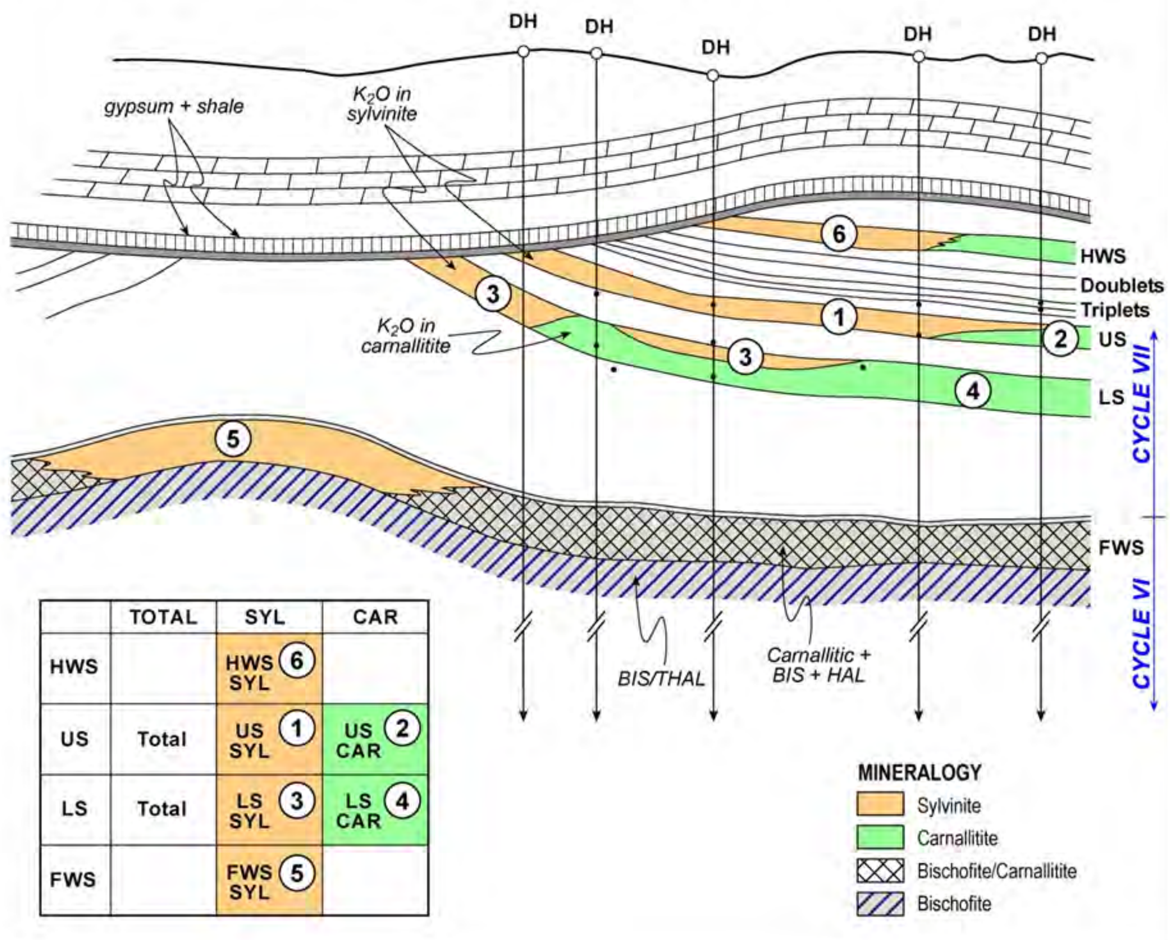
Source: CSA, 2012

Figure 5-10

Interpreted Geological Cross-Section (along cross line SP_45; vertical exaggeration of approximately 1:8)

Kola Major Potash Seams, Salt Types and Distribution



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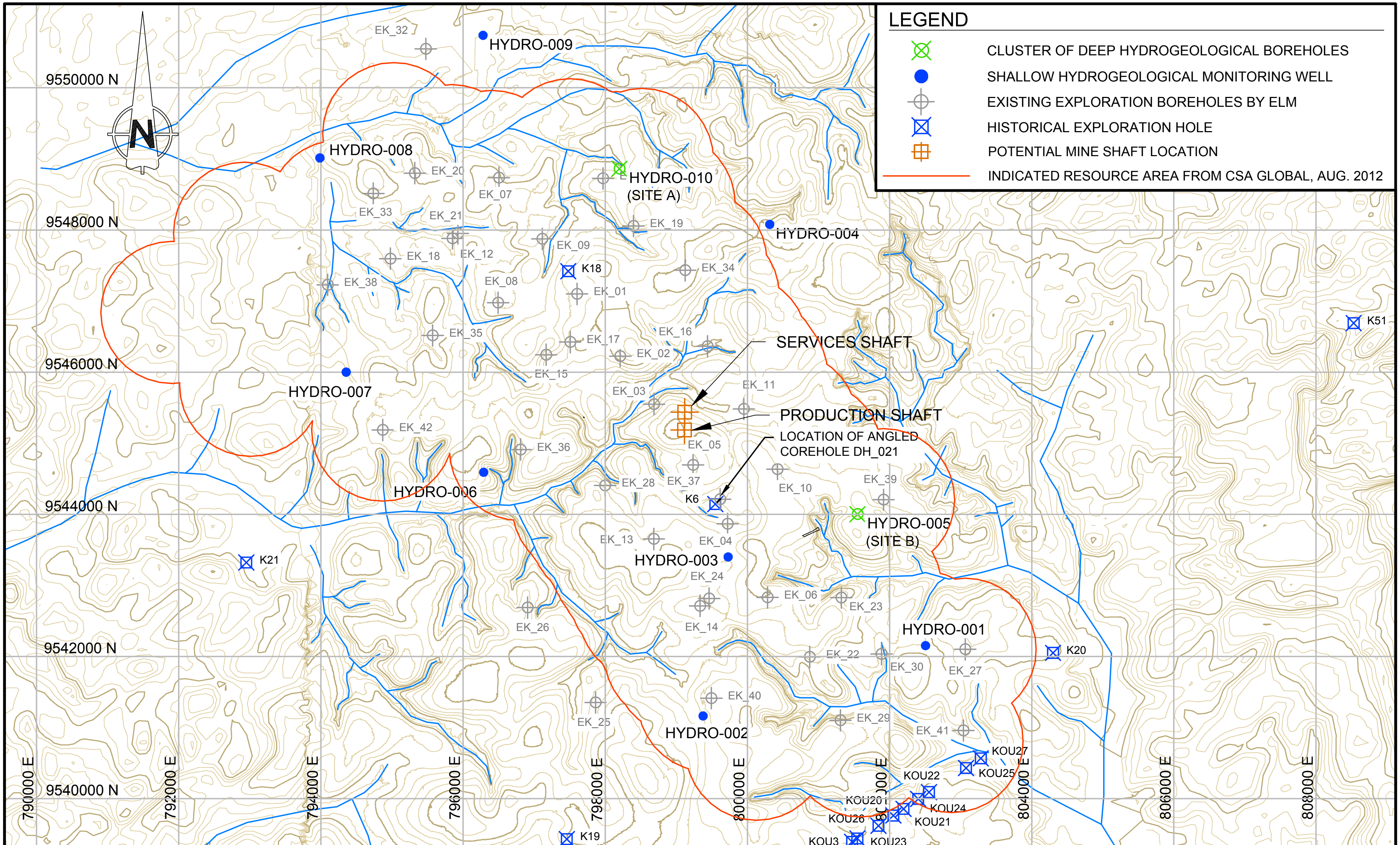
Source: CSA, 2012

Figure 5-11

Potash Salt Types and Distribution



0 1000 2000
Scale in Metres

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SRK JOB NO.: 340100.020

FILE NAME: 340100.020.Rev.A.Fig.5-12.Piezo.Install.and.Test.Locations.2012-09-12.dwg

SINTOUKOLA POTASH PROJECT/
REPUBLIC OF THE CONGO

PFS REPORT

PIEZOMETER INSTALLATION AND TESTING LOCATIONS

DATE:	APPROVED:	FIGURE:	REVISION NO.
SEPT. 2012	JB	5-12	A

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6 Deposit Type (Item 8)

6.1 Potash

The most common source of potassium (K) fertilizer is Potassium Chloride (KCl), which is also called Muriate of Potash (MoP), potash, or K60 potash product (K60 contains a minimum of 60% K_2O). A tonne of pure KCl contains an equivalent of 0.63 tonnes of K_2O. The potentially saleable product from the Sintoukola Project will be KCl.

In this report, the term "sylvinite" is used to describe the mixture of sylvite (KCl) and halite (NaCl) that is the source of the "potash". Although in insignificant amounts, sylvinite may also contain anhydrite ($CaSO_4$) as well as clay and dolomite crystals that are collectively identified in assay reports as water insolubles.

A rock consisting mainly of carnallite ($KMgCl_3{\cdot}6H_2O$), together with halite is called carnallitite. Subordinately the highly soluble salt mineral bischofite ($MgCl_2{\cdot}6H_2O$) also occurs, and although not yet identified at Sintoukola, tachyhydrite ($CaMg_2Cl_6{\cdot}12H_2O$) may also exist.

In general, sylvinite deposits consist of relatively thin beds, with KCl contents between 20% and 35%, whereas carnallitite deposits usually consist of relatively thick beds with KCl content between 8% and 17%.

6.2 Geological Model

The potash horizons targeted at the Kola deposit are the result of a natural chemical process that has its origin in a hyper-saline brine that probably originated from sea water evaporation. With increased evaporation the brine becomes saturated in various salt-forming alkaline metals and salts (chlorides) precipitate. The cyclicity within the salt formation and the internal mineralogical cyclicity within each cycle is a record of this process.

Whilst carnallite is a naturally occurring mineral, extensive research (e.g., Warren, 2006, Scheleder et al., 2008) shows that sylvite in ancient geological systems is not a primary precipitated mineral rather its formation relies on conversion from carnallite. The sylvinite horizons of the Congo coastal basin are likewise interpreted to be secondary in origin (de Ruiter, 1979).

The formation of sylvite requires that the precursor carnallite mineral reacted with a NaCl-saturated brine according to the reaction:

$$\text{Carnallite + Water = Sylvite + } MgCl_2 \text{ rich brine.}$$

It is thought that the ingress of water into the carnallitite layers was a consequence of water permeating through pore spaces or fractures in the anhydrite sequence over many millions of years and into the salt below. The rate of inflow would be expected to be higher in areas of higher permeability, or if disturbances are present, where the aquitard capacity of the cap rocks (anhydrite) is reduced.

At the Holle Potash Mine the sylvinite mineralisation was interpreted to be controlled by mostly structurally localised, diagenetic conversion of carnallite (Feuga et al., 2005). Mine plans and underground workings show that the Evaporite Formation in the Holle area, is laid into several parallel northwest-trending anticlines with sylvinite mineralization occurring along the flanks of these structures, where it was conventionally mined, (Feuga et al., 2005).

The genetic relationship between carnallitite and sylvinite mineralisation is supported by comparing the product of thickness (m) and grade (K_2O) for the two main seams at the Kola deposit (Figure 6-1). This graph is taken as evidence that the sylvinite mineralisation resulted through secondary processes (brine flushing) and removal of $MgCl_2$ from the carnallitite protolith. Most of the seam intersections fall within a very narrow band (+/- 20 m) of the average (106 m% for US and 104 m% for LS). This narrow bandwidth contrasts with the near 40% change in seam volume and grade within a single seam when comparing the two mineralogical end-members. Seam intercepts that show strong below average m% values have probably experienced significant K loss.

The process of brine flushing that is thought to affected the salt layers is also likely to be a continuous process that ultimately will lead to the complete removal of Mg and K from the system. The geological preservation of significant amounts of sylvinite in the salt sequence at the Kola deposit suggests that this process was widespread.

The graph also indicates that the “flushing” process occurred across geological boundaries. This is illustrated by the parallel and corresponding shift of the drillhole seam pairs. Overall, both seams were compositionally and volumetrically very similar, however many of the LS values are slightly lower than the US counterpart and this suggest that the LS conversion to sylvinite lags marginally behind that of the US.

6.2.1 Stratigraphy and Structure

The geology of the potash-bearing beds of the Middle Cretaceous Evaporite Formation has been documented by de Ruiter, (1979) and Feuga et al., (2005). Overall, the potash-bearing beds may be described as being a bedded sedimentary rock, deposited across the Middle Cretaceous Paleo-Atlantic seaway. These beds are remarkably consistent over large parts of coastal Congolese basin.

At the Kola deposit, drillhole and seismic data have been used to establish an internal evaporite stratigraphy (Figure 6-2) and correlate this project-wide to aid modelling of the potash-bearing horizons. Based on drillhole logs it is clear that a laterally persistent pattern of layered rock salt and potash occurs within the salt sequence (Figure 12-1 and 12-2) and that the stratigraphic position and the absolute spacing relationship remain relatively constant across the entire project area.

The recognition of the pattern of layering is based on a set of marker horizons, mineral textural changes and stratigraphic relationships. The potash layers are not of sufficient thickness or density contrast to be discernible in the seismic data. Also, given the sometimes angular relationship at the salt-anhydrite contact, only reflectors within the salt sequence can be used to support the interpretation of the potash seams.

The most useful seismic impedance occurs at the transition from halite to bischofite. The weak reflector associated with this boundary has been used as a proxy in the interpretation of the geometry of the potash seams. The parallel arrangement between this horizon and the US and LS justify this approach and from this it is inferred that the salt formations at the Kola deposit are in-large, sub-horizontally layered.

Although on a macro scale the sequence is sub-horizontal, internally it is interpreted that the salt layers at the Kola deposit occur as asymmetrically undulating layers. The amplitude of the undulations is generally expected to be gentle and a result of flexing of the layers and may not follow a regular pattern. In places, bedding/layering are inclined up to 40° relative to the axis of the core. This is not the norm (less than 10% of measurements) and is interpreted to have resulted from several causes including localized structural disturbances, salt internal plastic flow or volume loss through the conversion of carnallitite to sylvinite.

6.2.2 Mineralisation Composition

Compositionally, the salts of the Congo basin are of very high purity. Impurities in sylvinite can include carnallite and kieserite ($MgSO_4 \cdot H_2O$) and water insoluble such as gypsum and carbonates. No kieserite has yet been identified in any cores at the Kola deposit.

The determination of carnallite is based on the percent of MgO from assays (MgO is converted to $MgCl_2$ by multiplying by 2.362). Minor amounts of carnallite within within the sylvinite are considered an impurity as magnesium can be detrimental to the processing of sylvinite.

The Kola potash horizons are considered to be low in impurities and insoluble minerals compared to some other potash basins. Table 6.2.2.1 provides a tabulation of the average major element content and insoluble bulk content of individual seams (refer to Figure 6-2).

Table 6.2.2.1: Summary of the Chemistry of the Sylvinite Seams Determined by Analysis

Sylvinite Seam	K_2O %			Mg %			Insoluble %			Ca %			$ %			Na %		
	Min	Max	Av.	Min	Max	Av.	Min	Max	Av.	Min	Max	Av.	Min	Max	Av.	Min	Max	Av.
HWS	13,57	41.42	34,35	0.01	0.14	0.05	0.04	0.30	0.16	0.02	0.31	0.07	0.00	0,25	0.05	11.30	29.38	16.16
US	0.02	36.13	22,30	0.00	0.25	0.03	0.01	0.71	0.14	0.03	0.72	0.18	0.03	0.74	0.16	15.72	38.50	24.24
LS	0.11	29.74	19.57	0.01	0.62	0.04	0.02	0.24	0.11	0.09	0.49	0.21	0.07	0.60	0.18	19.30	38.20	25.87
FWS	0.24	30.58	17.45	0.01	1.38	0.10	0.30	8.53	1.50	0.05	0.82	0.31	0.04	0.65	0.23	10.98	37.10	26.15

6.2.3 Exploration & Development Considerations

It was recognized that potash beds consisting of carnallitite may be converted to high-grade sylvinite mineralisation when they occur within a certain distance from the gypsum/anhydrite contact or are laterally proximal to a “disturbance area”. Empirically this conversion may occur within a zone up to 60 m vertically below the salt/anhydrite contact, and a greater distance in areas where the FWS is developed. The process of lateral movement of pore water is likely to have been important, possibly controlled by more permeable carnallitite layers, although the lateral influence is much more difficult to quantify. This suggests that sylvinite mineralisation may develop laterally within carnallitite mineralisation from which it is derived.

The conversion of carnallitite to sylvinite coincides with an approximate 35% volume loss but is also associated with a reduction in porosity and an increase in density (Figure 6-2). It is possible that once a seam has undergone the conversion to sylvinite it becomes less permeable to formation brines and the process is slowed done. The fact that sylvinite horizons can occur below carnallitite horizons (e.g., EK_05, EK_29) further suggests that the process of conversion also took place laterally and is probably facilitated by original lower density and higher permeability of the carnallitite horizons.

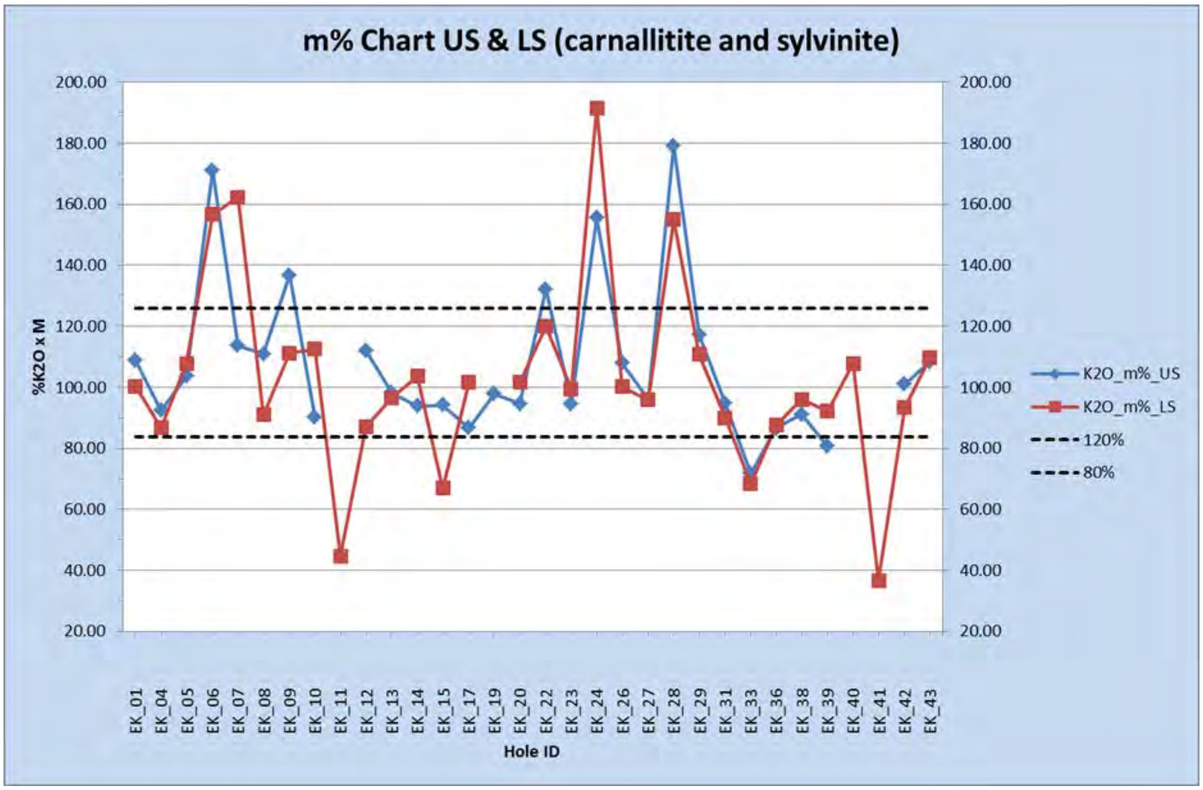
The presence of the FWS mineralisation is linked with the absence of the higher potash horizons and the conversion of carnallite at the top of the bischofite horizon. This further supports the model presented above.

The drilling also showed that several narrow sylvinite beds occur above the US (the Triplets and Doublets) and that thick stratified layers of predominately carnallitite mineralisation occur below the FWS. These narrow potash layers have not yet been modeled, but are useful marker horizons.

With a view to ongoing exploration in the project area, whatever model, or combination of models, is used to explain the origin of the potash salts, the most important economic aspects are grade, the geometry of the deposit and proximity to the anhydrite contact.

In the context of the Kola deposit setting, this is particularly important as the data to date, does not clearly indicate that the deposit area is directly affected by post deposition faulting which may provide connectivity for ground water to the mineralized horizons. Whether brittle faulting is affecting the salt/anhydrite contact and indirectly the salt underneath is a possibility that cannot be ruled out entirely. As a precaution, during development and mining the presence of faults should be a subject of ongoing investigation by underground sonic and radar technologies in horizontal (in-seam) drillholes as currently in practice at several German potash mines.

The geometry of the sylvinite mineralisation in the Kola deposit area can be simply described as a set of stacked, parallel, undulating and mostly continuous beds averaging approximately 4 m in thickness which are amenable to mining with continuous miners (CMs) and wheeled or trackless haulage equipment.



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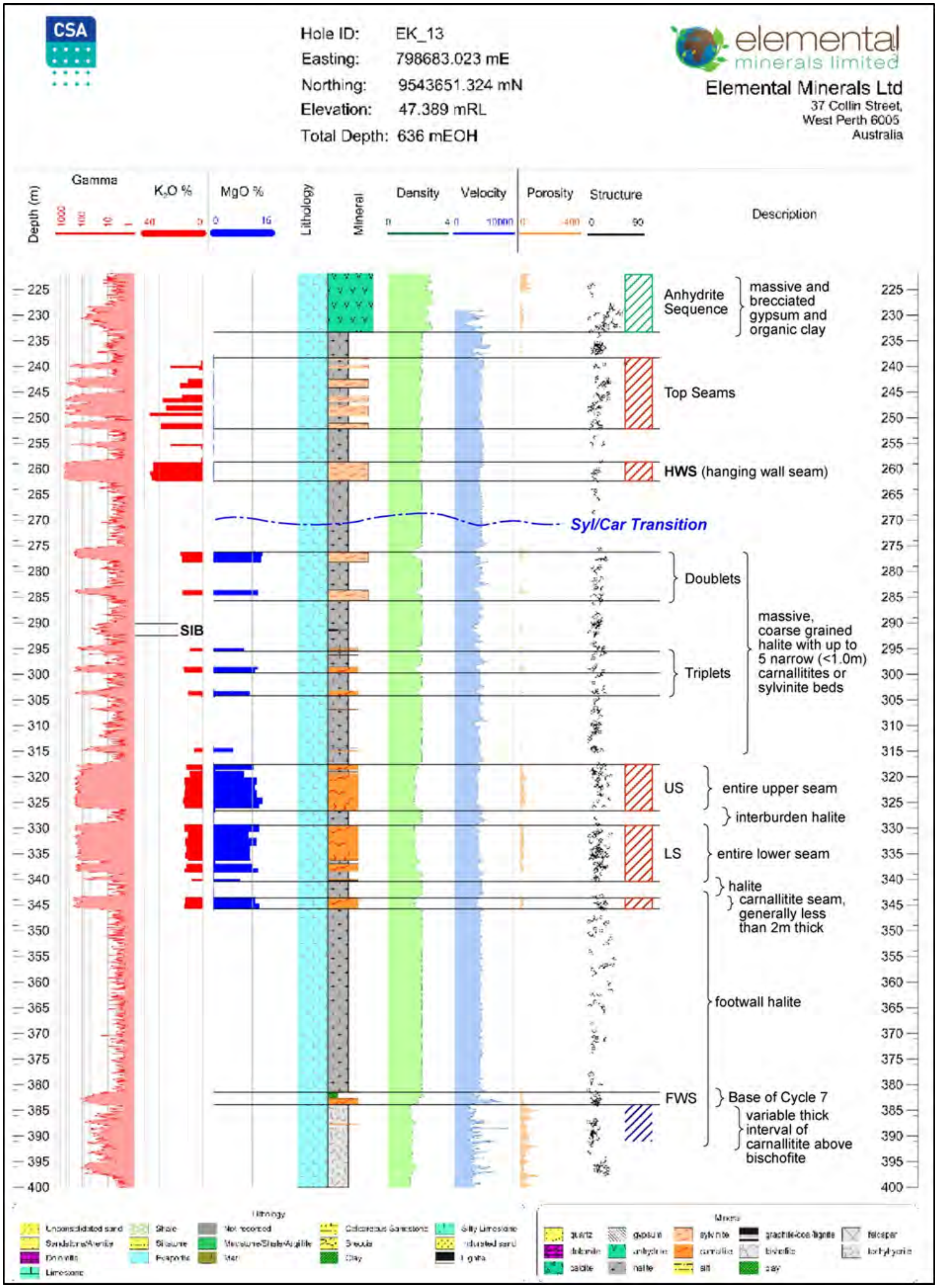
Source: CSA, 2012

Figure 6-1

Grade (K_2O) x Thickness (m) Graph for the Upper and Lower Seams



Sintoukola Potash Project, Republic of Congo

Source: CSA, 2012

Figure 6-2

Inter-Salt Layer Geometry and Marker Beds

7 Exploration (Item 9)

CSA completed an initial review of historic data for the Sintoukola Project in 2008 and 2009 on behalf of ELM to evaluate the potash potential of the project area and to develop a strategy for exploration (at that time in anticipation of being awarded an exploration permit). The evaluation of the prospectivity of the region was aided by reviews of publically available technical papers on potash projects in the region (Feuga et al., 2005, Lambert, 1967, Depege, 1967, Chouteau et al., 1997, de Ruiter, 1979).

The exploration work completed on the Sintoukola Project included topographic surveys, 2D seismic surveys, drilling and sampling (see Section 8) and mineralogical studies.

7.1 Topographic Surveying

In 2011, a detailed topographic survey using Light Detection and Ranging (LIDAR) was flown over parts of the project area. Maps were generated for the project and a digital elevation model (DEM) was produced with elevation contours at 0.1 m intervals.

Land-based topographic surveying was completed at the Sintoukola Project by GAP Geophysics (GAP) of South Africa who used a real time Differential Global Positioning System (DGPS) to position seismic lines.

Both the LIDAR and the seismic DGPS data has been reported in the Universal Transversal Mercator (UTM) projection referenced to the WGS 84 geodetic datum. The coastal area of the ROC is located in UTM zone 32 South.

7.2 Seismic Surveying

ELM has completed two seismic surveys at the Sintoukola Project both of which were completed by GAP Geophysics. Data from an additional survey has been received via a data swap with French oil exploration company Maurel et Prom (M&P). Details of the surveys, processing techniques used and methods of interpretation are provided below.

7.2.1 GAP Surveys

Between October and November 2010 and August to October 2011 GAP conducted two 2D vibroseis seismic surveys in the Kola deposit area. The seismic surveys consisted of a total of approximately 200 line km (34.5 and 164 line km respectively) of 2D seismic soundings. Vertical seismic profiling (VSP) was completed in the drillholes EK_1, EK_4, EK_6 and EK_17 to EK_36 (excluding EK_21, EK_34).

The survey extended over 54 dip and 3 strike traverses covering a block approximately 11 km (NNW) by approximately 4 km (ENE) over the Kola deposit (Figure 7-1). Dip traverses were spaced at approximately 120 to 150 m intervals and strike lines at approximately 1 km. Formation density and sonic velocity data acquired in a contemporaneous geophysical drillhole logging exercise were used for depth to two way travel time (TWTT) conversions and vice versa. Velocity/density breaks and GAP synthetic seismogram modelling predicted major reflection events at the top and base of the uppermost salt cycle (Cycle VII) and base of entire evaporite sequence.

Interpretation of the high resolution GAP seismic data over the Kola deposit area was based on drillhole ties derived from geological logs plus formation velocities, and on selected reflection signatures. Up to nine markers in the Tertiary to Lower Cretaceous-age sedimentary sequence were interpreted (with varying degrees of confidence) down to a depth of 700 m below the seismic datum of mean sea level, or 380 milli-seconds (msec) TWTT. This includes the roof of the host evaporite sequence (Figure 7-1).

The most important markers mapped at shallow depths, with a high degree of confidence include:

- The base of Fe Sandstone marker (top of dolomite;
- The base of Anhydrite Formation marker (top of evaporite/salt); and
- The base of Cycle VII rock salt (transition rock salt/bischofite).

The only marker in the evaporite sequence that was mapped (with a moderate to low degree of confidence) was the base of interburden halite (IBH).

Data quality ranges from fair to good and a predominant frequency of approximately 90 Hz within a bandwidth of 35 to 170 Hz allows for a nominal depth mapping resolution of 8 to 10 m over the more robust reflection events. Litho-seismic correlations were based upon partial geophysical sonic and density logs from 6 drillholes (4 shallow and 2 deep runs) and the corresponding drillhole geological logs. According to GAP, formation velocities vary from approximately 780 to 2200 metres per second (m/s) for Tertiary sandstones to 2700 to 4500 m/s for older sediments. Drillhole and seismic ties based on measured formation velocities were generally plausible, and mis-ties were generally less than 10 msec TWTT (approximately 20 m).

The base of Anhydrite Formation marker effectively maps the “top” of the upper salt cycle (Cycle VII) over the depth interval 160 to 240 m (below seismic datum). Mean depth is 200 m. The base of the Evaporite is mapped with a lower but still high confidence level over the depth interval 540 m to 650 m. Mean thickness of the entire evaporite sequence is around 400 m with a range of 350 to 460 m.

Despite representing the most significant density contrast within the salt sequence, the contact between the bischofite and the halite (at the base of the Cycle VII) is variably well imaged from the seismic data. This reflection event contact has been used, in combination with drillhole data, to assist with interpretation of the internal salt stratigraphy (Figure 7-1). The approximately 100 m thick Cycle VII zone is characterized by the presence of obvious seismic reflectors but extensive and coherent reflection events are absent. In particular, the potash beds of interest are only indirectly mapped below the base of Anhydrite reflection event despite thicknesses of up to 5 m or more.

The base of the evaporite marker along strike line SP-49, 50 and 51 is generally undulating. The undulations characterizing the base of the Evaporite Sequence along SP-49, 50 and 51 are only broadly mimicked along the “top” of the Evaporite (the base of Anhydrite marker).

Carbonate and anhydrite “cap” rocks are characterized by at least 5 closely spaced, largely conformable reflection events whose upper and lowermost members define the base Ferruginous Sandstone and base Anhydrite markers. The combined thickness of carbonate and anhydrite units varies between 50 and 80 m with a mean of 65 m. Some of this variance may reflect, here as elsewhere, interpretation errors in picking the wrong “leg” of a reflection event, especially over areas of seismic reflector disturbance.

Structural anomalous areas are characterized by discontinuous seismic signals and probably reflect areas of increased salt dissolution. The seismic disturbance areas are normally mapped as shallow inverse V-shaped features which may be symmetrical or asymmetrical in profile. The disturbance areas have extents of up to 400 m width (exceptionally 600 m) and are locally associated with topographic lows of up to 40 m elevation. Despite the highly visible signatures on the seismic sections, slope dips rarely exceed 15° to 20° on seismic data. Cap rock sequences appear to be generally preserved through these disturbance zones, but are disturbed or reduced in places.

7.2.2 Maurel et Prom Survey

During 2011, ELM entered into an agreement with oil explorer M&P of France. The agreement specified that a swap of drilling and seismic data would be undertaken. As a result of the deal, ELM acquired data from seven recent and historic oil exploration holes within the Kola deposit area and approximately 200 line km of recent 2D seismic (Figure 7-2). The seismic data was acquired in the late 1980's and reprocessed in 2006 and 2007 and was collected on a 3 by 3 km grid.

Despite different acquisition parameters GAP established that the two data sets could be combined and used for interpretation and therefore the same parameters were used to processes the M&P seismic data. Interpretation of the low resolution M&P data also followed the same principles and guidelines as for the GAP data, with some exceptions (discussed below) due to the lower frequency of the seismic signal used in the survey.

After applying a bulk shift of 50 msec to the data, the base-sandstone horizon correlated with a "stand-alone" black peak event that is correlated with the base Ferruginous Sandstone marker horizon which was mapped along the shallowest highest amplitude, continuous black peak event. The base-Anhydrite was mapped along the strong and continuous black peak reflection event some 25 msec to 40 msec immediately below the black-peak base-sandstone black-peak reflection event. This corresponds to a middling interval of 45 to 75 m and is in line with drilling results from the Kola deposit area (median middling is roughly 60 m). This black peak event correlates with GAP high frequency data, but for the most part the middling interval of the M&P data is, unlike that of the GAP data, is free of closely-spaced reflectors. This classic doublet signature is well imaged over the western and eastern sectors of the NW-SE traverses and particularly over the northern sectors of the NE-SW traverses.

7.2.3 Comments on the Seismic Interpretation

The conversion of time sections to depth sections, which are required to integrate the seismic data in the geological model, is approximate only. Thick overburden sand layers can attenuate the seismic signal which results in uncertain velocity models. This may result in errors in the range of up to 20 m for the depth of a particular horizon. The depth model generated from the velocity profiles indicated that the depth discrepancies were within acceptable accuracy for the current stage of the project and therefore these data were used in the generation of a 3D geological model.

Seismic techniques are however unable to delineate the potash beds due to insufficient contrast between the halite and sylvite densities. However, the density difference between halite and anhydrite allows seismic techniques to map the base of the anhydrite with reasonable certainty.

Contour maps of the potash beds are prepared from a combination of seismic data interpretation and from drillhole data. Because the contours of the base of the anhydrite (top of salt) and contours of the potash beds were produced relying on interpretations with different levels of confidence, the reader is cautioned that, although there is no real evidence indicating so, there may be areas of the deposit with thinner rock salt cover between the base of Anhydrite Formation and the roof of the USS.

The Phase 2b drilling program provided another opportunity to test the accuracy of the seismic model. Three exploration and two hydrological drillholes from the Phase 2b program were completed within the area of the high density 2D seismic interpretation. The modeled surface generated from the base of Anhydrite Formation reflector was compared with the actual depth of the base of anhydrite in the Phase 2b drillholes. The comparison determined that the average error of the marker was approximately 6.5 m which is considered acceptable given the depth and data density.

The seismic survey results suggest that on the scale of the deposit the reflectors are mostly continuous, although some local loss of continuity occurs and this is thought to be associated with isolated disturbance areas. The reflectors indicate that beds are subtly undulating and in places truncations of the seams may occur. Overall, the seismic data suggests good continuity of all horizons, with gentle dips towards the southwest and northeast.

7.3 Down hole Geophysical Logging

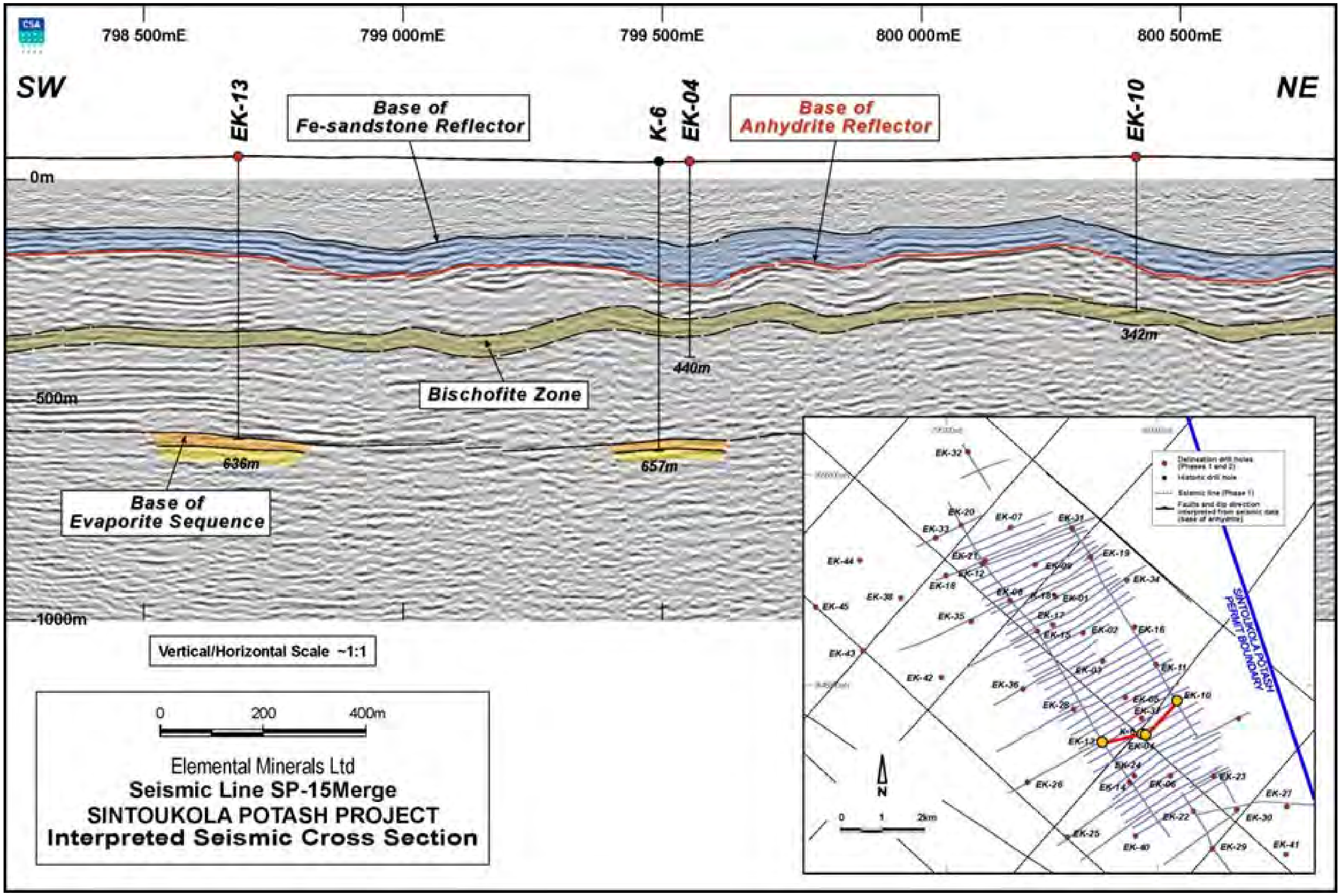
Upon retrieving the core, each drillhole was logged from total depth (TD) to surface with geophysical wireline tools. The geophysical logging was completed to provide detailed down hole information that can be used to cross-reference lithology, mineralogy, and geochemical assay data. Geophysical logging was always completed in tri-salt mud filled holes.

SEMM Logging, a subsidiary of Schlumberger of Paris, France, were contracted to complete the logging for both drilling programs. Of the geophysical parameters measured with the wireline tools, the gamma log, lithology density, resistivity, rock velocity and photoelectric logs were the most useful data when completing detailed geological logging. The gamma log provides a record of radioactivity and is displayed in American Petroleum Institute (API) units. The gamma emissions from the K40 potassium isotope are correlated to the potassium content of the rock. API units are proportional to the sylvite concentration of the rock; the gamma log therefore can provide an indication of the potash grade. The lithology density, porosity and photoelectric data provide means of assessing mineralogy.

7.4 Petrology

Between December 2011 and August 16, 2012, 39 core samples were submitted for petrographical analysis. The petrographic work was conducted at the Geology Department of the Technical University of Aachen (Germany). The samples represent a selection of rock types intersected in core. The petrographic work was accompanied by quantitative XRD analysis.

The objective of the petrographic work was to assess the mineralogical, textural and genetic properties of the salt minerals and their inter-relationships. The results of the work (Table 5.9.1) support the results of the quantitative mineralogical analysis discussed above (Section 9.7).



srk consulting

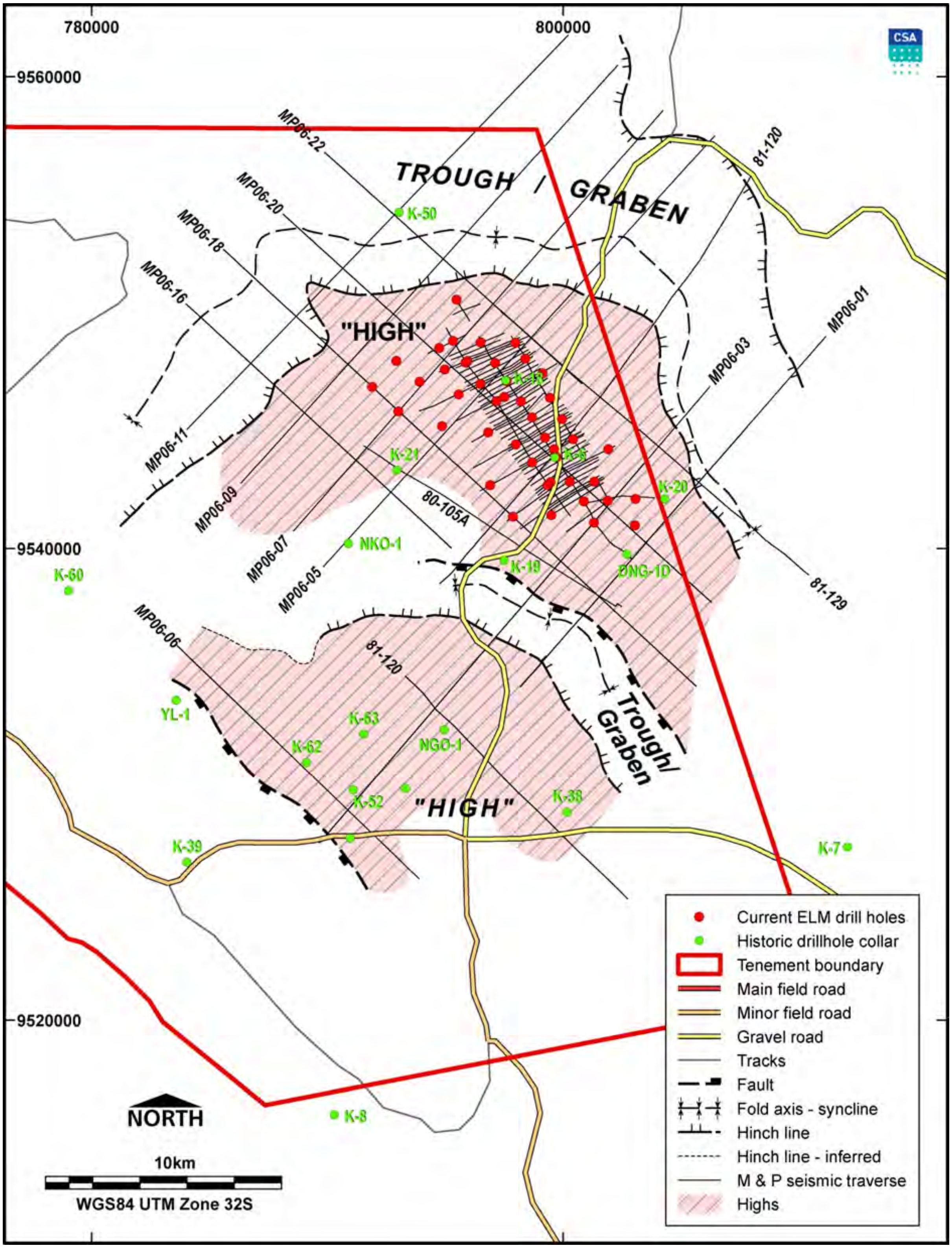
elemental minerals limited

Sintoukola Potash Project
Republic of Congo

Source: CSA, 2012

Figure 7-1

Illustration of Seismic Reflector Patterns and Main Reflectors



srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	**Figure 7-2** **Plan of Seismic Surveys and Drill Holes in the Vicinity of the Kola Potash Deposit**

8 Drilling (Item 10)

Drilling in the region of the Sintoukola Project has been completed during two main periods and these are classified as (i) historical drilling and (ii) ELM's drilling. ELM has now completed three phases of exploration drilling. The details of each campaign are described below.

8.1 Drilling

As part of the initial project review CSA collated as much of the historical regional exploration data as was available. The data largely comprised drilling records from the 1960's that were translated from French to English. The historic drill data was used to generate the initial geological model and exploration target upon which ELM's maiden exploration drilling program was planned.

Exploration drilling within the Sintoukola Project area has now occurred over three periods that are classified as (i) historical, (ii) ELM Phase I, (iii) ELM Phase 2a and 2b programs. A summary of the drillholes series, total metres and type of drilling of each campaign are listed in Table 8.1.1 and a plan of the drilling is provided as Figure 8-1.

Table 8.1.1: Summary of Drill Programs Completed in the Immediate Kola Deposit Area

Period	DH Series	Hole Type	No. DH	DD (m)	RC (m)	Total (m)	Purpose	Wireline Log
Pre-1969*	K-5,6,18,19,20,21*	DD	6	n/a	n/a	4,802	Oil exploration	Historic
2010/2011	EK_01-EK_16	DD/RC	16	2,999	3,530	6,529	K exploration	Yes
2011/2012	EK_17-EK_36	DD/RC	20	2,022	4,407	6,429	K Expl./Geotech./ Hydro.	Yes
2011/2012	Hydro_01-Hydro_10 (monitoring wells) and two sets of stratigraphic wells	RC	19		2,615	2,615	Hydrology	No
2012	EK_37-EK_45	DD	9	2,234		2,234	K Expl./Geotech./ Hydro.	Yes
2012	Hydro_05-Hydro_10 EK_37	RC	19		763	763	Hydrology	No
2011	Maurel & Prom	Oil	9	n/a	n/a	n/a	Oil exploration	Historic

*Drillholes that are historic and known to ELM.

8.2 Historical Drilling

Approximately 31 historic drillholes for which the company has records occur within and adjacent to the Sintoukola permit area. These holes were drilled during the early 1960's mainly by Syndicat de Recherches de Potasse au Congo (SDRDPC) as part of the initial exploration for potash in the region.

Of the historical drillholes within the project area, five drillholes returned significant potash results and two thereof were considered to be of very high importance. The two drillholes (K-18 and K-6) were the initial target for ELM's exploration program and are located at the centre of what is now known to be the Kola deposit. Summary data of the five historic drillholes of most importance are shown in Table 8.2.1.

Detailed Well Completion reports (in French) are available for all five drillholes. Drilling was completed using a universal drill rig (IDECO H-25) which drilled a precollar using the rotary method to the anhydrite and then cored to the end of hole.

The geological and analytical data from these drillholes served as a base to develop the initial stratigraphic column and when combined with the historical seismic data allowed creation of a geological model as discussed in previous chapters.

Table 8.2.1: Summary Details for Historic Drillholes in the Immediate Kola Deposit Area

Drillhole ID	Drillhole Type	East	North	Max Depth	Rotary Depth	No. of core Samples	Wireline Log
K_06	DD	799649	9543861	657	N/A	N/A	yes
K_18	DD	797563	9547128	629	N/A	N/A	yes
K_19	DD	797500	9539500	724	N/A	N/A	yes
K_20	DD	804300	9542100	617	N/A	N/A	yes
K_21	DD	792950	9543320	660	N/A	N/A	yes

More detailed technical information can be found in Schlund and Voucorbell (1961) and Schlund et al. (1960).

8.3 ELM's Drilling

8.3.1 Phase 1

ELM's Phase 1 drilling program was designed as a two part program comprising validation by twin hole drilling component of historical drilling results from the Kola deposit (then a prospect), and an exploration program to test extensions to the historically intersected potash mineralisation.

The twin drillholes were executed at the site of historical drillholes K-18 and K-6 in August and September 2010 and by February 2011 the entire program was completed after a total of 16 diamond drillholes for 6,429 m were drilled. The program comprised two validation / twin drillholes (see Figure 8-2) and 14 drillholes for mineralisation delineation purposes. Key parameters for the drilling program are listed in Table 8.3.6.1, a plan of all the drilling is provided as Figure 8-1.

The key objective of the drilling program was to delineate mineralisation on both the sylvinite and carnallitite horizons in the Kola deposit area and to collect sufficient information to allow preparation of the maiden Mineral Resource estimate.

Drillhole locations were selected based on historic information regarding the geology and structure of the area and an average 1 by 1 km spacing between adjacent drillholes collars. The following conceptual parameters guided drillhole planning:

- Inferred presence of laterally continuous potash-bearing horizons;
- Delineating characteristic and geometry of lateral extensive potash bearing horizons; and
- Acquiring drillhole data suitable to support estimation of a potash Mineral Resource.

The drillholes were designed to evaluate the potash mineral potential of the lower portion of the Upper Cretaceous Evaporite sequence. All drillholes were drilled vertically, planned to penetrated at least the upper potash-mineralized members of the evaporite sequence and were planned to retrieve core from the anhydrite marker through to the end of the drillholes.

The Phase 1 program was successful in that only one drillhole (EK_03) had to be abandoned due to drilling problems; all other drillholes were successfully completed to target depths. All but two of the drillholes were geophysically logged with the exceptions being EK_03 (abandoned) and EK_07 which collapsed before it could be surveyed.

8.3.2 Phase 2a

The Phase 2a exploration program commenced in August 2011 and was completed in March 2012. The primary aim of the drilling was to increase the level of confidence in the Mineral Resource estimate whilst also expanding the footprint of deposit. The drilling was also designed to provide geotechnical and hydrogeological data in support of engineering studies into the development of the Kola deposit.

The second phase of exploration comprised 20 vertical diamond drillholes (EK_17 to EK_36) for a total of 6,429 m. This combined with the 16 vertical diamond drillholes (EK_01 to EK_16) for 6,529 m from the Phase 1 exploration program at Kola deposit resulted in a total of 36 exploration drillholes into the deposit for some 12,958 m. Of the 20 drillholes three failed to intersect the salt formation due to technical difficulties related to drilling (EK_21, 31, and 34). The two phases of exploration completed by ELM provided the geological and analytical data on which to base the geological model and to update the Mineral Resource estimates.

8.3.3 Phase 2b

The Phase 2b exploration program commenced in April 2012 and was completed in July 2012. The primary aim of the drilling was to increase the level of confidence in the Mineral Resource estimate whilst also expanding the footprint of the deposit. In addition, a new potash seam was targeted with the objective of targeting additional resource potential. The drilling included further hydrogeological texting in support of engineering studies into the development of the Kola deposit.

The Phase 2b of exploration comprised 8 vertical diamond drillholes (EK_38 to EK_45) for a total of 2,734.25 m and 1 angled drillhole (EK_37) for hydrological test work. All 9 drillholes reached their target depth.

At the completion of Phase 2b ELM had completed a total of 45 exploration drillholes into the deposit for some 15,949.43 m. The three phases of exploration completed by ELM provided the geological and analytical data on which to base the geological model and to update the Mineral Resource estimates.

8.3.4 Surveying

Drillhole collars surveys were conducted using DGPS equipment by Kirchhoff Surveyors of South Africa. Survey reports were delivered to ELM by e-mail in Excel files.

A review of the drillhole collar data and comparison with the digital terrain model showed that recent drillholes have not been found to show any anomalous Relative Level (RL) values.

CSA also undertook to verify randomly selected drillhole collar position using a Global Positioning System (GPS) devise. CSA found that all selected drillhole collars were within accuracy of the instrument.

Down hole deviation was measured using the down hole wireline tool operated by SEMM Logging. Measurements were taken from the bottom of each drillhole upwards, and subsequent measurements were then taken at 1cm intervals and at the collar position of the drillhole. Down hole survey reports were delivered to Elements by e-mail in Excel® files.

Only one drillhole (EK_07) could not be surveyed due to the drillhole collapsing.

8.3.5 Validation Drilling

The validation phase (part of Phase 1) targeted the two historic drillholes (K-18 and K-6) by drilling a "twin" drillhole (EK_01 and EK_04 respectively) in proximity to each of the historic collar positions. The siting of the twin drillholes was refined following the positive (in-field) identification of the historic collars locations.

The twin drillhole program returned a near perfect geological match between expected and actual geology and mineralisation (see Figures 8-2)**.** The results confirmed the integrity of historic data and provided suitable confidence in the geological model and the prospectivity of the area to proceed with the delineation program as planned.

During Phase 2, two drillholes were drilled sufficiently close to two Phase 1 drillholes to be considered as reasonable tests of the previous resource model, the variability of the mineralisation system and its lateral continuity (Figure 8-3). The upper evaporite sequence can display some lateral variability in both the mineral composition, preservation of thickness and it position below the base of anhydrite.

As discussed in Section 5, the distance of the potash horizons from the base of anhydrite has a bearing on the composition of the potash horizons which is illustrated in Figure 8-3 comparing results of EK_15 and 17. In EK_17 the US and LS are sylvinite with a reduced thickness whereas in EK_15, where the potash layers are further below the anhydrite base, only the US occurs as sylvinite; the LS is carnallitite. Given the constraints and implications of the geological model a comparison of corresponding seams indicates good geological and seamgrade correlation.

8.3.6 Delineation Drilling

The Phase 1 and Phase 2a and 2b drilling programs were completed over an area of approximately 12 km x 5 km surrounding the historical drillholes. Drillholes were typically spaced at approximately 1 km intervals along a NW-SE baseline and at similar spacings along 1 km spaced cross lines (see Figure 8-1).

Table 8.3.6.1: Table of Drillholes Completed Between 2010 to 2012 by ELM

Drillhole ID	Drillhole Type	East	North	RL	Max Depth	No. of core Samples	DD core (m)	Rotary Depth	Wire-line Log
Phase 1									
EK_01	MR/DDH	797604.55	9547098.68	41.43	609.35	139	388.35	221	yes
EK_02	MR/DDH	798211.65	9546225.64	53.99	309	24	66	243	yes
EK_03	MR/DDH	798686.74	9545549.28	24.66	271.4		49.4	222	n/a.
EK_04	MR/DDH	799721.78	9543865.33	34.45	440.46	49	218.46	222	yes
EK_05	MR/DDH	799235.09	9544693.43	38.32	315.15	44	106.15	209	yes
EK_06	MR/DDH	800284.11	9542829.85	49.40	650.9	210	425.9	225	yes
EK_07	MR/DDH	796505.20	9548735.45	26.09	342.1	38	119.1	223	n.a.
EK_08	MR/DDH	796493.94	9546975.90	30.42	329.55	41	136.55	193	yes
EK_09	MR/DDH	797116.04	9547873.21	29.91	309.2	27	82.2	227	yes
EK_10	MR/DDH	800424.00	9544635.00	45.10	342.25	40	114.25	228	yes
EK_11	MR/DDH	799950.10	9545480.55	29.01	318.2	34	91.8	226.4	partial
EK_12	MR/DDH	795852.49	9547881.26	19.64	347.2	42	146	201.2	yes
EK_13	MR/DDH	798683.02	9543651.32	47.39	636	229	412	224	yes
EK_14	MR/DDH	799334.00	9542711.00	43.38	383.6	39	157.6	226	yes
EK_15	MR/DDH	797168.26	9546244.66	34.12	336.33	25	110.33	226	yes
EK_16	MR/DDH	799440.00	9546364.00	25.00	588	217	374.5	213.5	partial
Phase 2a									
EK_17	MR/DDH	797507.23	9546423.04	45.84	337.6	52	106.6	231	yes
EK_18	MR/DDH	794976.62	9547596.23	17.33	317.45	27	215.45	102	yes
EK_19	MR/DDH	798396.48	9548055.22	38.47	302.06	23	79.51	222.55	yes
EK_20	MR/DDH	795322.6	9548799.75	25.12	320.45	48	204.25	116.2	yes
EK_21	MR/DDH	795928.17	9547951.21	18.14	209.88		1.03	208.85	n.a
EK_22	MR/DDH	800876.83	9541992.75	31.92	378.16	31	137.21	240.95	yes
EK_23	MR/DDH	801320.4	9542828.09	35.14	362.45	55	107.45	255	yes
EK_24	MR/DDH	799462.12	9542814.67	38.77	345.22	72	117.36	227.86	yes
EK_25	MR/DDH	797864.56	9541351.31	36.31	287.3		28.6	258.7	yes
EK_26	MR/DDH	796908.88	9542686.81	37.31	383.25	48	131.25	252	yes
EK_27	MR/DDH	803063.39	9542099.4	34.08	365.35	45	100.35	265	yes
EK_28	MR/DDH	797998.95	9544406.69	37.17	339.22	42	118.33	220.89	yes
EK_29	MR/DDH	801309.48	9541101.01	27.44	368.4	58	138.9	229.5	yes
EK_30	MR/DDH	801888.23	9542032.48	14.91	237.6		6.8	230.8	yes
EK_31	MR/DDH	797969.273	9548724.189	35.172	344.25	76	130.36	213.89	yes
EK_32	MR/DDH	795475.7	9550547.55	18.199	302.3	16	39.08	263.22	yes
EK_33	MR/DDH	794738	9548513	15.31	332.3	60	140.8	191.5	yes
EK_34	MR/DDH	799125	9547432	53.08	264.15		30.65	233.5	n.a.
EK_35	MR/DDH	795573	9546516	22.81	278.3	15	56.6	221.7	yes
EK_36	MR/DDH	796809	9544911	33.56	353.3	13	131.6	221.7	yes
Phase 2b									
EK_37	MR/DDH	799616.00	9544212.00	34.00	318.20	63	206.6	50.9	n.a.
EK_38	MR/DDH	793905.57	9547076.10	17.21	347.20	43	206.3	129	yes
EK_39	MR/DDH	801914.25	9544206.86	42.46	636.00	13	244.65	105.7	yes
EK_40	MR/DDH	799497.66	9541413.90	44.69	383.60	17	252.64	90.61	yes
EK_41	MR/DDH	803046.56	9540983.55	11.40	336.33	46	250.25	79.15	yes
EK_42	MR/DDH	794865.16	9545182.98	34.89	588.00	83	219.25	134.15	yes
EK_43	MR/DDH	793004.43	9545808.29	20.11	337.60	28	190.65	170.3	yes
EK_44	MR/DDH	792925.71	9547953.53	20.36	317.45	79	200.75	116.5	yes
EK_45	MR/DDH	791897.51	9546839.83	25.72	302.06	63	173.2	171.15	yes
Total	**45**				**15,949.43**	**2251**	**6,068.23**	**9,881.2**	

8.3.7 Geotechnical Drilling

The Phase 2 drilling program was also used to provide geotechnical data for mine design. This information was derived from detailed geotechnical logging of drillhole core and the collection of samples for laboratory testing.

8.3.8 Hydrogeological Drilling

The Phase 2 program also included the completion of hydrogeological drilling and test work under the supervision of SRK. The program included a set of wide-spaced monitoring wells and two sets of close-spaced stratigraphic test wells (Table 8.3.8.1).

Table 8.3.8.1: Table of Hydrological Drillholes Completed During 2011and 2012

Drillhole ID	Drillhole Type	East	North	RL	Max Depth	Rotary Depth
Hydro 10.3	RC	798205.3	9548872	27.34	233	233
Hydro-005.2a	RC	801541.9	9543997	37.97	240	240
Hydro-005.2b	RCD	801544.5	9544008	37.32	241	241
Hydro-005.3	RC	801536.4	9544004	37.68	229	229
Hydro-005.4	RC	801641.3	9544049	31.93	232	232
Hydro-005.5	RCD	801554.7	9544009	37	228	228
Hydro-005B	RCD	801542.2	9544022	36.515	254.75	254.75
Hydro-010.4a	RC	798258.5	9548873	29.03	239	239
Hydro-010.5	RCD	798203.6	9548859	27.52	238.55	238.55
Hydro-010B	RCD	798202.7	9548844	28.07	250.5	250.5
Hydro_001	RC	802505.8	9542152	20.975	62	62
Hydro_002	RC	799376.4	9541162	35.8667	62	62
Hydro_003	RC	799729.3	9543399	41.806	164	164
Hydro_004	RC	800313.9	9548079	54.225	152	152
Hydro_005	RC	801550.1	9544000	38.073	152	152
Hydro_006	RC	796289.9	9544590	39.4763	62	62
Hydro_007	RC	794355.5	9546000	20.85	62	62
Hydro_008	RC	793986	9549013	15.56	62	62
Hydro_009	RC	796281.4	9550738	18.11	62	62
Hydro_010	RC	798200.4	9548861	27.479	141	141
Hydro_010_W2	RC	798190.7	9548851	28.1464	245	245
Total	**17**					**3,611.8**

8.3.9 Drilling Contractor

ELM engaged two drilling contractors to complete its Phase 1 and Phase 2a and 2b drilling programs.

Foraco S.A.S (Forarco) of France was contracted to complete the Phase 1 drilling program utilizing large-diameter oil-field drilling equipment that was capable of drilling to depths beyond that of the evaporite sequence (greater than 600 m). A photograph of the Foraco's drilling equipment is shown in Figure 8-4.

Meridian Drilling Ltd, registered in the UK, was contracted to do the Phase 2 resource and geotechnical drilling. The drilling machines that were utilized are as follows:

- Super-rock S5000: Rotary Drilling Machine capable of drilling 12.25” or lesser diameter drillholes to depths of approximately 600 m;
- GEMSA: Combination rotary/coring rig capable of drilling 12.25” or lesser diameter drillholes to depths of approximately 600 m; and
- Coretech: Coring rig, capable of drilling PQ and smaller size core to depths of approximately 400 m.

During both the Phase 1 and 2 programs the drilling contractors had three or four drill rigs on site. Although there were some differences with the drilling equipment employed the drilling methods were consistent with industry standards and similar for each program (Table 8.8.1).

All drillholes were commenced with the objective of retrieving PQ sized cores from the salt sequence. This was realised on almost all drillholes. In-hole drilling difficulties made it necessary to reduce the drill rod diameter in placed to be able to complete some drillholes.

8.3.10 Drilling Methods

All drillholes were drilled with large-diameter (approximately 12” that reduced to 8.5”) mud-rotary bits from the surface to the anhydrite horizon and then diamond cored to the bottom of drillhole. Standard drilling procedures were developed and were followed for each drillhole (Figure 8-5). A summary follows:

- Mud rotary drilling was completed from surface to just above or within gypsum/anhydrite horizon;
- The mud rotary section was drilled with environmentally friendly bentonite, soda ash, and sodium bicarbonate mud mixtures;
- 12.5” casing was employed from surface to the upper dolomitic horizon. This was then grouted and allowed 48 hours to set;
- 8.5” casing was employed through the dolomite into the underlying Anhydrite Sequence and this was again grouted then allowed to set for 48 hours before commencing diamond coring;
- Occasionally a cementation survey was completed using down hole geophysics to check that the cement had set properly;
- Diamond tails were commenced into the Evaporite Sequence and the underlying Cocobeach Formation. Coring was mainly completed using PQ sized bits, but HQ size bits were used occasionally;
- Diamond drilling was conducted using a tri-salt mud (a mud, which at the surface, is saturated with respect to $MgCl_2$, KCl and NaCl and has a composition of $MgCl_2$: 700 kilograms per cubic metre (kg/m^3), KCl: 70 kg/m^3, NaCl: 20 kg/m^3 and a salinity range between 250,000 and 280,000 milligrams per litre (mgpl);
- Coring continued through the mineralization to a point beneath the bischofite bed;
- Following the successful completion of the drillhole, down hole geophysical logging of the entire drillhole section was completed with recordings of natural gamma ray, calliper, neutron density, neutron porosity and sonic logs (see below); and
- The intermediate casing string was then retrieved and the drillhole was either, plugged and abandoned, or left open for water table and temperature monitoring (according to an abandonment program put in place by the company).

A total of 5,020.77 m of core was retrieved mostly from the evaporite sequence which represents approximately 37% of total metres drilled. Core recovery through the evaporite sequence was excellent.

The cored sections were described in detail and the intervals of sylvinite and carnallitite were sampled for chemical and mineralogical analyses. Intervals of rock salt or bischofite outside the mineralized intervals were generally not sampled and assayed as these did not represent the primary target.

As the dip of the potash bearing beds is minimal (i.e., they are predominantly gently dipping an undulating and laterally continuous despite local variations in the halite beds), the core length measured is assumed to be the true thickness of the seam.

8.4 Sampling Methods and Sample Quality

The sampling methods and quality control measures employed prior to preparation of samples at the analytical laboratory, along with the method or process of sample splitting, and the security measures taken to ensure the validity and integrity of samples taken are described below.

The methods and procedural protocols were developed by CSA and implemented under CSA's supervision by ELM staff. The methods and procedural protocols described below apply to all core handling related to ELM drilling campaigns.

8.4.1 Historical Sampling

There is limited information on the sampling procedures used during the drilling of the two historical drillholes completed within the Kola deposit area. The analytical results are recorded in the drillhole completion reports and are annotated on the drill logs. The records indicate that the sylvinite horizons intersected were specifically targeted for analysis although it is unclear what sampling and analytical procedures were employed.

The historical results were used as a guide for targeting mineralisation and for geological modelling but the data were not used for the mineral resource estimation.

8.4.2 ELM's Sampling

ELM cored all drillholes from approximately the base of the anhydrite to the end of drillhole. Summary details regarding cored intervals and number of samples are provided in Table 8.5.1 shows additional details on core handling and sampling.

Core retrieval

Core retrieval was completed by the drilling contractors (Foraco/Meridian) at the completion of each drilling run. Coring was done using 6 m long core barrels. After returning the core barrel to the surface the core was rubbed dry with a cloth, measured and was cut in meter sections and sealed in airtight plastic liners to prevent reaction of the deliquescent salt minerals, bischofite and carnallite with the humidity in the air.

In addition to the core hands, the Drill Supervisor, Well-Site Geologist and Project Geologist supervised the core retrieval process at the drilling rig.

Core handling

A routine set of procedures were strictly followed to ensure the stratigraphic sequence of the core was maintained and to prevent loss of material. The procedures are listed in Table 8.5.1

8.5 Sample Recovery, Quality and Potential Bias Factors

As discussed previously all diamond drilling used a tri-salt mud which theoretically is inert to most salt minerals. Observation on the some of the initial cores showed strong dissolution of bischofite intervals and some dissolution of carnallite, which should not have occurred. The mud engineer on site was able to adjust the drill mud with site specific experience and the problem was rectified.

Partial dissolution of bischofite potentially affects the representativeness of the samples, as it relatively increases the amount of carnallite and halite present in the core. In samples with little or no bischofite the preferred dissolution of some carnallite, will result in understatement of the carnallite content (KCl and MgCl).

Table 8.5.1: Sequence of Operational Steps Highlighting the Core Handling Process

Description	Documentation
The core was retrieved from the barrel in lengths approximately 3 m. The core barrel containing the core was released from the wire line and prepared for core removal. The Foraco drilling Foreman provides the geologist with the depths of both the upper contact of the core and the lower contact of the core. The length of the drill run is then calculated.	 
Upon retrieving a length of core, the core hand would mark a "B" on the stratigraphic bottom of each core length. Each run of core was laid out and wiped clean with a dry rag. All core segments were joined together for measurement of core recovery. Where possible, the core was rotated to have the steepest dip. A straight line is drawn on the top of the core, from the lower contact to the upper contact. The way-up direction is marked. The length is measured and compared with the driller's length. Any gains or losses are recorded, both on a table and on plastic core blocks. Any important info relating to the core is recorded on the table. The well-site geologist then inspects the core, its condition and completes a detailed geological, mineralogical and structural log.	 

Description	Documentation
The core is placed into core trays and placed on a rack for photographing. The core is removed from the trays and is segmented into lengths of core that fit the core trays (<= 1 m). Coloured ribbon is tied on each end of the core and the upper and lower depths are written, using permanent marker pen. The core is then wrapped in plastic liner and inserted into required lengths of tough plastic roll. One side is heat-sealed. As much air as possible is expelled from the bag, which is then heat sealed on the other end.	
The sealed core is removed from the drill site and brought to the core handing area. The core trays are laid out in stratigraphic order. The geological assistants place dots with a marker pen at 10cm intervals. A radeye (gamma scintillometer) survey of the core is done and plotted using MS Excel. The radeye survey is compared with the geological log and wire line gamma logs for identification of sampling interval and also confirmation of no core loss. The SEMM logging cable is calibrated to stretch 6cm every 100 m, therefore for 3 m, the stretch is negligible. Core trays with intervals selected for sampling are removed from the sequence and carried to the core cutting area. The core is cut using a dry saw blade in an ALMONTE manual core cutter.	 
One half of the core is selected for sampling and the corresponding half is wrapped in plastic film and then inserted into a sealed plastic sleeve as described previously.	
.The remaining and repacked half core is re-inserted into its original position in the core tray and prepared for final storage. Positions of field duplicates are marked in the core box with wooden blocks, indicating sample number and depth intervals. The processed core is stored in a secure, weather-proof storage warehouse.	

As the problem with partial dissolution only effected a few drillholes and the affected intervals did not represent an exploration target they were not sampled and therefore had no effect on the Mineral Resource estimate.

8.6 Rock Types Sampled and Relevant Compositions

The primary targets of the drilling program were the sylvinite horizons of the upper salt cycle. All sylvinite bearing intervals of each drillhole were sampled with one additional sample collected immediately above and below mineralisation. The sample interval was generally between 0.3 to 0.6 m within the mineralisation and up to 1.5 m long samples external to it.

The boundary between the sylvinite and rock salt is usually sharp and sub-horizontal. The geological thicknesses of the intervals were determined from the geological log in combination with the geophysical logs. As the strata and the contacts were generally horizontal the reported intersections represent true thickness of the mineralised interval.

Typical examples for the mineral composition based on XRD and SEM analysis of the US and LS compare qualitatively very well with the average composition determined from the geophysical wire line log (e.g. Figure 6-2).

8.7 Significant Results and Interpretation

As a result of the 2009 through 2012 drilling campaigns, the presence of significant potash mineralization has been clearly defined within several stratiform potash horizons hosted within a much thicker evaporite sequence. Good continuity of the potash mineralization has been found along corresponding horizons.

Potash mineralisation occurs as sylvite or carnallite, occurring as sub-horizontal, seams. Potash mineralization can be removed in places by leaching or flushing from the host horizons. Potentially economically extractable potash mineralization occurs in four seams identified to date.

In addition to the four main seams, minor potash mineralization is sometimes found in narrow seams, and comprises sylvite and carnallite. These are parallel to the main potash seams.

Selected potash intercepts in drillholes completed until the effective date are listed in Table 8.8.1. Interval lengths represent true thicknesses.

8.8 Conclusions

The drilling of the evaporite sequences of the Kola deposit has been completed using diamond coring and appropriate methods have been used to ensure the integrity and representativeness of the samples.

Although the historical drilling appears to have been completed using appropriate methods the methods cannot be verified and therefore the resultant analytical data has been appropriately excluded from resource estimation processes.

The three recent drilling programs were managed by ELM's staff following processes recommended by CSA and with periodic review by CSA. The procedures followed and the data produced is of general industry practice, and the procedures were appropriate for the deposit and style of mineralisation.

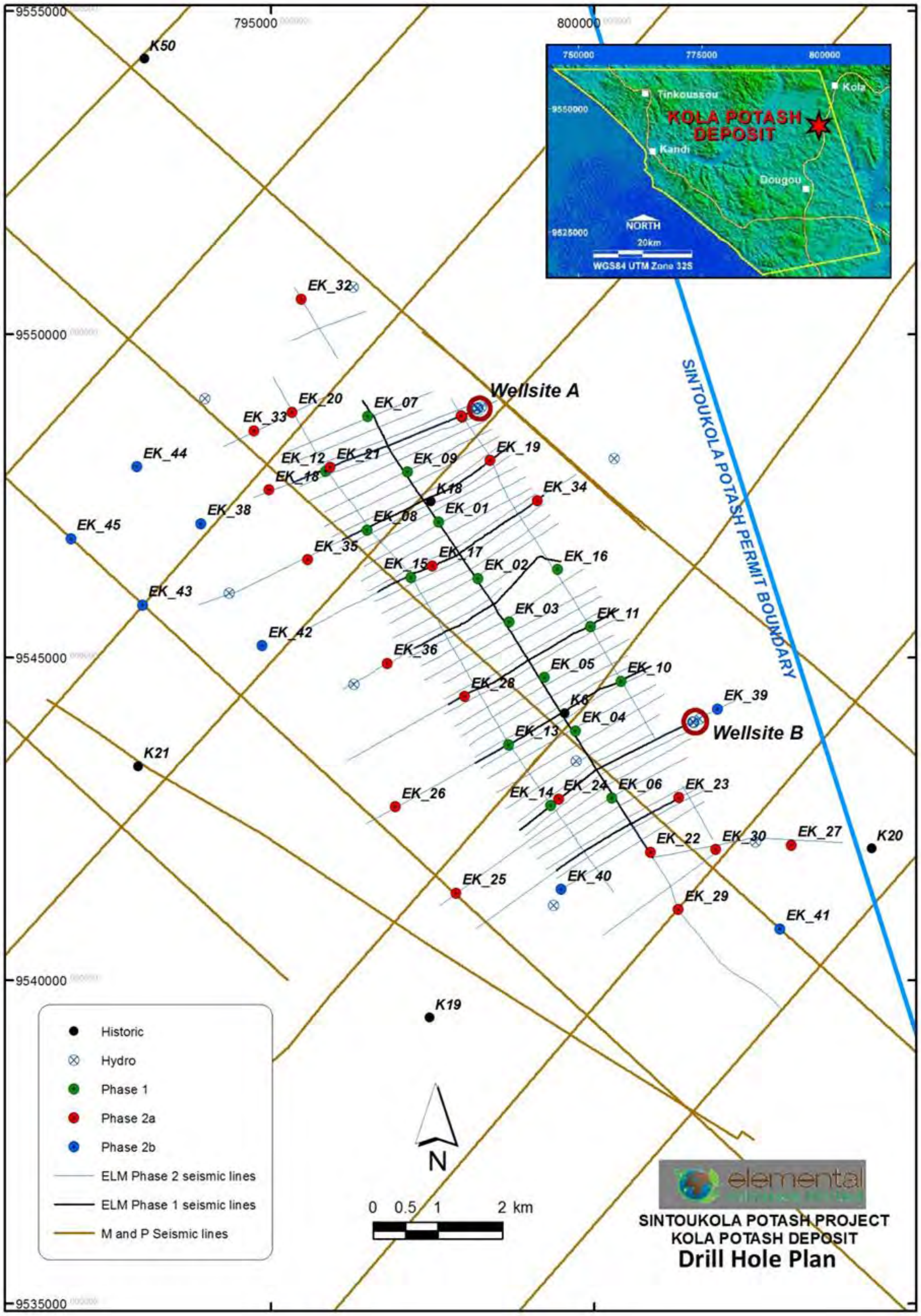
Some loss of recovery due to dissolution occurred in early drillholes in ELM's program. However as this was outside of the modelled mineralised intervals it has no effect on the resultant Mineral Resource estimates.

Table 8.8.1: Summary of Potash Intersections

Seam	Borehole	From(m)	To(m)	Thickness	Mg%	K_2O%	Insol %
HWS	EK_13	258.7	262.47	3.77	0.11	33.98	*
HWS	EK_33	214.11	217	2.89	0.02	33.59	0.11
HWS	EK_38	209.6	212.06	2.46	0.03	30.58	0.17
HWS	EK_43	222.58	225.71	3.13	0.09	37.65	0.18
HWS	EK_45	196.48	200.23	3.75	0.04	34.20	*
USS	EK_01	273.98	278.13	4.15	0.05	26.23	0.84
USS	EK_04	286.48	290.5	4.02	0.03	22.98	0.11
USS	EK_05	274.65	279.07	4.42	0.07	23.46	0.08
USS	EK_06	275	282	7	0.03	24.46	*
USS	EK_07	238.44	244.1	5.66	0.03	20.07	*
USS	EK_08	247.39	248.45	1.06	0.05	19.35	*
USS	EK_09	246.31	252.61	6.3	0.03	21.71	*
USS	EK_10	275.1	279.25	4.15	0.05	21.73	*
USS	EK_12	247.2	251.71	4.51	0.01	24.84	*
USS	EK_14	294.71	299	4.29	0.13	21.88	*
USS	EK_15	265	270	5	0.04	18.84	*
USS	EK_17	257.1	260.43	3.33	0.02	26.05	0.15
USS	EK_19	278.22	282.76	4.54	0.02	21.58	0.12
USS	EK_20	245.97	249.88	3.91	0.05	24.19	0.1
USS	EK_23	296.32	300.35	4.03	0.02	23.49	0.07
USS	EK_24	261.22	267.48	6.26	0.03	24.84	0.17
USS	EK_26	311.25	313.68	2.43	0.05	17.92	0.16
USS	EK_27	306.34	310.14	3.8	0.01	25.28	0.1
USS	EK_28	241.08	249.17	8.09	0.02	22.15	0.12
USS	EK_29	291.2	292.87	1.67	0.06	15.04	0.18
USS	EK_33	272.9	276.44	3.54	0.03	20.29	0.19
USS	EK_36	281.2	285.75	4.55	0.02	19.04	0.14
USS	EK_38	265.8	268.79	2.99	0.03	22.72	0.22
USS	EK_39	286.82	290.5	3.68	0.03	21.93	0.21
USS	EK_42	287.4	291.71	4.31	0.01	23.43	0.1
USC	EK_08	248.45	255.3	6.85	6.55	13.21	*
USC	EK_13	317.6	326.5	8.9	6.02	11.03	*
USC	EK_22	304.5	314.1	9.6	6.76	13.77	0.12
USC	EK_26	313.68	317.9	4.22	7.32	15.24	0.05
USC	EK_29	292.87	299.69	6.82	7.14	13.51	0.06
USC	EK_31	272.95	281.14	8.19	6.28	11.57	0.09
USC	EK_38	268.79	270.2	1.41	7.94	16.46	0.38
USC	EK_43	289.96	298.59	8.63	5.69	12.55	0.14
USC	EK_45	270.47	278.9	8.43	*	*	
LSS	EK_01	281.52	284.35	2.83	0.27	24.07	0.06
LSS	EK_04	293.62	294.45	0.83	1.13	23.00	0.08
LSS	EK_07	248	249.85	1.85	0.04	11.48	*
LSS	EK_08	258.31	259.67	1.36	0.57	14.09	*
LSS	EK_09	257.17	258.5	1.33	1.34	21.31	*
LSS	EK_10	282.25	288.05	5.8	0.10	19.41	*
LSS	EK_11	233.12	235.97	2.85	0.03	15.69	*
LSS	EK_12	255.82	260.67	4.85	0.04	17.93	*
LSS	EK_17	263.33	268.47	5.14	0.01	19.78	0.12
LSS	EK_19	285.93	288.29	2.36	0.03	20.98	0.32
LSS	EK_27	313.2	318.28	5.08	0.02	18.86	0.11
LSS	EK_28	254.61	262.37	7.76	0.03	19.97	0.10
LSS	EK_38	272.97	273.2	0.23	1.57	20.12	0.38
LSS	EK_39	293.49	298.63	5.14	0.05	17.93	0.17
LSS	EK_40	279.14	284.32	5.18	0.01	20.81	0.10
LSS	EK_41	267.38	269.92	2.54	0.02	14.41	0.10
LSS	EK_42	294.96	298.37	3.41	0.01	22.08	0.08
LSS	EK_44	231.65	235.5	3.85	0.03	20.24	0.17
LSC	EK_01	284.35	287.7	3.35	4.77	9.64	0.12
LSC	EK_04	294.45	301.14	6.69	5.31	10.11	0.12

Seam	Borehole	From(m)	To(m)	Thickness	Mg%	K_2O%	Insol %
LSC	EK_05	282.48	292.2	9.72	5.38	11.09	0.17
LSC	EK_06	287.15	301.2	14.05	5.96	11.16	*
LSC	EK_07	249.85	263	13.15	4.93	10.73	*
LSC	EK_08	259.67	267.57	7.9	4.84	9.10	*
LSC	EK_09	258.5	267.01	8.51	5.11	9.74	*
LSC	EK_13	329.5	338.51	9.01	5.75	10.72	*
LSC	EK_14	302.25	313.07	10.82	5.14	9.58	*
LSC	EK_15	272.91	279.7	6.79	5.04	9.87	*
LSC	EK_19	288.29	292.38	4.09	*	*	
LSC	EK_20	252.86	261.63	8.77	5.96	11.59	0.15
LSC	EK_22	318.16	328	9.84	5.82	12.19	0.35
LSC	EK_23	303.55	312.53	8.98	5.77	11.07	0.10
LSC	EK_24	273.51	292.42	18.91	5.28	10.12	0.06
LSC	EK_26	320.9	329.56	8.66	5.65	11.56	0.08
LSC	EK_29	303.2	314.1	10.9	5.32	10.16	0.07
LSC	EK_31	284	292.59	8.59	5.61	10.47	0.07
LSC	EK_33	279.2	285.19	5.99	5.83	11.42	0.10
LSC	EK_36	288.98	295.5	6.52	6.42	13.42	0.25
LSC	EK_38	273.2	280.5	7.3	6.03	12.52	0.79
LSC	EK_42	298.37	300.6	2.23	4.00	8.11	0.08
LSC	EK_43	302.02	311.61	9.59	5.21	11.46	0.13
LSC	EK_45	282.44	292.65	10.21	*	*	
FWS	EK_11	293	303.2	10.2	0.11	15.56	*
FWS	EK_16	298.39	299.14	0.75	0.03	22.27	*
FWS	EK_18	286.59	300.26	13.67	0.14	19.02	1.95
FWS	EK_32	289.87	294.82	4.95	0.11	18.10	1.30
FWS	EK_35	263.3	269.3	6	0.04	16.51	1.30
FWS	EK_39	342.08	344.92	2.84	0.33	13.09	1.46
FWS	EK_41	319.85	325.8	5.95	0.03	20.29	1.50
FWS	EK_44	296	305.25	9.25	0.04	16.90	1.13

Note: * Not yet assayed



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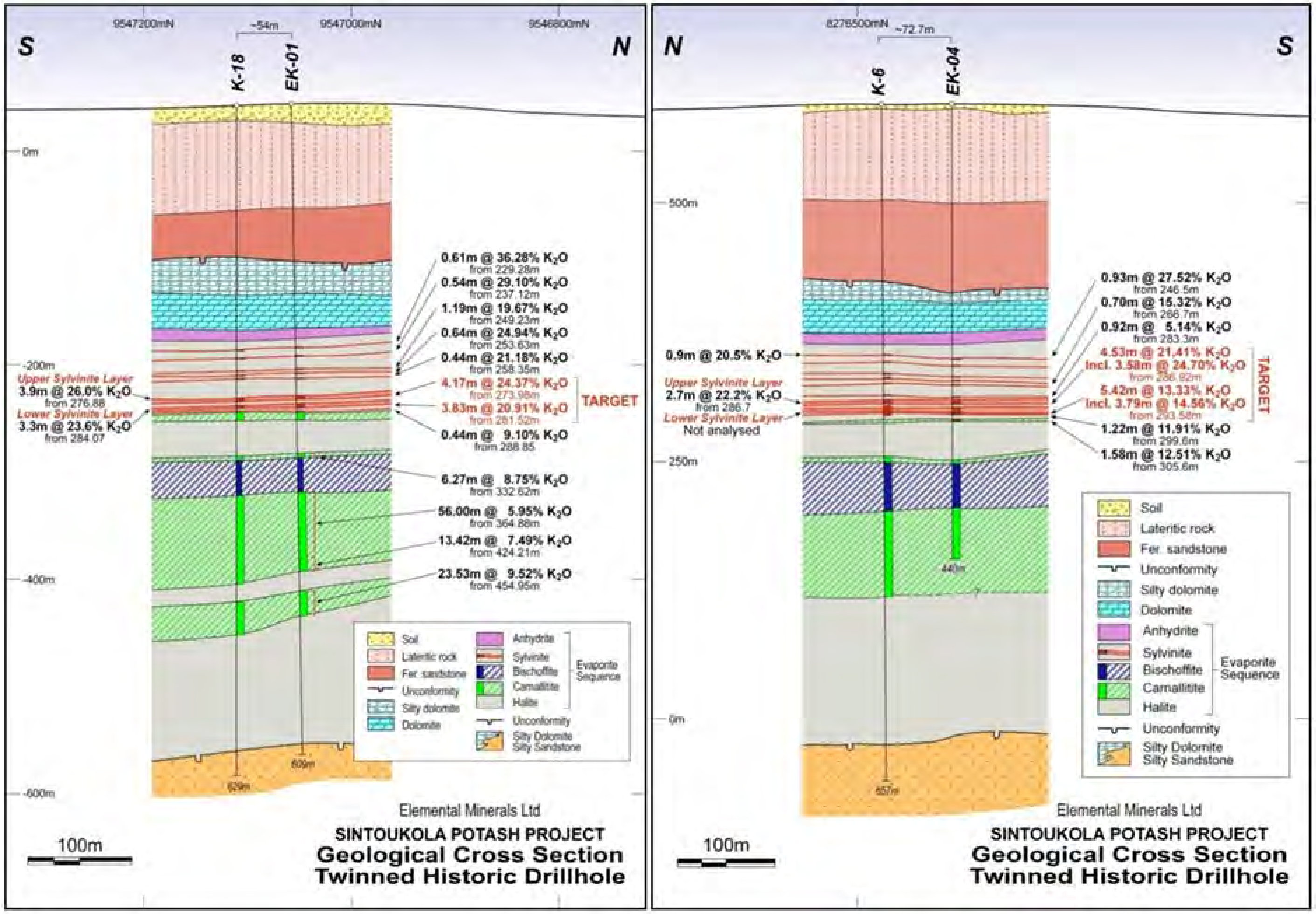
Sintoukola Potash Project, Republic of Congo

elemental minerals limited

Source: CSA, 2012

Figure 8-1

Drillhole Plan for the Kola Deposit

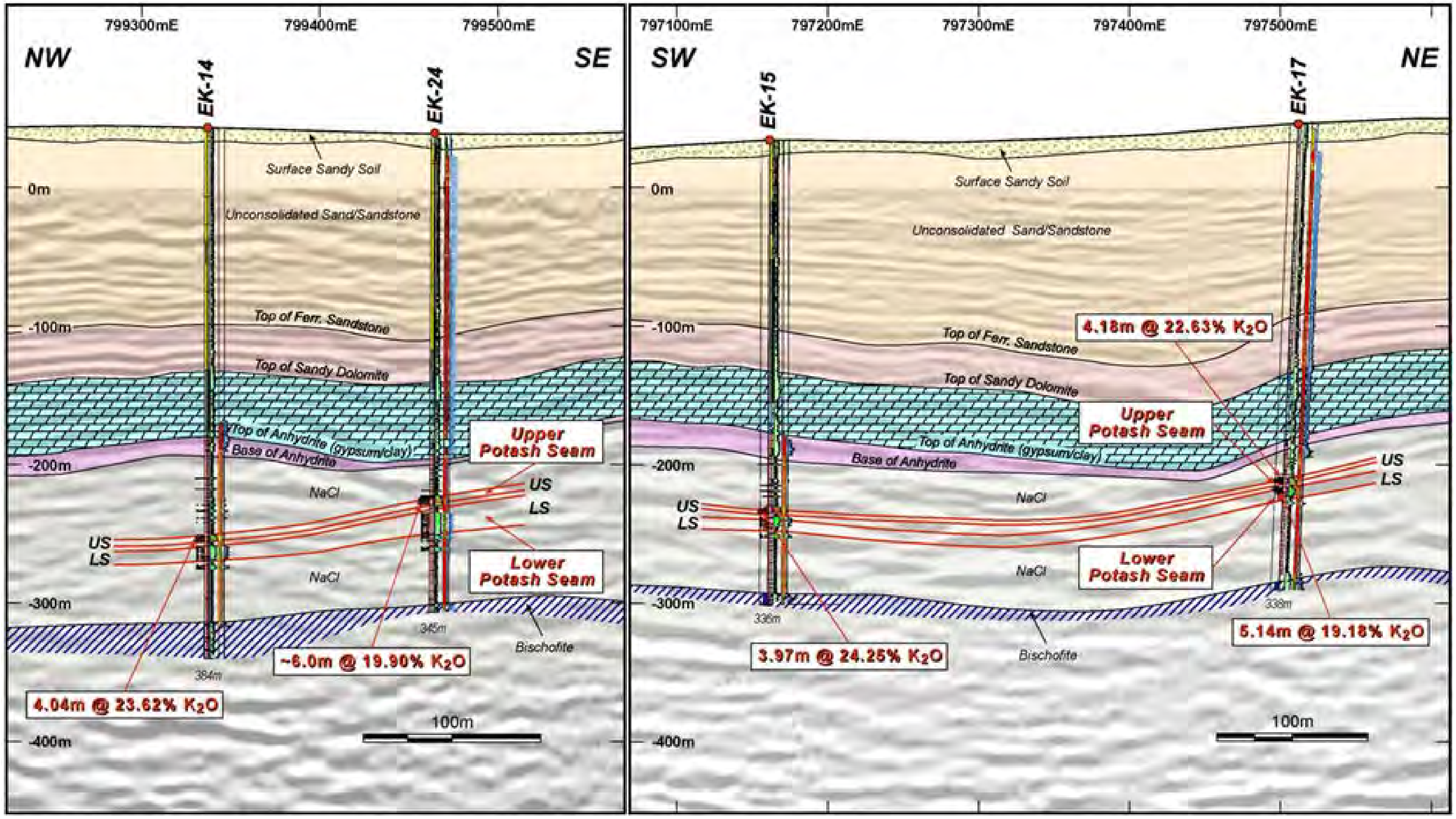


Sintoukola Potash Project
Republic of Congo

Source: CSA, 2012

Figure 8-2

Drillhole Sets from the Phase I Twin Drill Program



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Republic of Congo

Source: CSA, 2012

Figure 8-3

Phase II Close-spaced Drillholes

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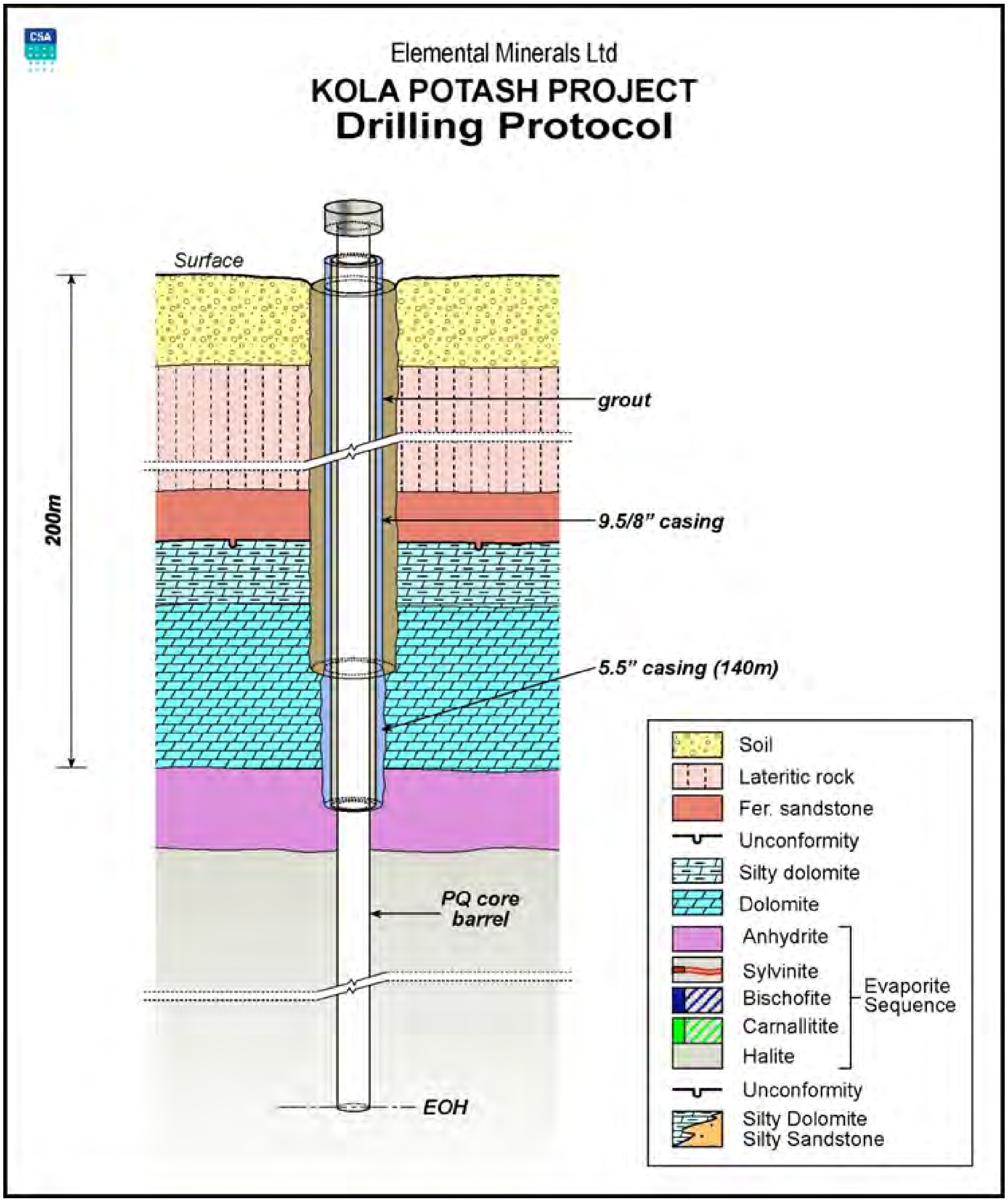
Republic of Congo

Source: CSA, 2012

Figure 8-4

Drillhole (EK_04) and Site Set Up at the Kola Project



srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	Figure 8-5 Schematic of a Standard Drillhole

9 Sample Preparation, Analysis and Security (Item 11)

A total of 2,247 primary half-core samples (1,192 in Phase 1, 683 in Phase 2a and 372 in Phase 2b) were collected and dispatched to the analytical laboratories over the duration of ELM's exploration programs. In addition, there were a total of 111 field duplicates, 263 field blanks and 98 standards for QA/QC purposes (Table 9.1). Twenty-one samples including one standard and one blank were sent to an umpire laboratory for testing.

ELM's personnel were responsible for the preparation and submission of all samples to the analytical laboratories.

Table 9.1: Summary Overview of Ordinary and QA/QC Samples Collected and Submitted to the Laboratories during the Course of ELM's Drilling Programs

Standards	No Of Samples
AMIS0031	3
BCR-113	33
BCR-114	35
CaF2	3
Fertiliser	27
KCl	9
KCl (10g)	30
KCl (Spike1000)	18
Na5_K5	15
NaCl	7
Nacl-10g	22
Quartz	8

Other QA QC	No Of Samples
Field Dup	111
Lab Check	123
Umpire	19
Blanks	263

9.1 Sample Selection

The core was transported from the drill site to core handling facility, approximately 4 km from the centre of the deposit. Upon arrival at the core yard, the core boxes were laid out in stratigraphic order on the ground. Cores were taken out of the plastic sleeves. Upon visual inspection of the core and review against the geophysical wire line logs, the sampling interval was determined.

9.2 Sample Handling in the Field

The following sample handling procedures were carried out to ensure the core was under the supervision of responsible personnel while maintaining the integrity of the core:

- The core was cut along its length for chemical and mineralogical testing. Cutting was done dry using a manual Almonte core saw. To avoid bias introduced by uneven cutting, the core was cut using a cutting cradle. This resulted in regular width of both samples. A sample was taken immediately above and below the mineralization;

- The half core to be analysed was gathered and put into a plastic sample bag. Written on the sample bag were drillhole number, top and bottom of sampled section and sample number. The bag was airtight sealed and put into another plastic bag, together with a note, with the same information, before airtight sealing of the second plastic bag; and
- The remaining core was kept as an archive sample and sealed air-tight in a plastic liner. If the remaining half core sample was taken for a field duplicate sample, a wooden marker was put in the sleeve with information about the top and bottom of the sample interval and the use of the sample at the original position of the sample taken. On the plastic liner at the top of the core, drillhole and depth interval of the core are marked. This information is copied on a note, which together with the packed core is sealed airtight in another plastic liner. The cores are stored in galvanised metal boxes containing approximately 4 m of core. On each tray the drillhole number, the depth interval of the tray and the direction of the top of the core are marked. All core material is stored for future reference in a container at the Sintoukola field camp.

9.3 Laboratories

ELM elected to engage two accredited laboratories to conduct the analytical work on the core material. These were K-UTEC Salt Technologies (K-UTEC) in Sondershausen, Germany and Genalysis Laboratories, Perth, Australia. Both laboratories have been certified in accordance with ISO/IEC 17025 (Appendix B).

9.4 Sample Transport and Security

After sealing the bags, the samples were put into a container in the secure drill camp, until enough samples were available to warrant transport to the laboratory.

The samples were packed into a wooden or aluminum box for transport to Pointe Noire, where representatives of ELM organised the air transport to the laboratory usually with the freight contractor DHL. Until this was arranged the samples were kept in a locked storage room at the ELM office. DHL used local transport agents to convey the samples from the respective airport to the laboratories.

Information accompanying the sample shipment included the client name and address, the e-mail list for distribution of assay results, type of geochemical analyses required, as well as a sample list detailing the sample numbers in each consignment.

Sample boxes were not locked or sealed during transport, because of custom controls. However, the double-sealed plastic bags the samples are packed in (and the use of uniquely numbered ties) allow recognition that samples have not been tampered with.

The geologists selecting, cutting and preparing the dispatch of the samples in the field are employees of ELM. The laboratory personnel at the various laboratories are independent and have no affiliation with ELM.

9.5 Sample Preparation and Analysis

When the samples arrived at the laboratories the laboratory sorted the samples and prepared a Shipment Receipt Report that was e-mailed to the ELM contact list to confirm arrival.

At K-UTEC the whole sample was first crushed in the double plastic transport liner with a hammer to a grain size of less than 1 cm. The whole content of the bag is then homogenised in a polyvinyl chloride (PVC) vessel by agitation. An aliquot of the sample, approximately one third, is then crushed with a hammer in a polyethylene foil to a grain size smaller than 5 mm and again homogenised. Approximately 100 g of the sample are afterwards milled by a disk-swing mill with a milling time of 120 seconds. Following pulverisation an aliquot of approximately 3 g of the sample is manually milled with an agate mortar for XRD analysis.

In the pulping procedure for ion chromatography, 10 g of the analysis fine sample powder are mixed with 990 ml water and agitated for 24 hours. The insoluble residue (being always present with salinar rocks) is separated by a 0.45 µm membrane filter, which was weighed before and after filtration to determine the proportion of the water insoluble fraction.

At Genalysis the samples were crushed to a nominal minus 2 mm. A 100 g sub-sample of the pulverised material was collected and dissolved with doubly deionised water and shaken for half an hour at a speed of 90 revolutions per minute (rpm) or until the soluble component of the sample has dissolved. Dissolution is checked by placing sample for a short time in an ultrasonic bath. The sample volume was 1,000 ml.

The soluble solution is then analysed by ICP-OES (Inductively Coupled Plasma Optical Emission Spectrometry) or ICP-MS (Inductively Coupled Plasma Mass Spectrometry) for K, Na, Ca, Mg and SO^{-4}. Chloride (Cl^-) is determined by titration with silver nitrate. Loss on drying and the insoluble component is determined gravimetrically.

The instrument detection levels for both laboratories are shown in Table 9.5.1. Routine international-standard QA/QC procedures were used by both K-UTEC and Genalysis. With each set of 40 samples the laboratory analysed two potash standards, and one quartz blank as well as a split sample replicate.

Table 9.5.1: Lower Detection Limits for Analyses

Element	Method	Detection Limit Genalysis	Detection Limit Ku-Tech	Unit
Ca	SWs/OE	0.001	0.001	%
Mg	SWs/OE	0.002	0.01	%
Na	SWs/OE	0.1	0.01	%
K	SWs/OE	0.1	0.01	%
Loss on Detection	LOD/GR	0.01	0.01	%
Insolubles	SWs/GR	0.01	0.01	%
Cl	SWs/VOL	0.2	0.0003	%
S	SWs/OE	0.01	0.01	%
SO4	/CALC	0.03	0.01	%

Upon completion of the assaying and QA/QC procedures, the geochemical results were downloaded into a CSA managed database and e-mailed to ELM. After receiving each batch of results from the laboratory CSA completed checks to ensure integrity and reliability.

9.6 QA/QC

Several types of control samples were used by ELM for quality control and assurance purposes. These included the following:

- Field or Coarse duplicates: approximately every 20th samples the second half of the core was selected as a duplicate sample. The duplicate samples were assayed by the same laboratory as the original samples, and provide information about the sub-sampling variance introduced during the splitting process and/or about the in-homogeneity of the sylvinite on the scale of the core (cm sized rock salt horizons);
- Check/Umpire samples: after differing stages of sample preparation pulp duplicates were-submitted to an external second laboratory. Check samples were used to check on the precision of the results at various stages of sample preparation;
- Blank samples: were inserted in approximately every 20th position, and submitted to same laboratory as the original samples. The blank samples were used to assess the quality of the laboratory's operational procedures; and
- Standard samples: Two certified standard reference materials were available for use by ELM at the time of the drilling program. An additional internal standard was prepared from commercially available potash fertiliser material. The expected composition was indicated on the product label but was also determined by the laboratory in Perth. The standards were inserted into the sample stream at a frequency of about one per batch of approximately 20 ordinary samples.

In addition to ELM's control samples the laboratories had their own QA/QC procedures which involved the use of internal blanks, standards and the repeating of some results as discussed above.

A further QA/QC measure relating to sampling and analysis was the comparison of down hole geophysical data (mainly gamma data) with both geological records and assay data. Results for each of these QA/QC measures are discussed below.

9.6.1 Field Duplicates

Field duplicate samples were taken from the cores of almost every drillhole. No field duplicate samples were taken from the cores of EK_03, 21, 25, 31 and 34 because these were abandoned prior to intersecting the mineralisation and were not sampled. A total of 111 field duplicate samples were submitted to the laboratories during the course of both programs.

A geostatistical assessment of the results of original and duplicate analyses was conducted via correlation analysis. The results indicate a very high level of correlation, analytical and geological integrity for potassium shown in Figure 9-1. The same level of good correlation of the potassium is reflected in the graphs magnesium and sodium, data which are not shown here.

The data set includes field duplicates analyzed at both laboratories and highlights the robustness of the procedures put in place for sampling and the integrity of the sampled material and the procedures in place at the laboratory.

9.6.2 Check/Umpire Samples

Two types of check samples (CS) were undertaken. Type one, CS_1 relates to a set of 21 samples of coarse reject material that was sent as one batches from K-UTEC to Genalysis laboratories. Type two, CS_2, check samples were taken from the pulp samples prepared for the ICP_OES/MS at both laboratories. A total of 123 samples were selected randomly across the entire sample set.

To analyse the results, scatter plots were made for the main ions Na^+, K^+, Mg^{2+}. An analysis of the results shows that no highly anomalous sample pairs occur for the three main ions in either sample set. This result underlines the homogeneity of the sample medium and the compatibility of analytical methods and procedures applied at both analytical laboratories.

9.6.3 Blank Samples

A total of 262 blank samples were submitted during the Phase 1, 2a and 2b drill programs with 9 samples included in the samples dispatched to K-UTEK and 253 samples included in the batches tested by Genalysis. A review of the results indicated that no significant contamination was detected during assaying at either laboratory (Figure 9-2).

9.6.4 Standard Samples

A total of 71 standard samples were submitted during the Phase 1, 2a and 2b programs with 7 samples included in the samples dispatched to K-UTEK and 64 samples included in the batches tested by Genalysis.

Q-Q plots of the results of the standards by batch were prepared and reviewed for potential problems. The results of the certified reference materials (BCR-113 and BCR-114) suggest that the laboratory results are within acceptable limits of accuracy (Figure 9-3). The outlier observed in the BCR-114 plot represents a mislabeled BCR-113 sample. The error has been rectified in the database. The results of the “fertilizer standard” (not shown here) showed a wider scatter about the indicated average composition and this is thought to be due to insufficient homogenization of the original sample material and limited number of analyses to determine what the expected, repeatable result should be.

9.6.5 Laboratory QA/QC

The laboratory QA/QC measures comprised blanks, lab internal standards (5 different types) and repeat samples. The data was collated and the results were reviewed and these were all found to be within acceptable limits.

9.6.6 Geology to Gamma Correlation

A qualitative correlation study between the geological-mineralogical logs and the associated gamma curves for each drillhole of the 2010 and 2011 drilling program has been completed. The study was completed manually for each drillhole data set. The objective of this study was to cross-reference the two data sets as a data verification procedure. In this study, the gamma profile was plotted against a detailed geological log. The depths recorded from the drilling were taken as true depths and the gamma wireline curve were adjusted to these intervals using a best-fit approach. These adjustments were completed on the sylvinite zones only over the sampled intervals for each drillhole. The qualitative correlation between the gamma curve geometry and the geology is very strong.

9.6.7 Assay to Gamma Correlation

A qualitative comparative assessment between the assay-derived % K_2O data and the associated gamma curves (API) for some of ELM's drilling has been commenced. Initial work has shown good qualitative correlation between the assay and gamma data. This study is ongoing and is planned to be completed for all drillholes. It is expected that this work will lead to a project-specific correlation factor to assist with interpretation of mineralisation. There are no plans to calculate K_2O based on gamma using a derived algorithm.

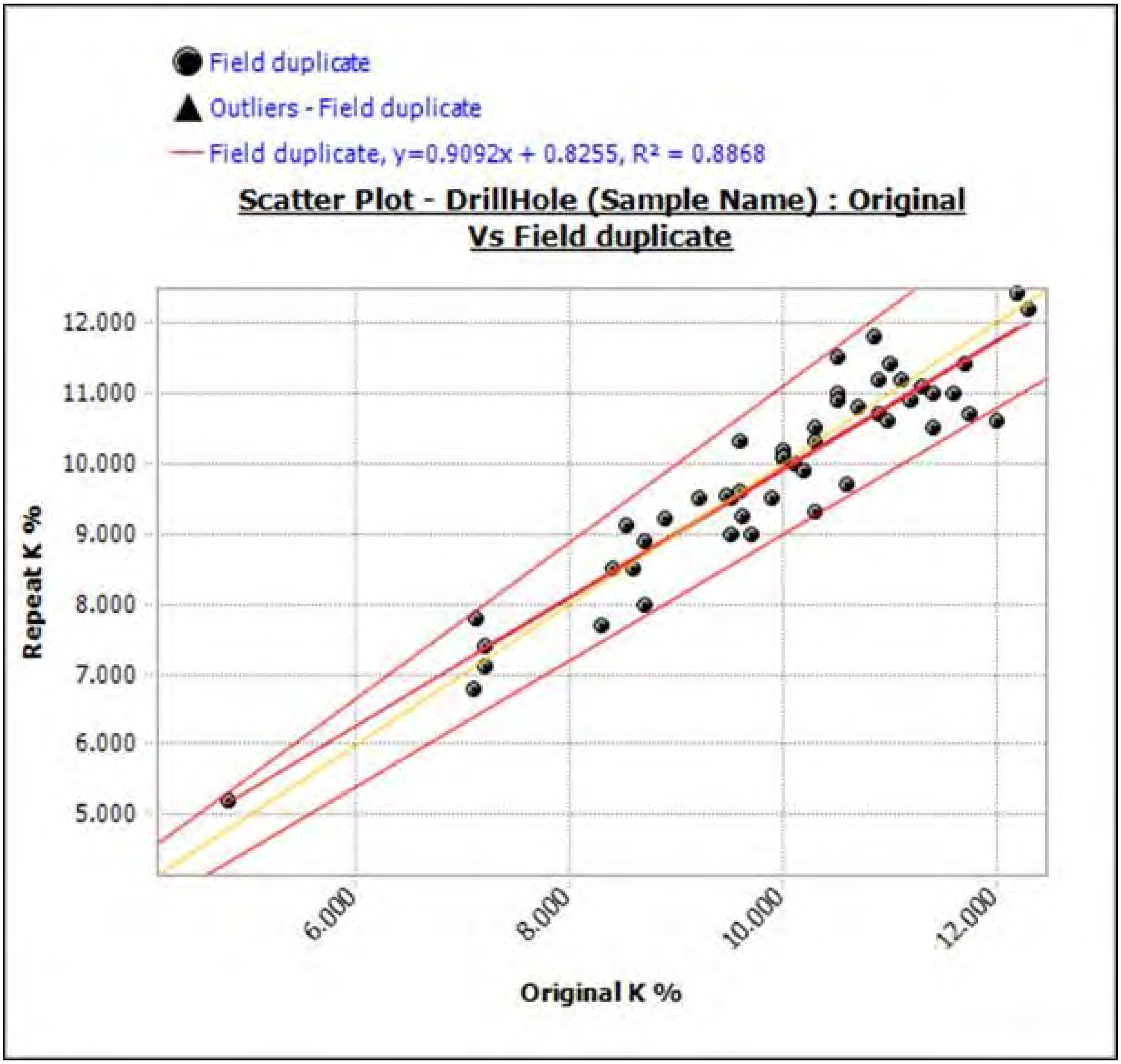
9.7 Mineralogical Analysis (XRD and SEM)

Twelve samples were mineralogically analysed at the Microanalysis, Perth, Australia, using XRD and energy dispersive spectroscopy (EDS). EDS is a semi-quantitative analytical technique which detects the characteristic X-rays emitted from samples whilst being examined by electron beams in instruments such as SEM's and TEM's. The analysis is qualitative only as there has been no calibration to determine the relative amounts of each salt mineral in the sample.

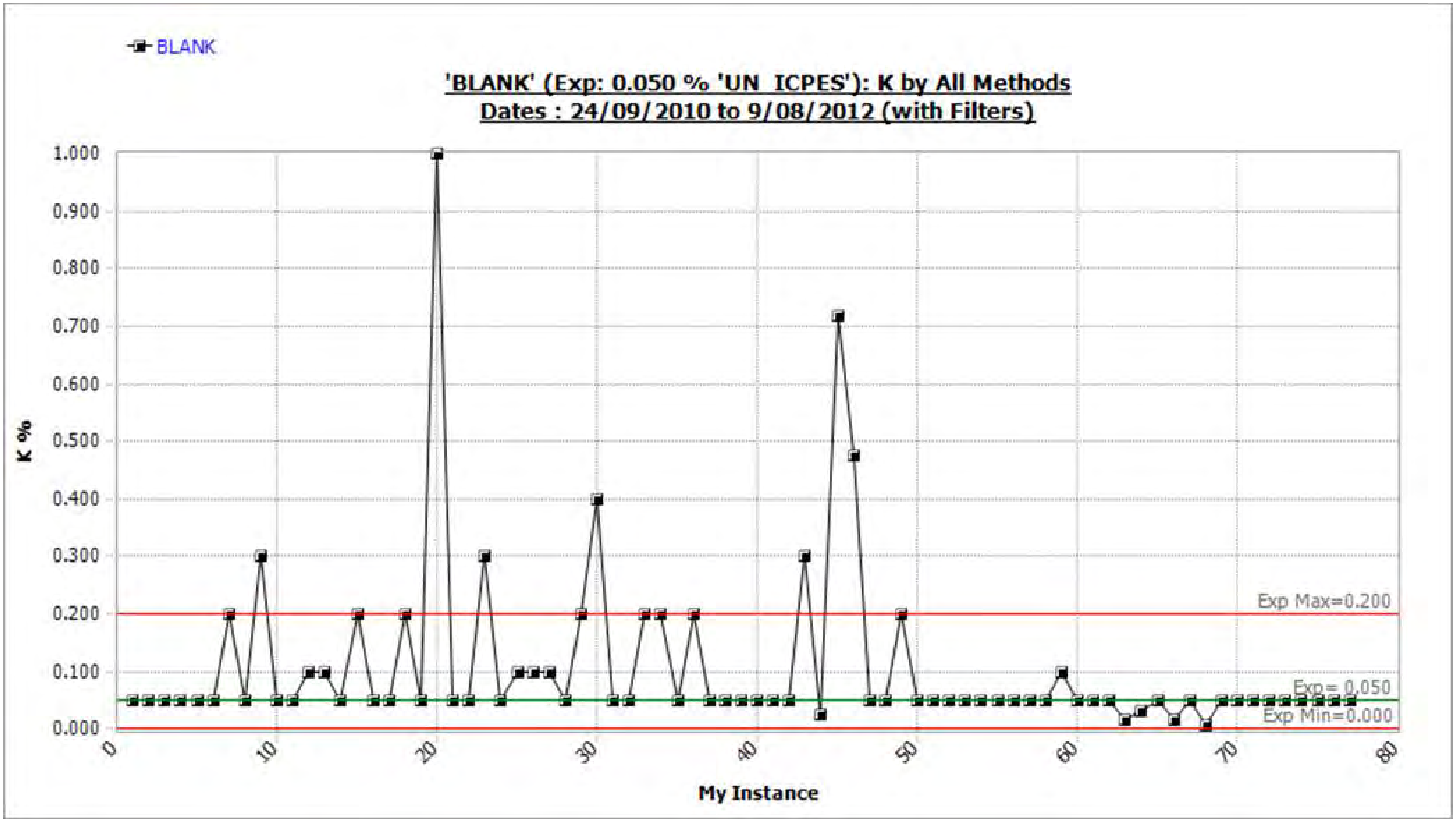
9.8 Conclusions

All sample preparation and analysis were carried out at laboratories that have been certified in accordance with ISO/IEC 17025. No aspect of laboratory sample preparation or analysis was conducted by an employee, officer, director or associate of ELM.

ELM has used a combination of duplicates, checks, blanks and standards to ensure suitable quality control of their assay testing. The procedures and QA/QC management are consistent with industry good practice and are deemed fit for purpose. Results of recent sampling have not identified any issues which materially affect the accuracy, reliability or representativeness of the results. CSA is satisfied that the procedures followed in sample preparation, security and analysis were adequate, and the data produced is suitable for Mineral Resource estimation.



srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	Figure 9-1 QAQC Graphs of Original Sample vs. Field Duplicate for K



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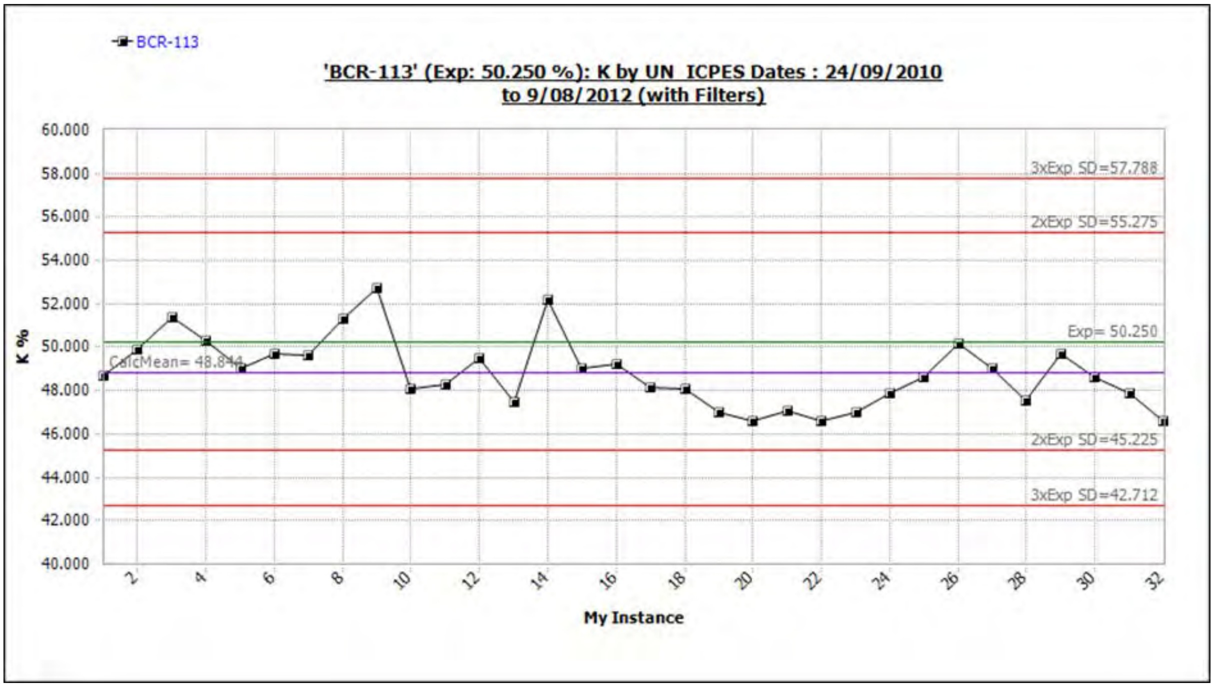
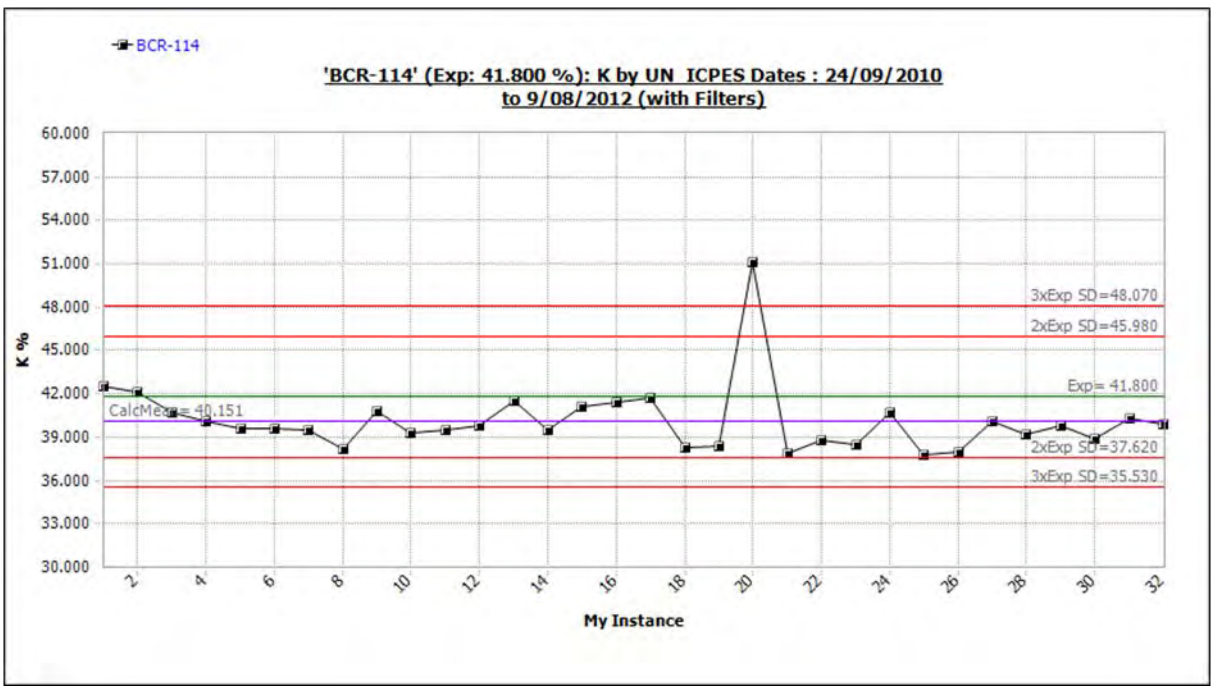
Republic of Congo

Source: ELM, 2012

Figure 9-2

Control Blanks From the Main Sample Stream Analysed for K





srk consulting	**Sintoukola Potash Project, Republic of Congo** elemental minerals limited **Source: CSA, 2012**	**Figure 9-3** **Standard Results for K (for BRC-113 and 114)**

10 Data Verification (Item 12)

ELM employs a number of quality assurance and control processes during drilling, sampling and data management to ensure the veracity of the data. These include:

- Twin diamond drillholes;
- Down hole geophysical surveying;
- Sample dispatch procedures;
- Testing of duplicates, checks, blanks and standards;
- Data transfer from paper logs to digital form by geologists; and
- Data validation procedures.

The data verification procedures are adequately documented and have been reviewed by CSA and as a result the data is considered reliable. The quality control protocols implemented at the project are considered to represent good industry practice.

10.1 Data Capture and Validation

The field data and geological logging was completed using paper logs. Data entry of the collar, survey, geological and sample information was completed by site personnel using MS Excel 2010 format data templates. All data entry was systematically validated against the paper copy and any corrections were made by the ELM's Exploration Manager.

The field data was sent via email from site in MS Excel 2010 format to CSA for importing into ELM's master database. The field data included collar tables, geology interval information, sample registers (including QA/QC samples), and related metadata tables.

Down hole geophysical data was obtained directly from SEMM Logging contractors. The data included raw text files from the down hole probe comprising of gamma; density; neutron attraction; resistivity and calliper measurements. In addition SEMM Logging provided drillhole imaging logs also comprising of dates, time, drillhole diameter, casing diameter, GPS reading, depth, and the gamma analytical data.

Assay data was loaded into the database from the original Genalysis and K-UTEC laboratory files.

The master database is located at CSA's Perth office and utilises DataShed software (with a SQL server backend). As the data was loaded into the database it was filtered through validation triggers and constraints highlighting any invalid data. This data was verified and corrected with the field geologist before proceeding with data loading. A further stage of validation was performed in Micromine (both visually and statistically) by CSA. Any problems were highlighted and verified before the data was corrected in the database.

10.2 Independent Verification

The CSA QP responsible for geology and exploration is able to provide independent verification of the ELM 2010 through 2012 exploration programs and associated data as CSA had a supervisory role in the project since commencement.

The CSA QP responsible for geology and exploration has completed four site visits to the project and has inspected drillhole sites, drilling, sampling, data capture procedures and quality control measures to ensure the quality and reliability of the exploration data.

10.3 Conclusions

Based on the data verification undertaken, the drilling, sampling and analytical standards are considered at or above industry standard and in CSA's opinion are suitable for use in Mineral Resource estimation.

11 Mineral Processing and Metallurgical Testing (Item 13)

ELM engaged Saskatchewan Research Council (SRC), Saskatoon, Canada to undertake a metallurgical test program on material from the Sintoukola Project. A representative sample of approximately 100 kg of ¼ core material was sourced from sylvinite mineralisation of the US. The following is a summary of the work completed and results obtained.

Mineralogical analysis showed the sample was composed of 38% sylvite (24.2% K_2O). Insoluble content was less than 1%, and consisted of anhydrite. The sample received was categorized as coarsely intergrown sylvinite.

The metallurgical testwork program described was developed and performed under the supervision of AMEC who are satisfied that the testing program and results are at a level that supports a FS level of evaluation and design.

The results of completed metallurgical testwork, along with process design, are presented in Volume VI (AMEC, 2012).

11.1 Testing and Procedures

11.1.1 Head Grade Determination

The core samples were photographed as received at the SRC lab and catalogued. All of the core samples were then mixed together and crushed to -6.0 mm. A representative sample was cut and split into two. One of the samples was submitted for whole rock assay using ICP-MS and the other was submitted for mineralogical analysis. XRD was used to determine the mineralogical composition of the composite sample.

11.1.2 Liberation Size Determination

Liberation size was determined using heavy liquid separation. A sample was cut from the composite and classified into the following 7 size fractions: -6.00 mm to +3.36 mm, -3.36 mm to +2.83 mm, -2.83 to +2.00 mm, -2.00 to +1.68 mm, -1.68 mm to +1.00 mm, and -1.00 mm to +0.84 mm. The -0.84 mm fraction was removed. Each size fraction was then separated in six specific gravity ranges: 2.00 g/cm^3, 2.03 g/cm^3, 2.06 g/cm^3, 2.09 g/cm^3, and 2.12 g/cm^3. The products in each specific gravity range were weighed and assayed. After the liberation size determination the sample was crushed to -2.8 mm prior to insoluble liberation and flotation testing.

11.1.3 Insoluble Liberation Test Using Attribution Scrubbing

A series of single stage and double attribution-scrubbing tests were performed using a Denver machine and a double impeller. A rectangular cell was used to simulate the agitation tank. The content of residual insoluble material was used to evaluate the liberation of insoluble and efficiency of de-sliming. The agitator was set at 900 rpm and the sample was slurried with brine to 60% solids.

11.1.4 Flotation

Laboratory rougher and regrind flotation tests were performed in a bench scale Denver flotation machine. This type of cell is representative of the mechanical flotation cells found in potash plants. Laboratory cleaner flotation tests were performed in a pneumatic flotation column. This type of cell is representative of the column cells found in cleaner flotation in potash plants. Results of the laboratory flotation tests were used to determine the parameters for the locked cycle flotation tests.

11.1.5 Locked Cycle Flotation

On a lab scale, locked cycle flotation testing is considered one of the better test methods to replicate plant flotation conditions. Locked cycle flotation testing allows for the inclusion of recirculating loads which are common in plant operations. Locked cycle testing included rougher, cleaner and scavenger (regrind) flotation. The rougher tails were reground to -1.00 mm prior to scavenger flotation.

11.1.6 Additional Testing

Subsequent to the original metallurgical test program, testing was conducted on a sample of US2 in May 2012. In addition, tests were conducted on material from the LSS, HWS and FWS in July 2012. However, the HWS and FWS are not included in the current mining plan so discussion in this report will only be on the LSS testwork. The objective of these test programs was to determine if the insoluble material composition and liberation size of the US2, LSS, HWS and FWS were different from the US1.

11.2 Relevant Results

11.2.1 Head Grade Determination

The ICP-MS results are shown in Table 11.2.1.1. XRD analysis showed that the composite sample is mainly composed of sylvite (KCl) and halite (NaCl). XRD also identified less than 1% insoluble material in the composite sample. The only insoluble mineral present in a sufficient quantity to identify was anhydrite ($CaSO_4$).

Table 11 2.1.1: ICP MS Results

Sample	K_2O (%)	Na_2O (%)	CaO (%)	MgO (%)	Fe_2O_3 (%)	S (ppm)	Insoluble (%)
Composite Sample	24.2	31.8	0.32	0.07	<0.01	1890	0.32
Composite Sample Repeat	24.1	31.7	0.32	0.07	<0.01	1950	0.20

11.2.2 Liberation Size Determination

The liberation test indicated that the composite sample is coarsely intergrown sylvinite with a liberation size of approximately 2.00 mm. Results of the liberation testing are shown in Figure 11-1 KCl liberation curve.

11.2.3 Insoluble Liberation Test Using Attrition Scrubbing

Due to the low insoluble content of the Sintoukola Project sample, only a short duration, single stage of attrition-scrubbing is required. Normally in potash mines it is necessary to de-slime the ore to less than 1% insoluble material prior to rougher flotation. Because the Sintoukola Project ore sample is less than 1% insolubles, it appeared that attrition-scrubbing de-sliming might not be necessary. However, the rougher flotation test performed without attrition-scrubbing de-sliming had a significantly lower recovery than the rougher flotation test after attrition-scrubbing and de-sliming. Therefore, all flotation test work included a 4 minute single stage attrition scrubbing to de-slime the sample prior to flotation.

11.2.4 Flotation

Rougher flotation KCl recovery was between 75.3% and 91.9% at a grade 92.3% to 94.2%. One stage regrind-scavenger can recover 84% to 87% of KCl in the rougher tails at a grade 66.2% to 74.3% to increase the overall KCl recovery up to 96.5%. These results are consistent with results seen from ores in Saskatchewan.

11.2.5 Locked Cycle Flotation

Overall KCl recovery ranged between 93.0% and 94.8% with a concentrate grade between 93.9% and 95.9% KCl. Grade variation in laboratory testing is not uncommon because with the small volume of material floated at each stage, consistent mass pull in every stage can be challenging. More consistent operation should be expected in an operating potash plant.

11.2.6 Additional Testing

The insoluble content and composition of the US2 sample was similar to the US1 sample. The insoluble content of the LS sample was lower than the US1 sample (0.1% versus 0.2%) The composition of the insolubles in the LS sample included anhydrite (similar to the US1) and trace amounts of quartz.

The liberation size of the US2 and LS samples was also similar, with the liberation size of the US2 sample slightly coarser than the US1 sample and the liberation size of the LSS is between that of the US1 and US2. The same crushing process used for US1 will be suitable for US2 and LS material as well.

11.3 Recovery Estimate Assumptions

Locked cycle flotation testing is considered one of the better test methods to replicate plant flotation conditions. Locked cycle flotation testing of the composite sample from the Sintoukola Project achieved flotation recoveries between 93.0% and 94.8% with KCl grade between 93.9% and 95.9%. The mass balance was developed using METSIM (metallurgical process simulation software). A combination of the SRC test results and AMEC's potash experience was used to determine the inputs into the model. This resulted in a plant recovery of 91% at a feed grade of 39.6% KCl (25.0% K_2O). The process simulation resulted in a recovery of 89.5% at the projected mining grade of 31.3% KCl (19.8% K_2O).

11.4 Sample Representativeness

Core samples from 12 of the drillholes from the original exploration program were sent to the SRC in Saskatoon for mineralogical and metallurgical testing. The composite sample consisted of ¼ cores from the 12 drillholes totaling 101 kg.

CSA in their report “SINTOUKOLA POTASH PROJECT – Metallurgical Sampling Program” dated February 12, 2012, concluded that the composite sample used for the metallurgical test program is representative of the mineralisation of US1, and that the characterization of the US1 can be extrapolated to include the mineralization of US2, with the consequential changes to average grade and composition of ore.

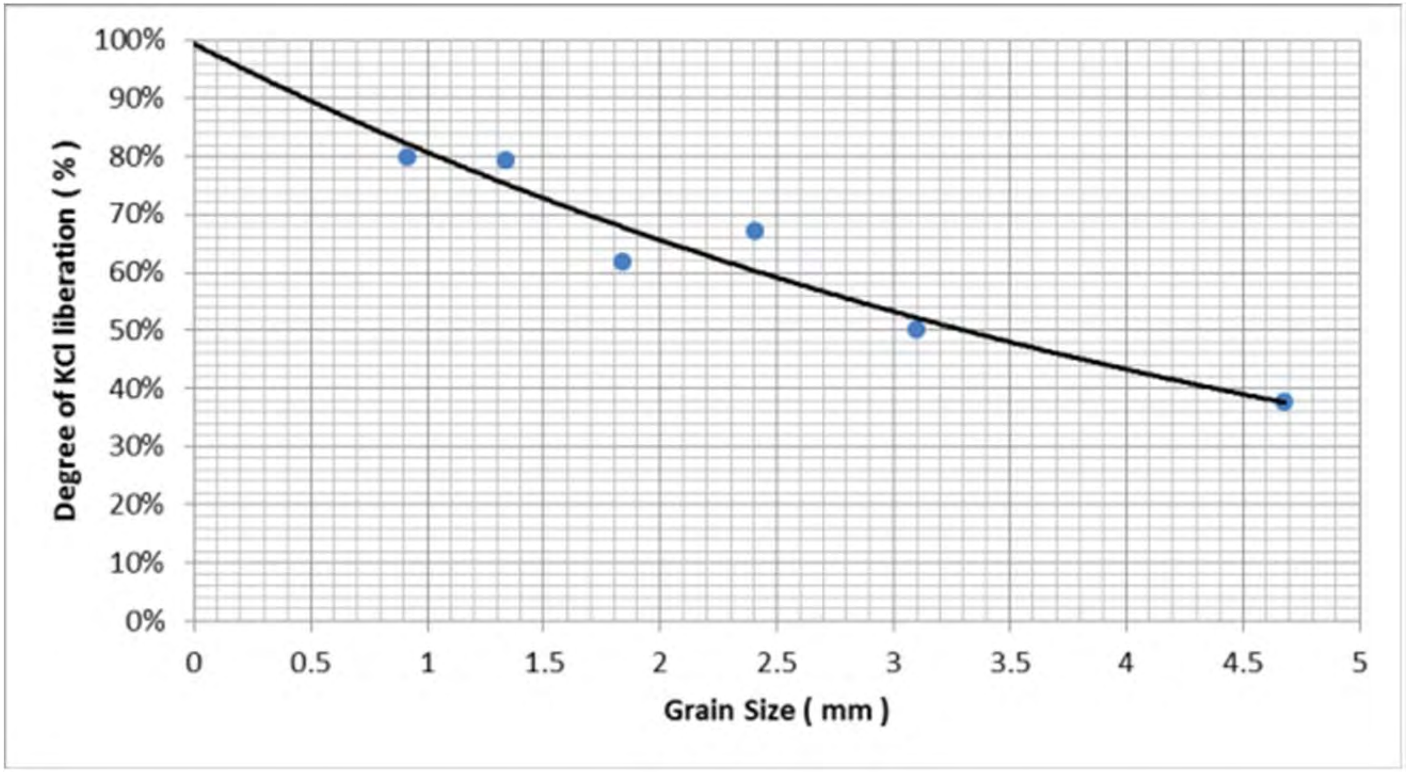
CSA in their Memo “Re: Review of Mineral Properties of the Lower Seam Sylvinite Mineralisation – Kola Potash Project” concluded that the nature and characteristics of the sylvinite mineralisation of the US and LS are indistinguishable and that the US and LS sylvinite intervals are very similar in composition, mineralogy and insoluble contents. Therefore it is expected that the flotation results for LS material would be similar to those of the US. Subsequent to the collection of the metallurgical test sample, ELM has conducted a Phase 2 and a Phase 2B exploration program. The knowledge gained from these 2 exploration programs, in addition to the previously completed exploration program has increased the understanding of the seams that make up the Sintoukola Project. This has led to the conclusion that the original metallurgical test sample was not totally comprised of material from the US1. It has been determined that the ore intersection from drillhole EK_11 is actually from the FWS (20.7 kg from the total of 101 kg) and the ore intersection from drillhole EK_13 is actually from the top seams (9.35 kg from the total of 101 kg).

The sample from EK_11 is from the sylvinite part of the FWS (FWSS). The main difference between the FWSS and the US is that the FWSS contains numerous small halite, clay and anhydrite layers which are reflected in the significantly higher insoluble content and lower grade of this seam. Due to the higher insoluble content it is expected that the flotation recovery of this material would be lower than the US1 material. Therefore, it is expected that the inclusion of this material in the composite sample would result in lower recovery than for a sample of only US1 material.

The sample from EK_13 is from the TS. The TS is a group of 6 small sylvinite seams between 0.2 and 1.0 m thick, separated by halite layers between 0.3 and 2.5 m thick. Individual sylvinite seams may be very high grade. The nature and characteristics of the sylvinite mineralization of the TS is similar to the US. In addition the insoluble content of the TS is in a similar range to the US. Therefore it is expected that the flotation results for TS material would be similar to those of the US.

11.5 Significant Factors

The potentially deleterious elements present in the Sintoukola Project are the insoluble material and carnallite. The overall insoluble content is low at less than 1% and is not expected to have a significant impact on recovery. The mine plan results in a carnallite content of 0.91% which equates to a Mg grade of 0.08%. This level of carnallite is below the concentration known to have a detrimental effect on a conventional flotation plant (0.25% Mg). Therefore carnallite is not expected to have a detrimental effect on the process plant recovery.



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Republic of Congo

Source: AMEC, 2012

Figure 11-1

KC1 Liberation Curve

12 Mineral Resource Estimate (Item 14)

Since 2009, CSA has provided geological services to the Sintoukola Project. CSA has been responsible for assisting with planning exploration, interpreting the results of exploration, managing data and estimating the Mineral Resource.

The Mineral Resource estimate described in this section of the report is the updated Mineral Resource for the Kola deposit that was announced by ELM on August 21, 2012.

12.1 Data

As described in previous chapters, CSA compiled and validated ELM's exploration data. The database cut-off date for the updated Mineral Resource estimate was July 29, 2012. Whilst the database contains both historic and recent data only ELM's drillhole data (EK_01 to EK_45) and only analytical results from these drill core were relied upon for Mineral Resource estimation.

12.1.1 Density

The in situ bulk density determinations for each potash mineralization type (sylvinite and carnallitite mineralisation) across the deposit are based on a total of 192 physical density measurements by the pycnometer method and supported by qualitative assessment of mineralogy, mineralogical composition and wire-line density recordings. The application of the pycnometer method was supported by petrographic studies, stoichiometric estimates and via comparison with physical bulk density measurements on core samples.

A total of 101 pycnometer measurements were undertaken on samples of sylvinite mineralization, 43 across samples of carnallite mineralization and the remainder of halite samples.

Based on the review of the data above and comparison with other deposits with similar mineralogy, average in situ bulk densities of 2.07 tonnes per cubic metre (t/m^3) were assigned for the HWS, USS, LSS and FWS sylvinite mineralisation and 1.70 t/m^3 for the USC and LSC carnallitite mineralisation.

12.2 Modelling

Micromine® 2011 software was used for interpretation of geological, seismic and drillhole data which then formed the basis of the revised geological model. GEMCOM Minex® 6.1.0 software was used for geostatistical analysis of seam intersections and to create seam and assay grids for estimation of Mineral Resources.

The drillhole collar data were imported into Minex to create a drillhole database according to Minex conventions (BOREID, X, Y, Z, FINALD, AZIMUTH, DIP, TYPE). Down hole survey data was imported and loaded into the Minex database as sample data according to convention (BOREID, TOSURVEY, AZIMUTH, DIP). Assay data were loaded into Minex as "QUALITY" sample data according to convention (BOREID, FROM, TOQUAL), after having established sample variables and mapping across from the assay file to match Minex variables.

12.2.1 Lithological Domains

The three geological surfaces modelled were the top of carbonate, base of anhydrite and an interburden halite reflector within the salt sequence. These surfaces and associated model codes are presented in Table 12.2.1.1.

Table 12.2.1.1: Lithological Domain Descriptors

Wireframe (.wf)	Inversion	Variable “lith”	Description
Top of carbonate	Below this	Carbonate	Contact between carbonate rocks and predominantly clastic rocks (good)
Base of anhydrite	Above this	Anhydrite	Hanging wall of the salt sequence, seismic reflector (good)
Interburden halite	Below this	IBH	Intra-salt seismic reflector (poor)

12.2.2 Mineralisation Domains

CSA digitised polygons representing stratiform mineralisation in Micromine. The polygons were built on a combination of mineralogy, mineral composition and 2D seismic reflector geometry which covered the deposit area.

A stratigraphic sequence was established in Minex with the FWS Sylvinite mineralisation (FWSS) as the lowermost potash mineralised seam, followed upwards by the Lower Seam Carnallitite (LSC) and Lower Seam Sylvinite (LSS), then the IBH, the Upper Seam Carnallitite (USC) and Sylvinite (USS) and finally the HWS Sylvinite mineralisation.

12.2.3 Seam Correlation, Gridding and Geological Cross Section Interpretation

A “SEAM PICKS” comma delimited file was compiled according to Minex protocol (BOREID, FROM, TO, SEAM NAME) and imported into the database such that the stratigraphic succession could be plotted in 3-D space. Minex suffixes for seam floor (SF), seam roof (SR) and seam thickness (ST) were appended to the seam names.

Drillhole seam validation was carried out according to Minex protocol to ensure no negative interburden thicknesses, to check the stratigraphic sequence and to define barren drillholes.

Strings representing the IBH and Anhydrite were digitized as reference surfaces in Micromine on the basis of seismic reflector patterns and drillhole intercepts. The seismic reflector strings were imported into Minex as xyz points which were then assigned the suffix “SF” in Minex geometry files, so that they could be incorporated into the seam floor modelling process. Minor adjustments were made to the seismic strings near some drillholes where needed, to enable as close a fit as possible of the IBHSF and ANHYSF with the seam picks (see Figure 12-1 and Figure 12-2).

Missing seams were set and interpolated using Minex Bore Seam Modelling functions so as to complete the stratigraphic sequence and for purposes of applying seam washouts. For the purpose of modelling seam geometries correctly, in certain drillholes the interpolated seam positions are above the logged anhydrite sequence, in accordance with the geological model in Figure 5-10. Drillholes EK_21, 30, 34 and 37 were stopped short and were used to model the anhydrite only. The Minex suffix “E” was used for estimated intersections and “I” for interpolated intersections.

Seam thickness is the underlying control for Minex gridded seam modelling, for which an east-west grid mesh dimension of 25 x 25 m was chosen to honour both the closely spaced seismic points and drillhole intersections.

Subsequent seam building involved generation of seam floor and seam thickness grids, followed by addition of seam roof or floor grids according to seam thickness. Minex grid arithmetic was used to calculate interburden between seams, after which the Strata Build function was applied to build the evaporite sequence based around the interburden halite control seam.

Where potash seams were interpreted to have been removed by dissolution, the seams were truncated by utilising the Minex "washout" function. The washout function takes into account drillhole seam information to estimate the extent of mineralization around drillholes without seam intercepts. The function requires inputs of a maximum distance (search distance surrounding all drillholes to ensure complete overlap of the model grid) and a valid distance (the maximum washout radius allowed).

The washout valid distance was derived by an iterative process and in particular by assessing the effect of increased valid distance on interpolated USS positions in the vicinity of adjacent drillholes EK_03, EK_11 and EK_16. These three barren drillholes (no USS intersections) are between 1,000 m and 1,200 m apart and it was observed that a 1,500 m valid distance resulted in a washout area deemed appropriate according to the exploration data and geological model. Minex applies the washout function to each seam independently.

Disturbance area polygons were converted into 3D shapes by applying a 'cookie cutter' methodology and these were excluded from seams within the Measured area. The anhydrite seam floor was used to cut the evaporite seams during the final Model Build process.

The gridding method used was Minex's general (or growth) method of gridding to generate geological surfaces. This growth algorithm was considered to be ideal for the Sintoukola Project geological data, as the method gives smooth surfaces which replicate the anticipated regional geological trend. The program first calculates values for the four grid intersections surrounding each data point. After the nodes around all drillholes are calculated, the drillholes are removed from further consideration. The program then makes a series of passes over the grid. At each pass it calculates values for any grid node (centre of a grid cell) that have not been assigned a value and that are adjacent to an assigned node. Each iteration enlarges the calculated region around the original drillhole locations.

12.3 Seam and Model Validation

Model validation was carried out visually, graphically and statistically to ensure that the seam model geology and grades accurately represented the drillhole data.

12.3.1 On-screen Validation

Drillhole cross sections were examined visually in both Micromine and Minex to ensure that the model honours the geological interpretation (Micromine) and seam elevations (Minex) of individual drillholes.

The seam grids were also validated on-screen in three dimensional views to confirm that the Minex grids honoured the seam picks in the drillholes.

12.4 Statistical Analysis

12.4.1 Histograms – Frequency Analysis

Histograms were compiled for all composited seam intersections. The histograms support the selection of populations based on a combination of assay and lithological characteristics in that the main sylvinitic intersections (USS and LSS) average between 19.0% and 22.1% K_2O compared with the carnallitic intersections that average between 10.7% and 13.4% K_2O (Figure 12-3 to Figure 12-5). The HWS seam averages 34.0% K_2O and the FWS 17.7% K_2O.

12.4.2 Variography

Based on the 2011 model, it was acknowledged that there would unlikely be sufficient sample pairs to construct meaningful variograms, especially as Minex treats each complete seam intersection as a single composited sample. In addition, the revised four-seam geology model, with internal subdivisions according to mineralogy, resulted in reduced data sub-sets insufficient for reliable variography. However, as part of the review process, variograms were modelled for seam thickness and K_2O% in the Upper Seam Sylvinite (USS) and this indicated ranges of approximately 2500 m along a major direction parallel to the drill grid baseline.

12.4.3 Extents and Parameters

The final seam model dimensions and parameters are listed in Table 12.4.3.1.

Table 12.4.3.1: Gridded Seam Model Parameters

Gridded Seam Model Parameters	
X origin and extent	789,032 origin / 16462 m extent
Y origin and extent	9,538,176 origin / 14,787 m extent
Grid limits	Inferred Mineral Resource boundary
Mesh size	25 m x 25 m (east-west and north-south)
Computed mesh points	473, 344
Gridding method (SF, ST)	General growth algorithm
Scan distance (SF, ST)	2, 500 m
Data Limits (ST)	To data
Washout valid distance	1,500 m (maximum, constrained by valid input seam picks around interpolated seam positions)
Washout maximum distance	15,000 m (approximate project dimensions)
Gridding method (assays)	Minex Modified Standard Inverse Distance Weighting to power 2.
Points per sector and node (assays)	Maximum 3 points per Octant sector, Minimum 3 points per node.
Anisotropy (assays)	Major axis on 328 degrees : Minor axis = 1 : 0.8
Scan distance (assays)	6,000 m
Data boundary (assays)	500 m
Data limits (assays)	Limited to actual assay data

Minex's Modified Inverse Distance Weighting to the power of 2 (IDW^2) along a major axis 328^o and with an anisotropic ratio of 1 (major axis): 0.8 (minor axis) was the selected method for all grade estimation runs. An assessment of different estimation methods was completed (including Minex's growth algorithm and Ordinary Kriging) from which it was decided that IDW^2 best reflected current interpretation of the mineralisation controls and domains.

The default Minex Modified IDW^2 setting uses an octant based search around each grid cell, in which the nearest three points that it can locate in a sector to determine a value for that sector. This has the effect of declustering data. These three points are described during the modelling process as the *maximum points per sector*. A minimum of three data points must be found by Minex within the area defined by other parameters such as search (scan) distance before it can determine a value for any given grid node (centre of a grid cell). These three data points are described as the *minimum points per node*. It should be noted that the IDW^2 method automatically assigns less weight to the furthest data points.

The grade estimation of the resource model is controlled by hard boundaries (mineral domains) in the model. This together with IDW^2 constrained the effect that distal samples have on the grade and tonnage estimate.

The grade estimation used the seam pick depths at the base and roof of the four modelled seams. The sampling tolerance was set to 95% of seam to be sampled. This was set so as to accommodate some drillholes where the geological boundary had been redefined in the present program and did not coincide exactly with the assay intervals. This is not believed to materially affect the overall grade, but allows the Minex "variable Z" block modelling process to extract complete composited seam assays.

Plan views showing the extent of modelled mineralisation for the HWS Sylvinite (HWSS), Upper Seam (USS + USC), Lower Seam (LSS + LSC) and the FWS Sylvinite (FWSS) are shown in Figure 12-6 to Figure 12-8.

12.5 Mineral Resource Classification

The classification of Mineral Resources at the Kola deposit considered the following factors:

- Level of confidence in geological understanding;
- Data quality;
- Area of influence of, and spacing between drillholes;
- Extent of 2D seismic coverage and interpretation;
- Grade and thickness variation within the seam;
- Number and quality of density measurements;
- Structural complexity and structural interpretation; and
- Geological/mineralogical information outside the immediate deposit area.

The evaluation, weight and influence given to the parameters above also took into consideration studies of other potash deposits with analogous geological features and geometries. This included a review of historic information for the Holle Potash Mine, similar underground operations that exploit potash in a very similar geological setting in Brazil (where the exploited potash salts formed in an analogous and temporal equivalent geological environment).

It was concluded that some of the most valuable guidance could be gained from the exploration, development and mining records of the Holle mine (Feuga et al., 2005). The Holle underground mining operations were similar to the conceptual mine plan anticipated for the commercial extraction of potash from the Kola deposit. Compositionally similar potash horizons (similar grade, mineralogy and geometry) are considered for mining at the Kola deposit and this is only 70 km from the historic Holle mine.

The classification boundaries for the Mineral Resources at the Kola deposit are described below. For the US and LS mineralisation the Mineral Resources were classified as:

- Measured if within a polygon that is based on the end-points of the Phase 2 seismic grid (pseudo 3D) and with drillhole density of approximately 1 km spacing;
- Indicated if within an area that was immediately outside the area classified as Measured, which includes 1 km and 3 km spaced seismic data, drillholes and is delineated by the combined intersecting perimeters of a 1 km radius around those drillhole collars. (Where no additional drilling was conducted since the previous estimate the May 2012 classification boundary was maintained); and
- Inferred if within an area within a 1 km buffer around the Indicated Mineral Resources. The area classified as Inferred Mineral Resources includes some recent drillholes, regional seismic and regional (but adjacent) historic drillholes.

Based on this, the “Measured” area for the US and LS was defined as a polygon which had a high density of 2D seismic lines and tie-lines that resulted in a grid with approximately 1 km by 0.15 km intersection points where, given the nature of the deposit, each point was treated as a drillhole position. Indicated classification was applied to an area of the US and LS which was delineated by the outer intersecting perimeter of the intersection radii of 1 km surrounding each drillhole. In this area, 1 km and 3 km-spaced seismic data is available and several widely scattered drillholes confirm the extrapolation of the geological model and potash mineralisation. The radius of 2.0 km (1.6 miles) from a cored drillhole was selected to limit the extent of the Inferred Mineral Resource. This precluded the area between drillholes EK_02, EK_03 and EK_16 from inclusion in the Indicated Mineral Resource classification. The extent of the Measured, Indicated and Inferred Mineral Resource areas are presented in Figures 12-6 to 12-8.

Given the variable preservation of the HWS and the variable development of the FWS compared to that of the US and LS the classification protocol was adjusted to reflect the level of current geological understanding of these areas. This took into consideration the limited number of intercepts by drillholes, the proximity of drillholes with similar stratigraphy and mineralisation, the density of seismic data and regional geological context. For the HWS and FWS mineralisation the Mineral Resources were classified as Inferred if within the area defined by the US or LS “Indicated” or “Measured” boundaries as described above. This boundary generally coincided with a 1km radius around those drillholes which intersected the mineralisation of the relevant seam and also was in areas with supporting seismic data.

The HWS and the FWS, in the form they are modeled and interpreted will require further work to be reclassified as Indicated and Measured Mineral Resources.

Whilst the Kola deposit area has significant remaining exploration potential, it cannot be assumed that all or any part of the Inferred Mineral Resource as presented in this report will be upgraded to an Indicated or Measured Mineral Resource as a result of additional exploration. Inferred Mineral Resources are considered too speculative geologically to be included in any economic considerations associated with a PFS or FS.

12.5.1 Disturbance Areas

Those portions of the Kola deposit that have been identified to be disturbance areas are excluded from the Mineral Resource estimates. The interpreted spatial location of the uncertainty zones is based largely on the high density seismic data which is restricted to the central part of the drilled area of the Kola deposit and affects approximately 7% of the area classified as Measured Mineral Resources.

In the areas peripheral to the Measured Mineral Resources (Indicated and Inferred Mineral Resources) the lack of high density seismic data prevents confident interpretation of the spatial location of additional areas of uncertainty. As a consequence of this the Indicated and Inferred Mineral Resources reported below represent a classified geological estimate which has been adjusted for the loss due to “potential disturbance areas” (a factor of minus 7% was applied to the volume of the Indicated and Inferred Mineral Resources reducing the estimated tonnage in a similar amount).

12.6 Mineral Resource Estimate

The Mineral Resource estimates have been reported and are based on a “four seam” model with a Cut off Grade (CoG) of 10% K_2O and with reference to Table 12.6.1 and Table 12.6.2, are presented as two scenarios:

- As a sylvinite estimation only, and
- As a combined sylvinite and carnallitite estimation

The Mineral Resource estimates have been reported and are based on a “four seam” model with a lower Cut off Grade (CoG) of 10% K_2O for combined sylvinite and carnallitite mineralisation and sylvinite only mineralisation.

The Mineral Resources are reported in accordance with the Australian Joint Ore Reserves Committee guidelines for reporting of Mineral Resources and Ore Reserves 2004, (the JORC Code) which is consistent with the CIM definition standards and hence complies with NI 43-101.

The result of this work is a substantial increase in the Mineral Resource estimates. The Mineral Resource estimate for the modelled mineralised zones at the Kola deposit are classified as Measured, Indicated and Inferred. This classification is based primarily on confidence in, and continuity of, the results from the drilling campaigns, and subsurface mapping of high density 2D seismic data. The results of the Mineral Resource estimate are presented in Table 12.6.1 and Table 12.6.2.

Table 12.6.1: Mineral Resource Estimate for Sylvinite Mineralisation only (Base case) at a 10% K_2O CoG, as announced August 21, 2012

	Measured			*Indicated*			*Inferred*		
	Tonnes (Mt)	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**	**Tonnes (Mt)**	**% K_2O**	**% KCl**
HWS							47	34.75	55.01
USS	171	22.45	35.54	159	22.04	34.89	96	21.78	34.48
LSS	93	19.22	30.42	150	19.06	30.17	107	19.14	30.30
FWS							225	17.63	27.92
Total	264	21.32	33.74	309	20.59	32.59	475	20.39	32.27

Notes:

1. *A bulk density of 2.07 g/cm^3 was applied for all sylvinite mineralisation and 1.70 g/cm^3 for carnallitite mineralisation.*
2. *Zones of geological uncertainty have been excluded.*
3. *Table entries are rounded to the second significant figure.*
4. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
5. *Insoluble contents for the resource were not estimated but insoluble content of the seam intersections are provided in Table 8.8.1.*

Table 12.6.2: Mineral Resource Estimate for combined Sylvinite and Carnallitite Mineralisation at a 10% K_2O CoG, as announced August 21, 2012 (Includes Resources in Table 12.6.1)

	Measured			Indicated			Inferred		
	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl	Tonnes (Mt)	% K_2O	% KCl
HWS							47	34.75	55.01
USS and USC	245	19.53	30.92	310	17.76	28.11	278	16.33	25.84
LSS and LSC	313	13.26	20.99	448	13.74	21.75	398	13.12	20.77
FWS							225	17.63	27.92
Total	559	16.01	25.35	758	15.38	24.35	948	16.20	25.64

Notes:

1. *A bulk density of 2.07 g/cm^3 was applied for all sylvinite mineralisation and 1.70 g/cm^3 for carnallitite mineralisation.*
2. *Zones of geological uncertainty have been excluded.*
3. *Table entries are rounded to the second significant figure.*
4. *Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, marketing, or other relevant issues.*
5. *Insoluble contents for the resource were not estimated but insoluble content of the seam intersections are provided in Table 8.8.1.*

12.7 Comparison with Previous Estimates

The current Mineral Resource estimate represents a continuation of the geological model, concepts and principal criteria used in the May 2012 Mineral Resource estimate (CSA, 2012). Since the May 2012 Mineral Resource estimate, additional drilling data has been included in the database. The new data could seamlessly be integrated into the geological model and interpretations applied in the May 2012 resource estimate. The latest model is built on the combined knowledge gained during ELM's exploration programs.

The current Mineral Resource estimate is based upon high quality analytical, stratigraphic-structural interpretation of drillhole data and high-density 2D seismic data. The geological framework as interpreted from historic drillholes, was refined following the 2011 drilling and has been reconfirmed and further differentiated with the 2011-2012 drilling and seismic information. The tighter drillhole spacing and seismic data allowed improved interpretation of the geometry of the cover sequences and the evaporite formation.

Diamond core drilling in 2010 through to 2012 has confirmed that the bulk of high-grade and potentially extractable mineralisation occurs in four seams or layers that are separated by intervals of halite (Figure 6-2 and 5-10). New drilling has now also confirmed that previously identified potash horizons are laterally continuous and can be included in estimates of Mineral Resources.

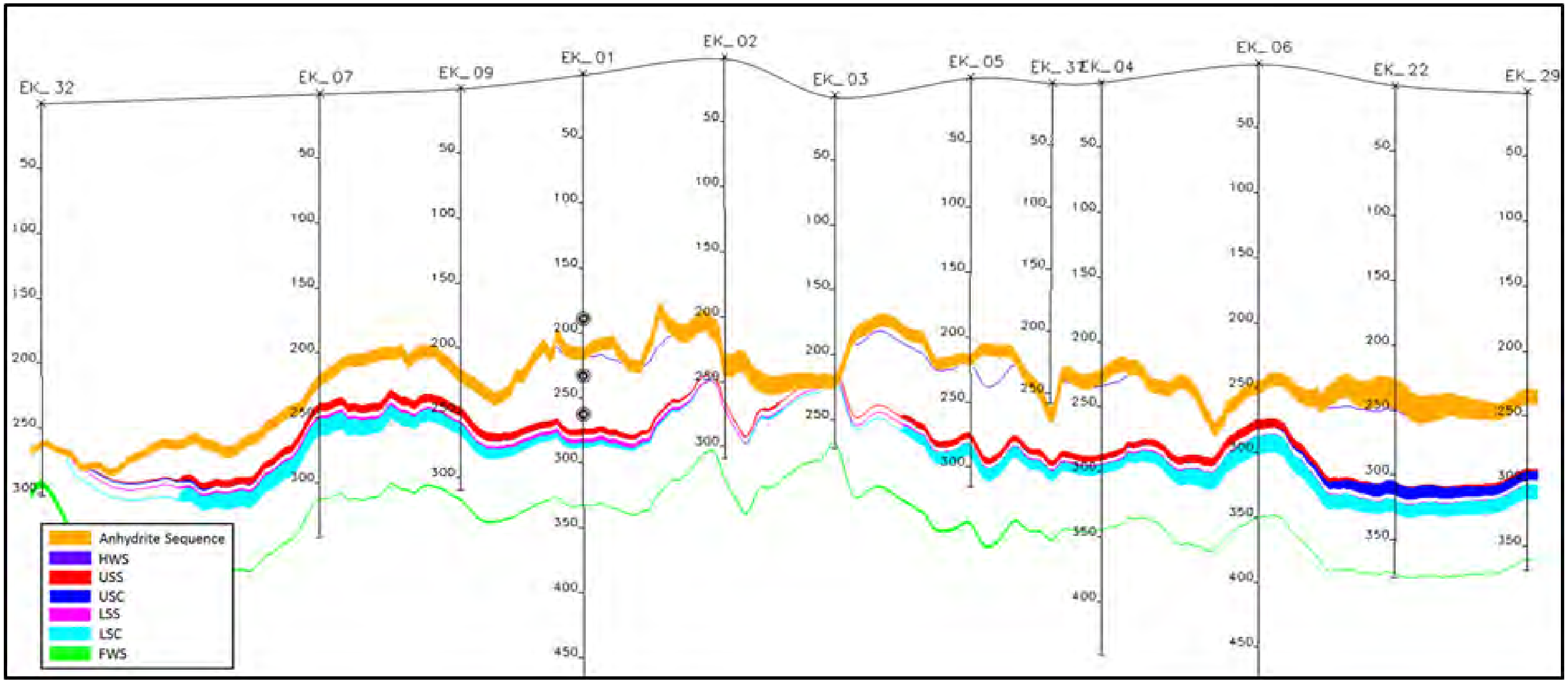
For the May 2012 model, the discovery of the FWS allowed the inclusion of another sylvinite seam. The continued exploration efforts led to the delineation of a new and additional high-grade potash seam in the upper part of Cycle VII in the form of the HWS.

The recognition of discrete mineralogical domains within each potash seam is therefore maintained from the May 2012 model and is a major change from the geological and resource model applied for the maiden Mineral Resource estimate which occurred primarily along grade constraints. The "domaining" of mineralisation along mineralogical lines is supported by predictions from the revised and improved genetic geological model and by the distribution and geometry of the potash horizons in the salt sequences in general. It has been recognised that the high-grade potash mineralisation is systematically developed depending on the distance of the potash seam from the gypsum/anhydrite contact. This suggests that sylvinite mineralisation may revert laterally into carnallitite mineralisation from which it is derived. By restricting the grade-extrapolation through mineral domains, a downward effect is exercised on volume of high-grade mineralisation.

The drilling also showed that several narrow sylvinite beds occur between the US and HWS (Triplets and Doublets), and that thick stratified layers of predominately carnallitite mineralisation occur below the FWS. These have not been modelled, but are useful marker horizons.

The high density seismic data has also led to a revision of the interpretation of the disturbance areas. Contrary to the view held after the Phase 1 program, namely that the project area is transected by several project-wide linear faults or deformation zones that are extensional in nature and show a maximum inferred displacement of approximately 40 m, the view is that the project area is affected by a dissolution front developed to a variable depth at the base of the anhydrite/gypsum horizon. This process has led, in places, to related features (disturbance areas) that may have an adverse effect on mining. The disturbance areas are compared with the Canadian “collapse” structures observed in seismic data (Nemeth et al., 2002). Due to the uncertain impact of these structures onto the distribution of mineralisation and its mineability these areas of structural complexity have been excluded from the Mineral Resource estimate. This can be re-assessed once a better understanding of the disturbance areas is gained.

The extended delineation drilling program completed by ELM at its Sintoukola Project has defined a Mineral Resource with substantial tonnage at a moderate average K_2O grade by comparison with current global potash mining provinces. The revised Mineral Resource estimate has contributed to a 38% increase in the sylvinite Mineral Resources in the Measured and Indicated categories by comparison to the May 2012 estimate.



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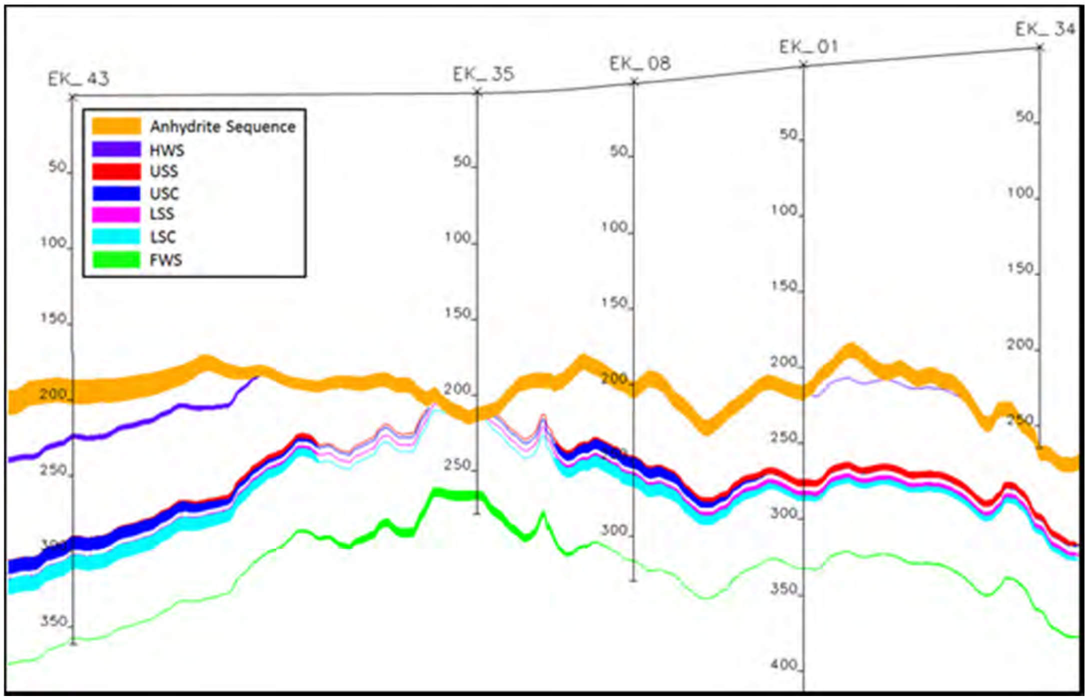
Sintoukola Potash Project
Republic of Congo

Source: CSA, 2012

Figure 12-1

Example Long-Section of the Minex Seam Model (1: 10 Vertical Exaggeration)



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Republic of Congo

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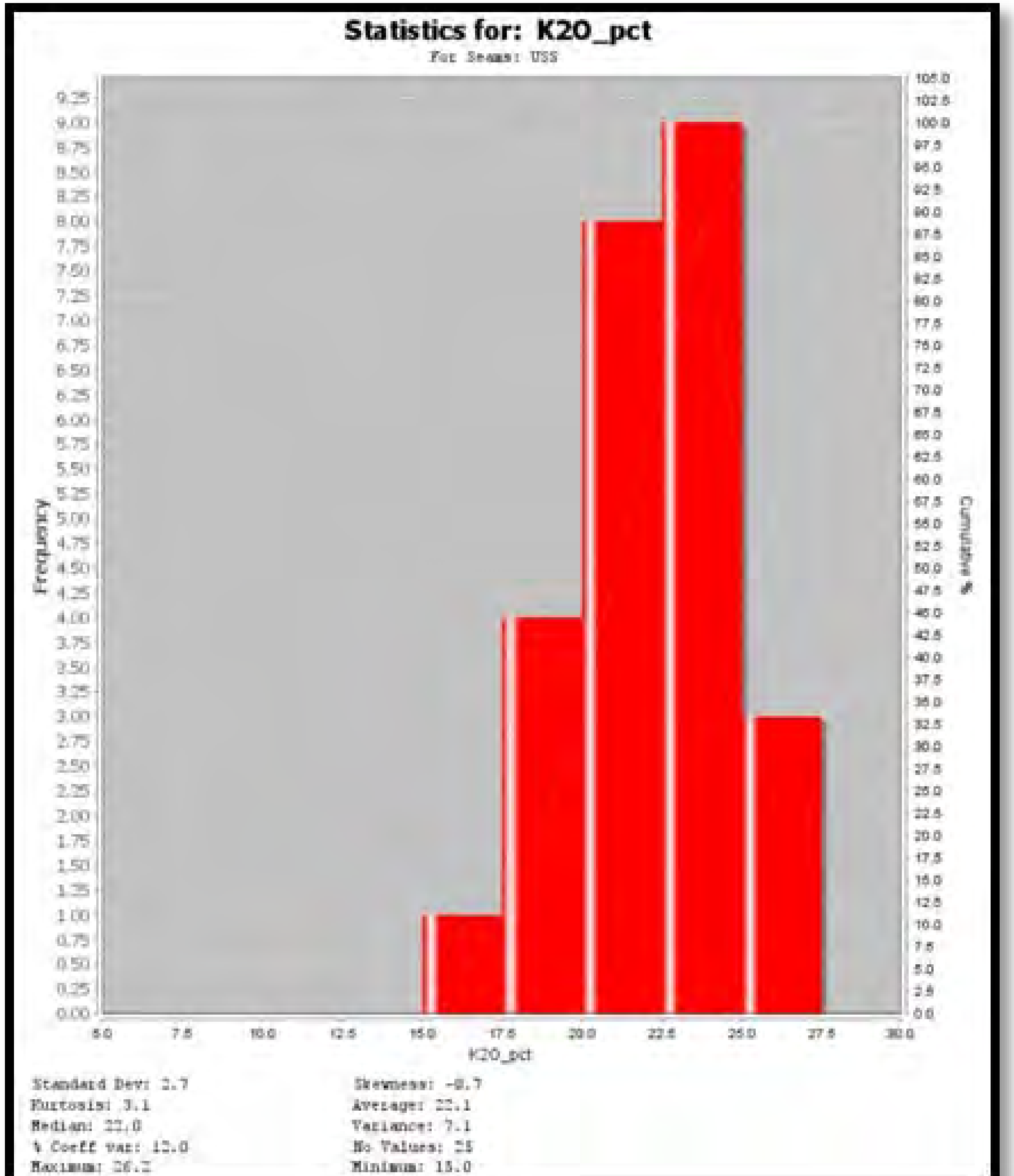
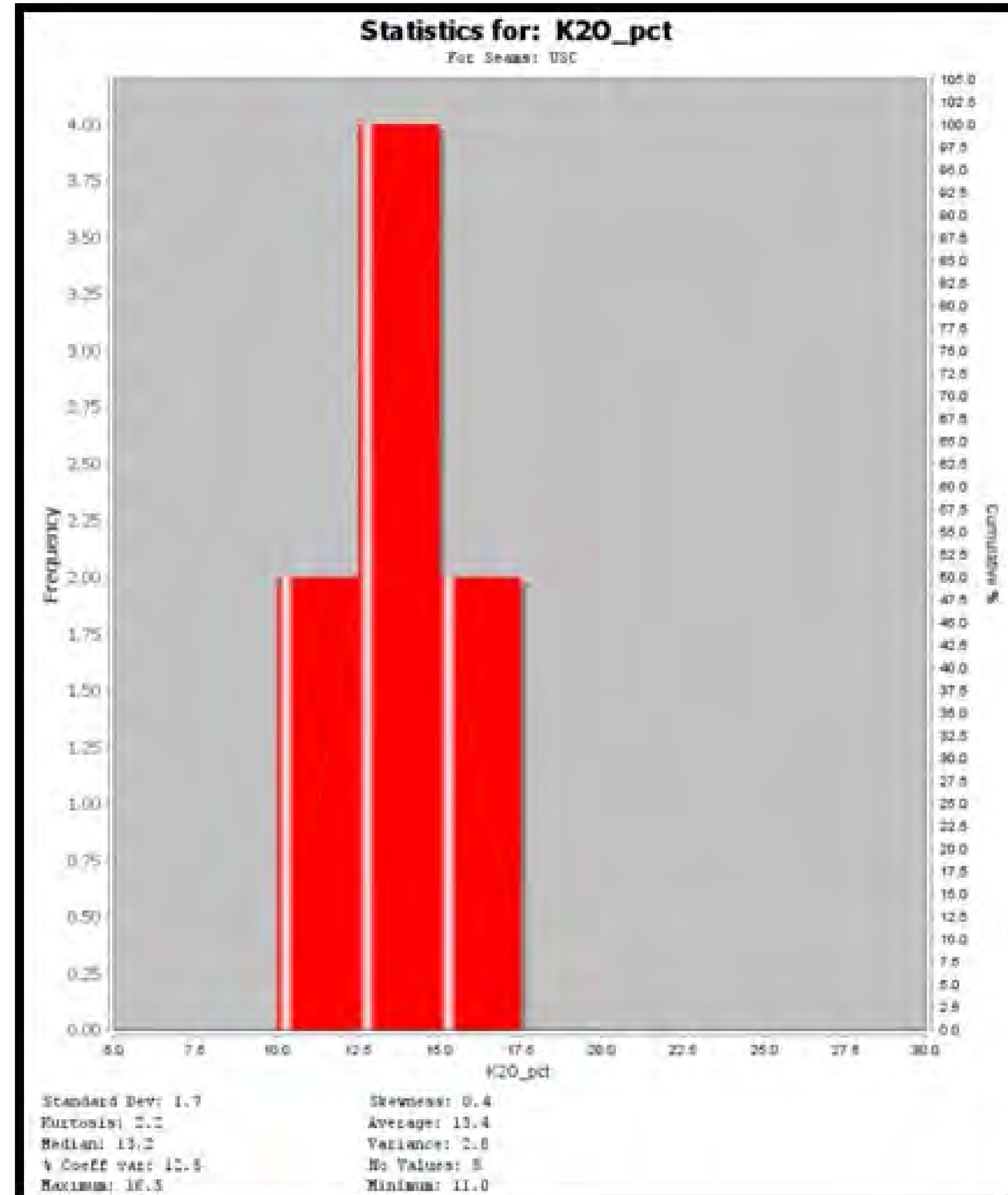
Source: CSA, 2012

Figure 12-2

Example Cross-Section of the Minex Seam Model (1: 10 Vertical Exaggeration)





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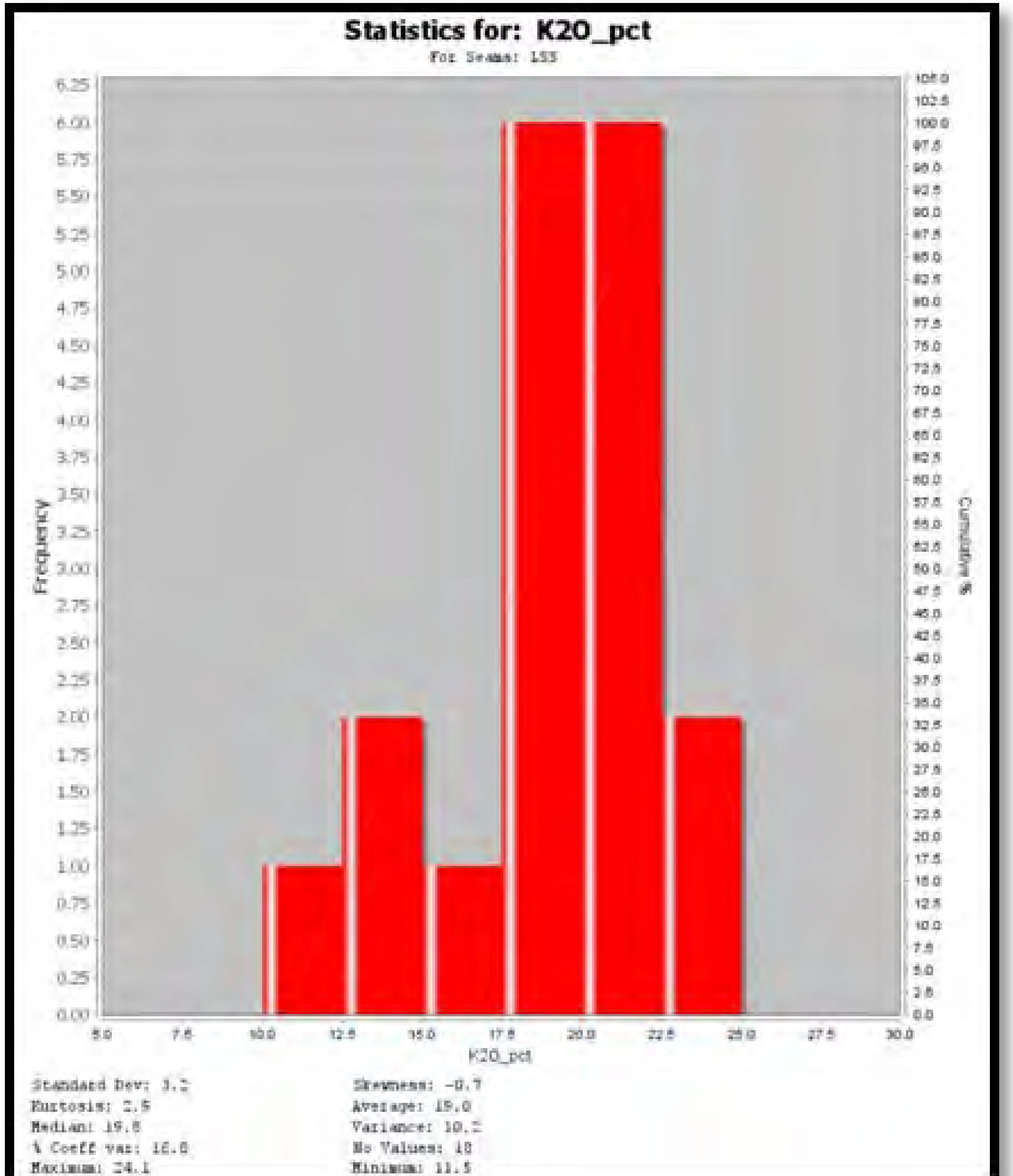
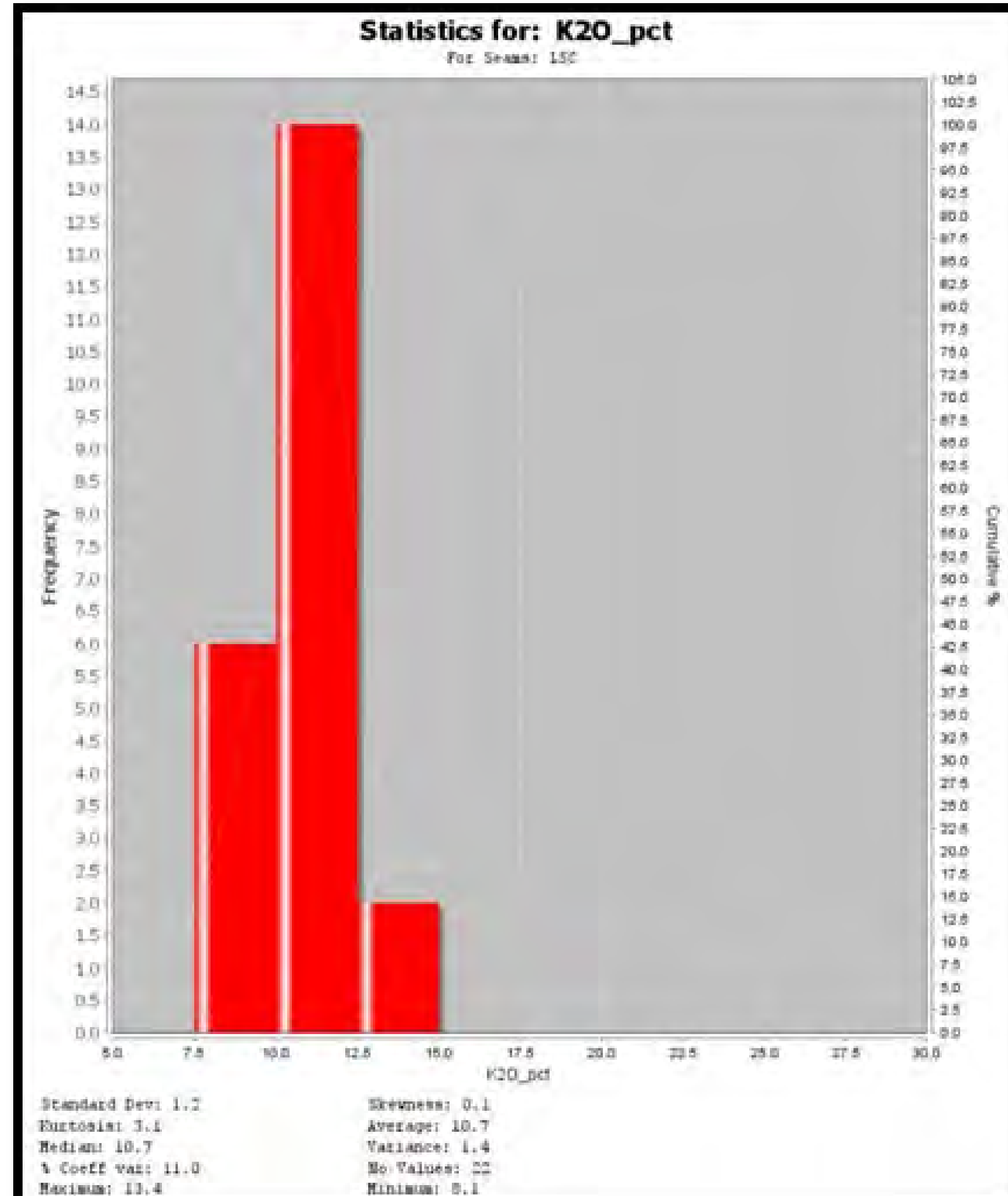
Source: CSA, 2012

Figure 12-3

K_2O% Histogram for USS (left) and USC (right)





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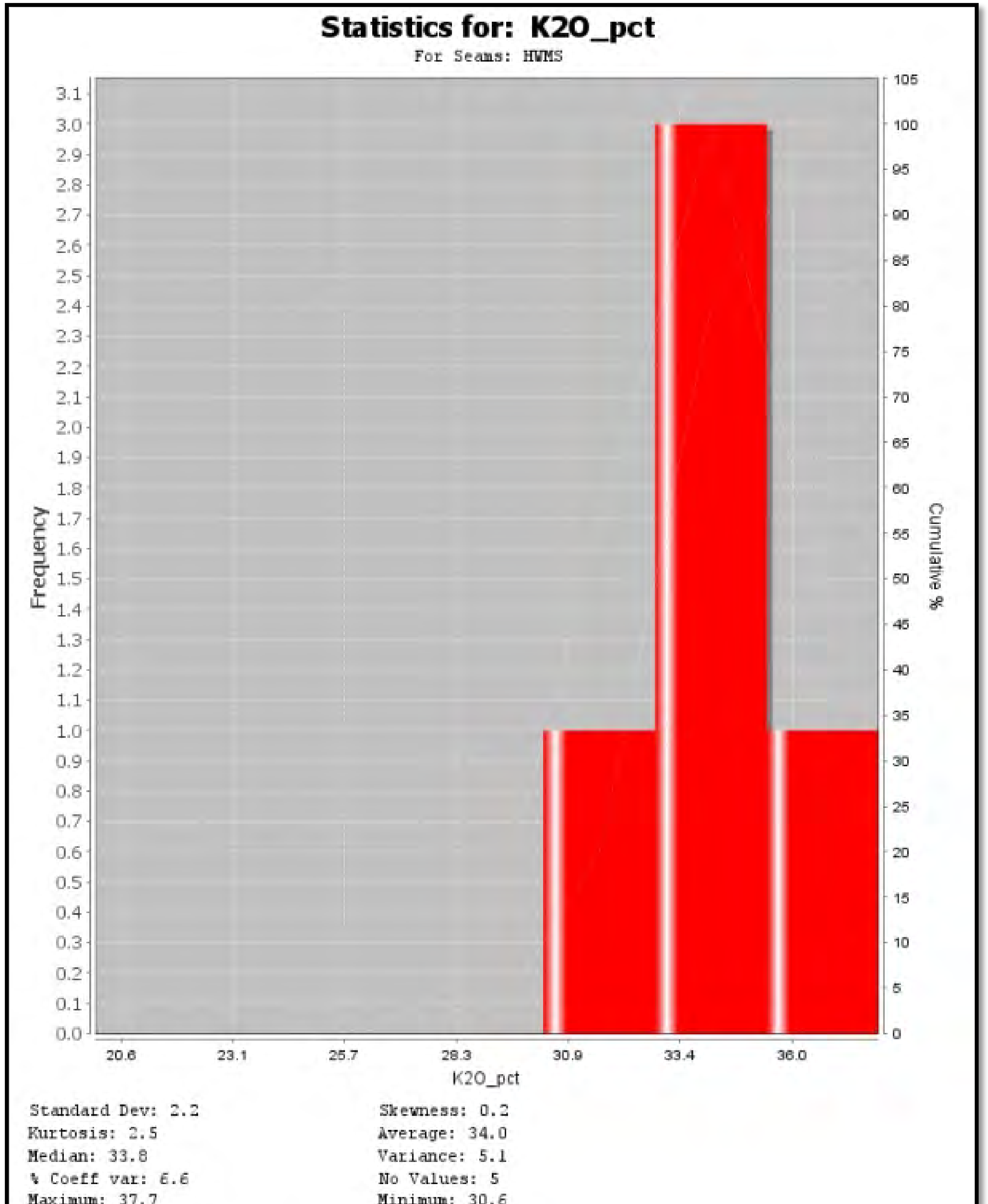
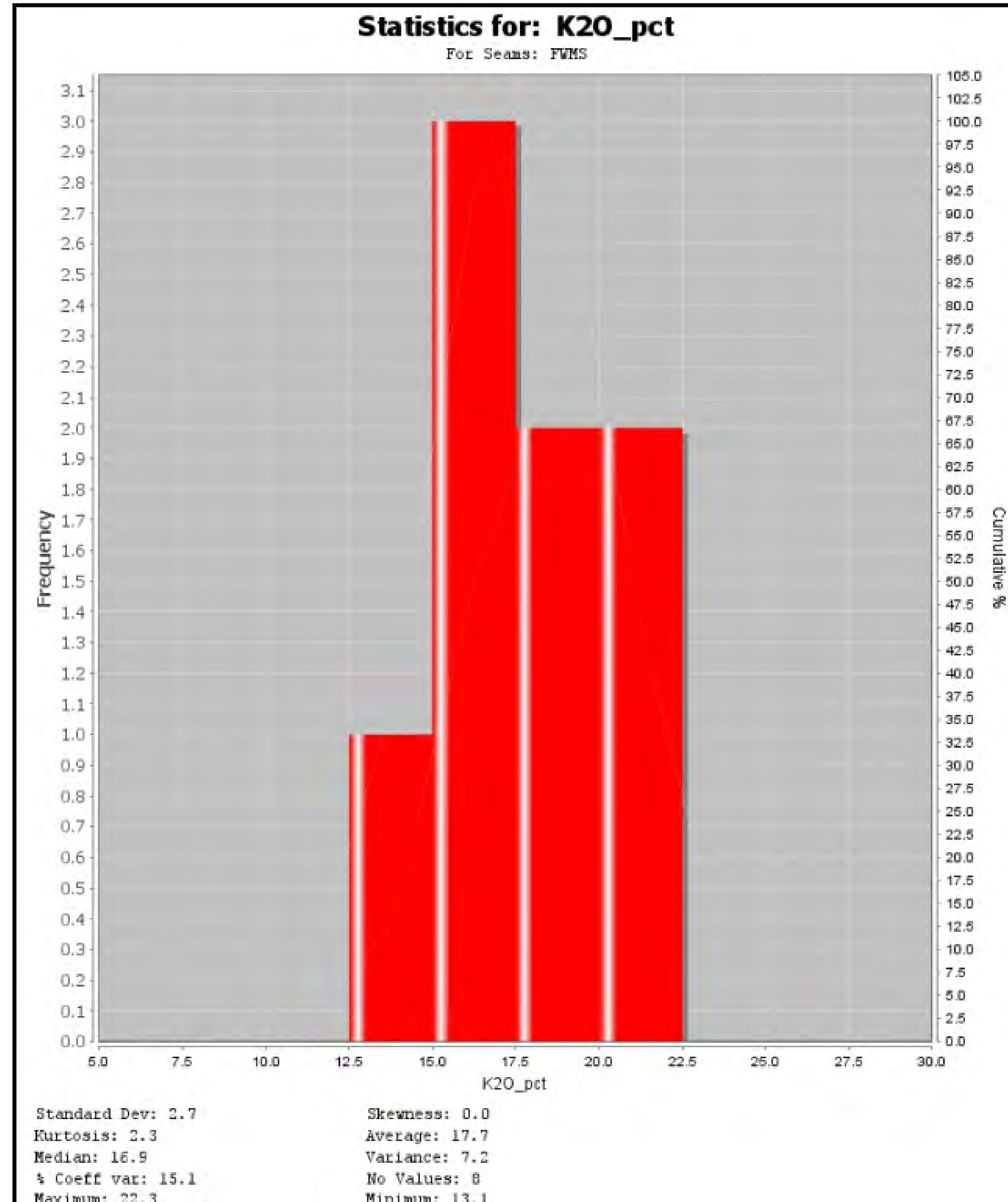
Source: CSA, 2012

Figure 12-4

K_2O% Histogram for LSS (left) and LSC (right)





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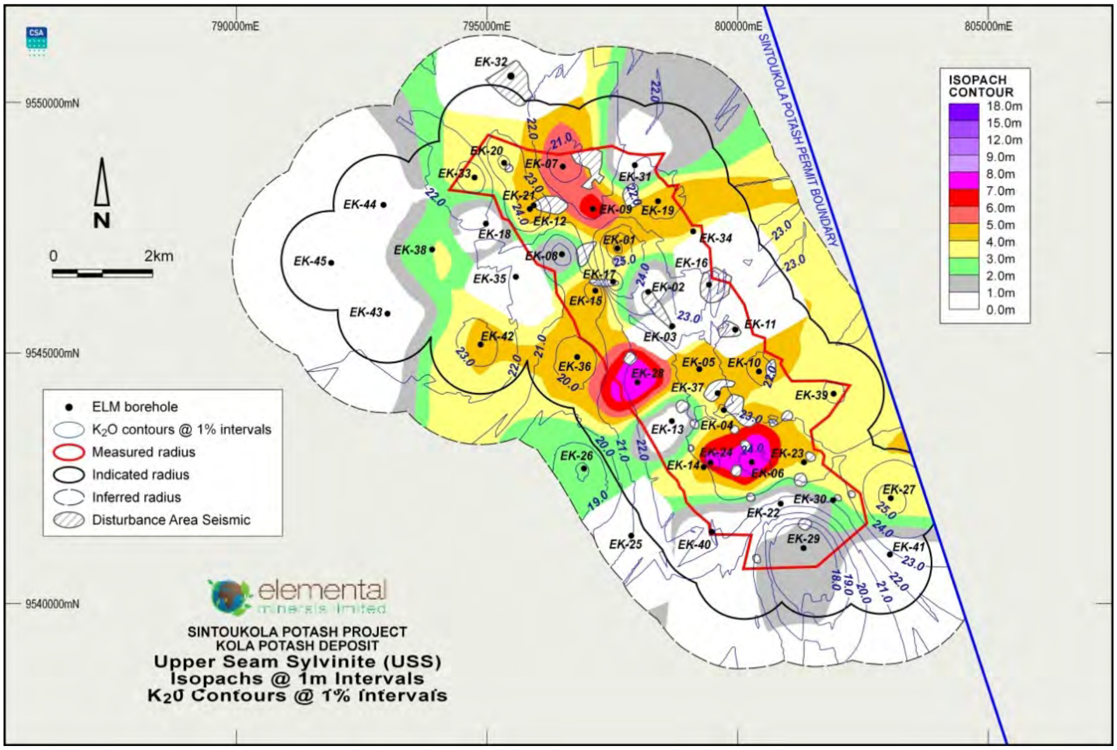
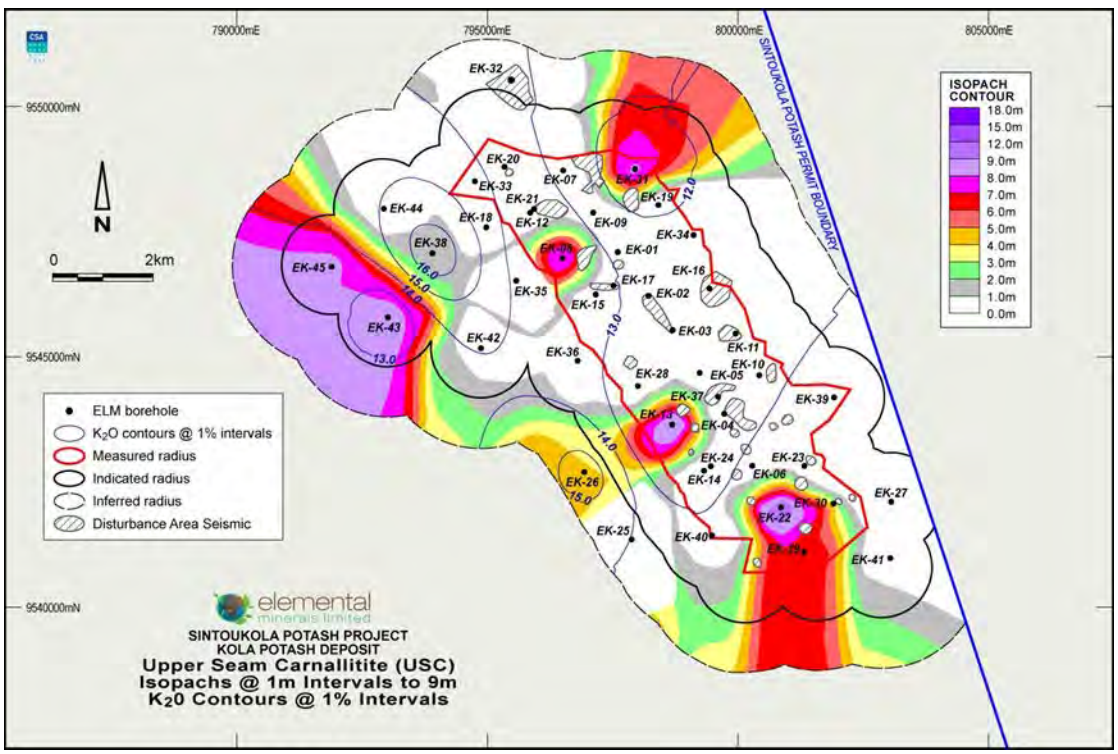
Sintoukola Potash Project, Republic of Congo

Source: CSA, 2012

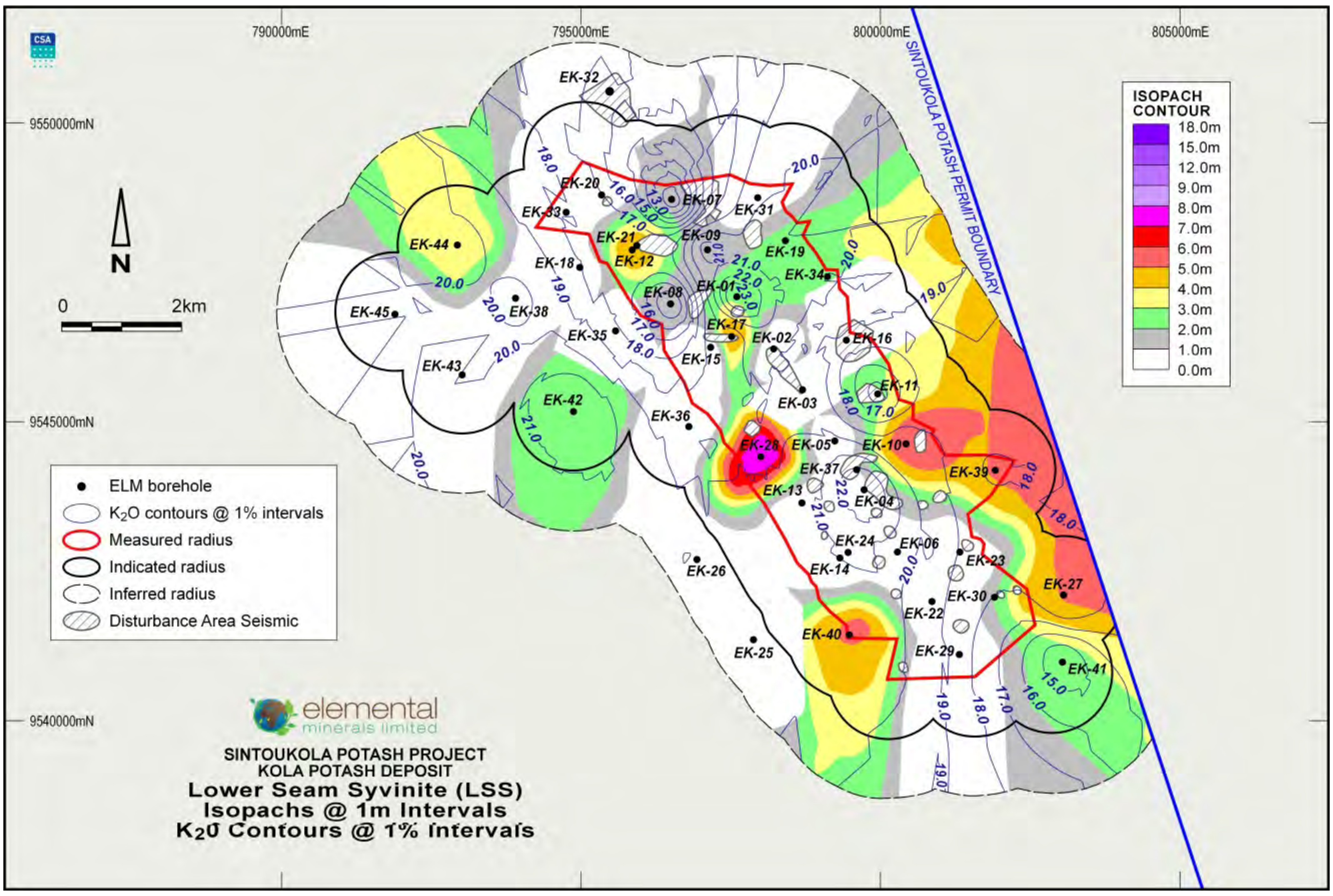
Figure 12-5

K_2O% Histogram for HWS (left) and FWS (right)

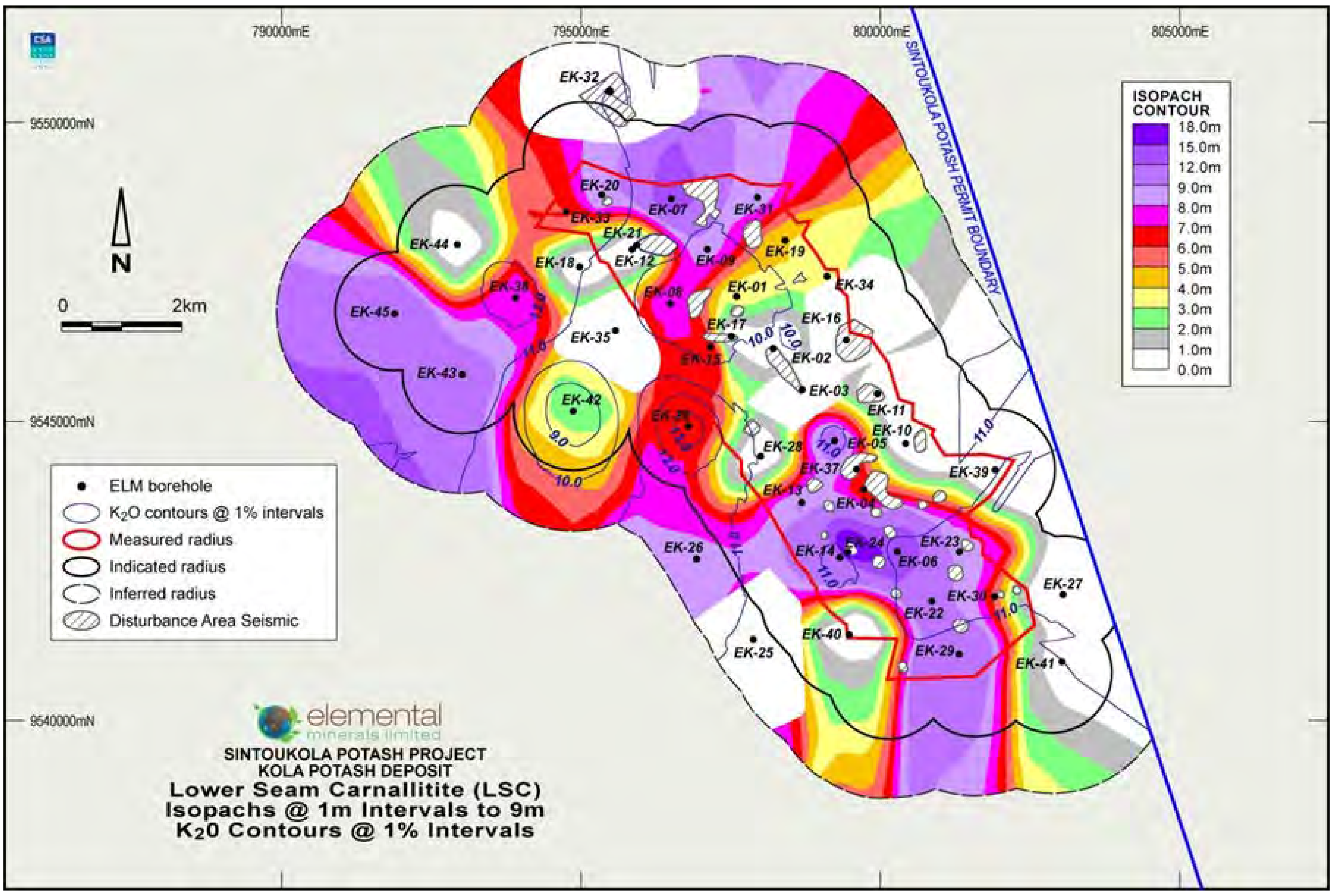




srk consulting	Sintoukola Potash Project, Republic of Congo elemental minerals limited Source: CSA, 2012	**Figure 12-6** **Plan View Showing Extent of Modelled Mineralised Upper Seam Mineralisation.** (Isopachs at 1m intervals. K_2O values contoured at 1% intervals)



790000mE
795000mE
800000mE
805000mE
9550000mN
9545000mN
9540000mN
SINTOUKOLA POTASH PERMIT BOUNDARY
ISOPACH CONTOUR
18.0m
15.0m
12.0m
9.0m
8.0m
7.0m
6.0m
5.0m
4.0m
3.0m
2.0m
1.0m
0.0m
N
0
2km
ELM borehole
K2O contours @ 1% intervals
Measured radius
Indicated radius
Inferred radius
Disturbance Area Seismic
elemental
minerals limited
SINTOUKOLA POTASH PROJECT
KOLA POTASH DEPOSIT
Lower Seam Syvinite (LSS)
Isopachs @ 1m Intervals
K2O Contours @ 1% Intervals



790000mE
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SINTOUKOLA POTASH PERMIT BOUNDARY
ISOPACH CONTOUR
18.0m
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12.0m
9.0m
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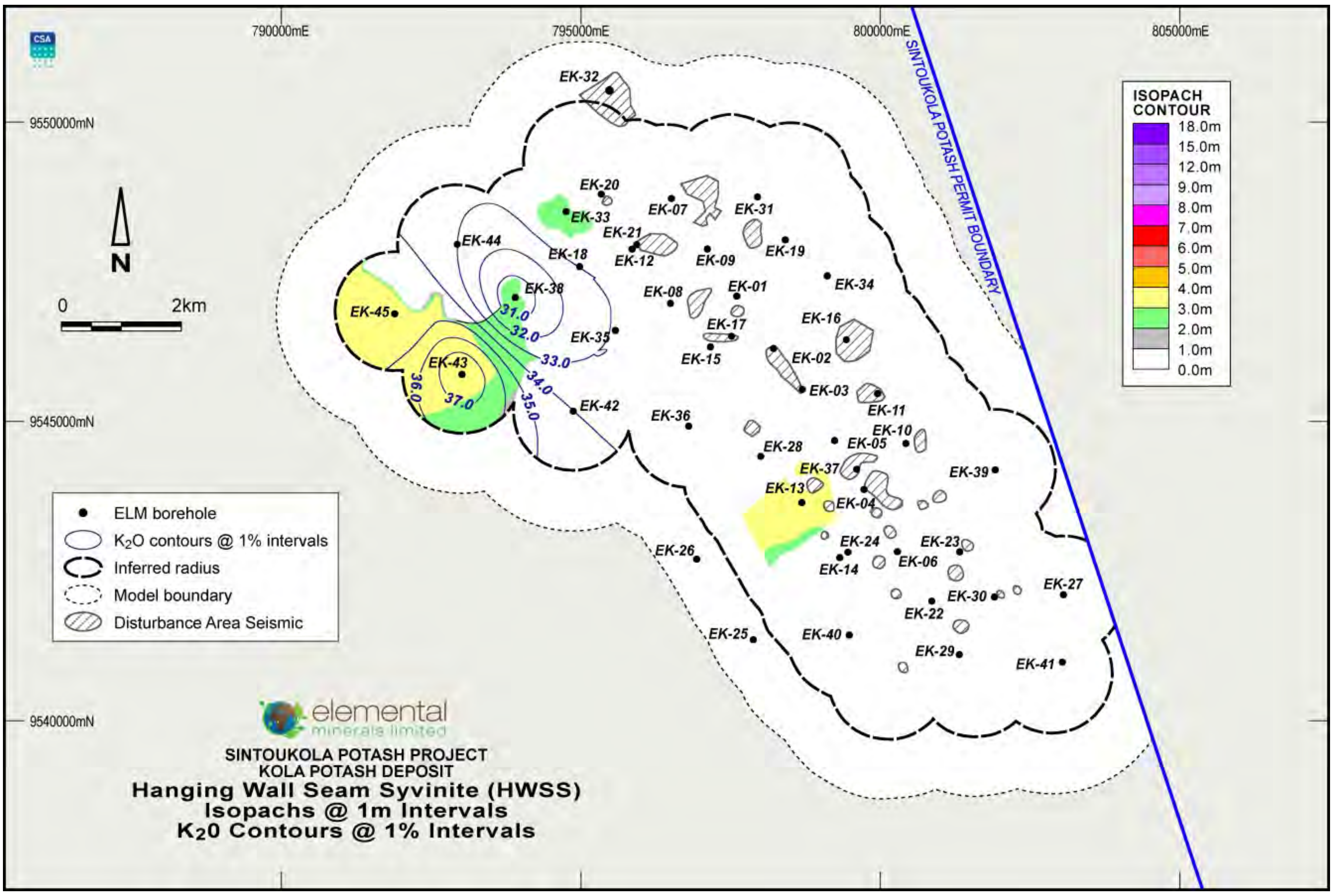
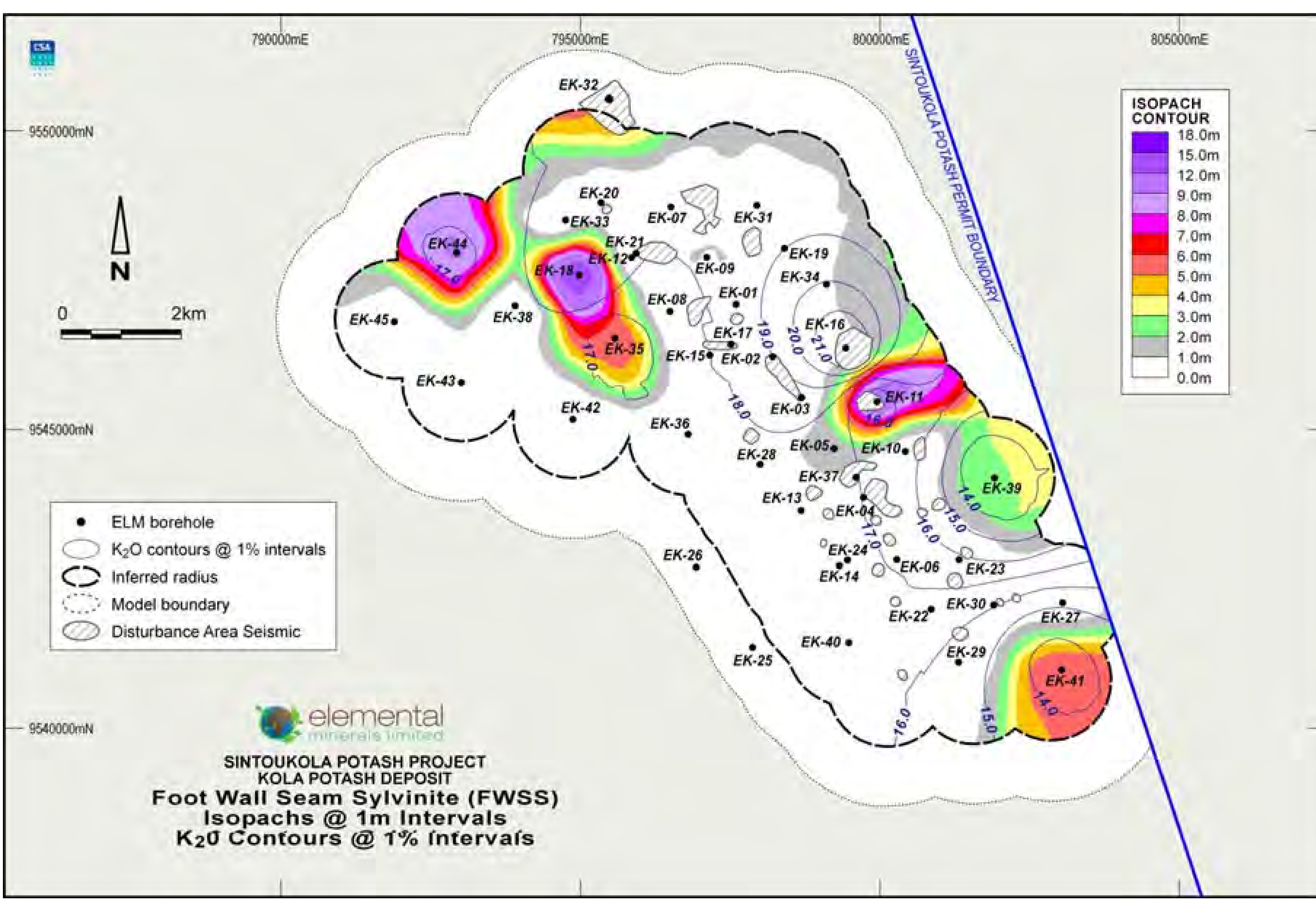
N
0
2km
ELM borehole
K2O contours @ 1% intervals
Measured radius
Indicated radius
Inferred radius
Disturbance Area Seismic
elemental
minerals limited
SINTOUKOLA POTASH PROJECT
KOLA POTASH DEPOSIT
Lower Seam Carnallitite (LSC)
Isopachs @ 1m Intervals to 9m
K2O Contours @ 1% Intervals





srk consulting	elemental minerals limited Sintoukola Potash Project, Republic of Congo	Figure 12-8
	Source: CSA, 2012	Plan View Showing Extent of Modelled Mineralised HWSS (above) and FWSS (below). (Isopachs at 1m intervals. K2O values contoured at 2.55% intervals)

13 Ore Reserves (Item 15)

13.1 Reserve Estimation

The underground ore reserve for the Sintoukola Project is presented in Table 13.1.1. The reserve estimation was carried out using Vulcan software.

Table 13.1.1: Sintoukola Project Underground Ore Reserve

Classification	Ore Reserves Tonnage (kt)	K_2O (%)
Proven	87,918	20.01
Probable	63,820	20.02
Total (Proven & Probable)	**151,738**	**20.02**

A CM room and pillar mining method is used as the basis for the design and reserve estimation. This mining method is suitable due to the orebodies characteristics of shallow dip angle, thickness, and ore strength which are conducive to mechanical cutting.

Table 13.1.2 shows Mineral Reserves by seam as specified in the mine design.

Table 13.1.2: Sintoukola Project Underground Ore Reserve by Seam

Seam	Classification	Ore Reserves Tonnage (kt)	K_2O (%)
USS	Proven	58,004	21.19
	Probable	53,065	20.46
LSS	Proven	29,914	17.73
	Probable	10,755	18.85
Total	**Proven**	**87,918**	**20.01**
	Probable	**63,820**	**20.02**

The reserve estimate was determined from only Measured and Indicated Resources. The overall conversion of resource to reserve is 26%.

13.2 Conversion of Mineral Resources to Mineral Reserves

Two aspects are evaluated during the conversion of Mineral Resources to Mineral Reserves:

- The ore extraction method(s) used in relation to the orebody characteristics which determine mining dilution and recovery; and
- Sintoukola Project operating costs and resulting CoG.

The mine plan included:

- Mining recovery based on geotechnical extraction ratios;
- Exclusion of disturbance zones; and
- Pillars around existing drillholes.

In accordance with the CIM classification system only Measured and Indicated resource categories can be converted to Mineral Reserves. In all mineral reserve statements Inferred mineral resources are considered as waste.

13.2.1 Cut-off Grade Strategy

Given the characteristics of the Kola orebody, the concept of an economic or break even CoG was not considered to be applicable. Because of the sharp grade boundaries of the sylvinite seams and the fact that the economic CoG is below the resource CoG of 10% K_2O, all sylvinite in the Measured and Indicated Resource was considered for reserve conversion.

14 Mining Methods (Item 16)

The mine plan relies on the resource model provided by CSA. Items provided include drillhole data, mineralization roof and floor profiles, grades, material classifications, and seismic interpretations. Both sylvinite and carnallitite are listed in the CSA resource statement, however only the sylvinite data was provided and used for mine planning purposes due to the selected process design and economics. SRK did not verify modelling or classification parameters used by CSA, as CSA is the QP responsible for the resources.

Details on the reserve and mine planning are presented in Volume III (SRK, 2012b).

14.1 Mining History

Potash was discovered in the coastal Congolese Basin during oil prospecting in Gabon in 1930's. Subsequent exploration between the 1930's and 1950's revealed that the salt formations continued through ROC into Gabon. In the 1960's significant exploration for potash was completed across he Kouilou portion of the Congo Basin. The exploration discovered the Holle potash deposit, located to the south of the Sintoukola permit area, and identified sylvinite and carnallitite mineralization at the Kola deposit and other areas within the current Sintoukola permit area.

14.2 Selection of Mining Method

Prior to commencement of the PFS, a tradeoff study between conventional and solution mining was conducted for the Kola deposit. Conventional mining was chosen as the preferred option. The reasons for the decision were:

- Conventional mining showed a higher NPV;
- Conventional mining has a lower US$/t-product operating costs and comparable capital costs when compared to solution mining;
- Conventional mining has a smaller environmental impact on the surface. Solution mining requires surface roads and wellheads throughout the property where conventional mining only requires infrastructure around shafts leaving other areas undisturbed;
- Solution mining has a considerably higher water demand and therefore has an increased environmental impact on water resources;
- Solution mining operating costs are much more highly focused on energy and water which are in the variable cost category;
- Solution mining is generally better suited for thicker (greater than 10 m) potash seams, which are not encountered at Kola; and
- ELM is targeting 2.0 Mtpa, which is more aligned with conventional mining production scales.

At the time of the trade-off study, the geologic model was considerably different from the current interpretation showing a more continuous deposit and assuming only a single seam could be mined conventionally, whereas both seams could be mined using solution mining. As the model has evolved, it has become evident that both seams can be mined by conventional mining, which further enhances its advantage over solution mining.

The Sintoukola orebody is generally flat lying with local undulations, and local dips do not exceed a maximum of 15 degrees in the model. No faults or offsets have been identified. Potash deposits, such as the Kola deposit, are typically comprised of low strength materials lending themselves to non-explosive mining methods. Non-explosive mining methods allow for tighter mining control and less fracturing of neighboring rock. Ventilation requirements in a non-explosives environment, as well as utilizing a majority of electrical powered equipment rather than diesel are lower and the mining cycle is less interrupted as it doesn't require personnel to vacate areas of the mine at blasting times.

For the Sintoukola Project, underground extraction will be carried out using a room and pillar type mining method with drum-cutter, continuous mining machines. The soft properties of the mineralization are within the operating range of CM. Ore from the CMs will be loaded directly into shuttle cars that tram the ore to conveyor dump points (feeder breakers). Secondary conveyors transport the ore to main conveyors which in turn transport the ore to the shaft area for vertical conveying to surface. Figure 14-1 shows a CM and conveyor system.

Note that borers were not selected due to their inability to accommodate significant variability of mineralization, thickness, and undulations.

14.3 Geotechnical Design

Full details of the geotechnical study are presented in Volume IV of the PFS document (SRK , 2012c). In order to develop mine design criteria the geotechnical PFS work included the following:

- Development of a project specific geotechnical core logging manual accompanied by training in geotechnical core logging procedures to the client geologists;
- The detailed geotechnical logging by ELM and SRK of hard rock overburden and evaporite core from 11 of the PFS resource drillholes;
- Undertaking a suite of laboratory engineering characterisation tests on samples of loose drillhole materials, overburden core, waste and ore evaporite core;
- Development of a sectional deposit geotechnical model based on a combination of the interpretation of the geological model and the engineering characterisation of the various litho-geotechnical units derived from geotechnical core logging and laboratory testing;
- Development of 2D finite element models (using the Phase^2 software code) to determine stable pillar and drive dimensions and roof beam thickness and to examine the sensitivity of these elements to changes in lithology and model parameters;
- Estimation of ground movement at surface (subsidence) and at the critical anhydrite/halite interface;
- Estimation of underground support requirements;
- Estimation of shaft pillar thickness and distance between shafts; and
- Check analyses using the Phase^2 software to verify the design parameters by the construction and analysis of a number of cross sections through the updated geological model.

Engineering characterization of the deposit rock mass was based on the detailed logging and subsequent laboratory testing of material from drillholes EK_17 to EK_29. Specific overburden and evaporite logging manuals were prepared for the Sintoukola Project and SRK provided geotechnical logging training to ELM geologists. QA/QC checks were conducted on data collected by site geologists in the form of periodic site visits, check core logging and review of core photographs.

Samples of bulk overburden, overburden core and evaporite core were collected and shipped to a number of laboratories around the world for testing, as described below:

- Salt samples were sent to the Institut für Gebirgsmechanik (IfG) rock mechanical laboratory in Leipzig, Germany which specialises in testing the creep behaviour of salt materials;
- Soil samples were sent to Structural Soils Ltd, Bristol, United Kingdom;
- Rock samples for UCS and direct shear testing were sent to Advanced Terra Testing Inc in Lakewood, Colorado, USA; and
- Mineralogical tests and thin section analysis were carried out at SRK in conjunction with the Cardiff University mineralogical laboratory.

From the results of this work, characteristic strength parameters for use in numerical modelling were developed. The strength parameters used are presented in Table 14.3.1.1.

Table 14.3.1.1: Analysis Strength Parameters

Material	Density (kg/m^3)	E Mod (GPa)	Poisson's Ratio	Friction Angle (º)	Cohesion (MPa)	Source
Unconsolidated O/B	2300	26	0.29	20	0.05	Engineering Judgement
Weathered Sandstone	2300	26	0.29	20	0.05	
Sandstone	2700	50	0.26	35	30	Estimated from limited core testing and geotechnical logging.
Limestone	2700	50	0.26	35	30	
Anhydrite	2500	50	0.26	35	30	
Halite	2160	35	0.28	30	10	Laboratory Testing
Sylvinite	2100	27	0.28	36	7	Laboratory Testing
Carnallite	1702	19	0.30	40	5	Laboratory Testing

The strength of the evaporite materials is critical in determining the response of pillars and excavations to mining. Due to limited availability of core for testing the test results obtained were benchmarked against a database of evaporite strength properties, held by the IfG laboratory to improve confidence in the parameters used for analysis.

The main requirements for the development of geotechnical mine design criteria were to ensure that mining did not produce movement in the halite roof beam that could give rise to dislocation and failure of the beam. This could potentially cause the development of connectivity between the mine workings and overlying aquifer which may result in flooding of the mine. All panel pillars were therefore designed as permanent pillars. A minimum factor of safety criterion of 1.5 was chosen for panel pillars. An average factor of safety criterion of 2.5 was chosen for those pillars surrounding main and sub-main excavations.

A series of sensitivity analyses was carried out to determine the effect of various physical parameters on the stability of the model, results are shown in Table 14.3.1.2. Parameters that were varied included:

- Sylvinite orebody thickness;
- Pillar width and pillar height in the panels;
- Pillar width between the mains excavations and between the outer mains and panels;
- Distance between the anhydrite and the orebody (the halite roof beam thickness);
- Thickness of the halite separating the base of the sylvinite and the underlying, weaker carnallitite (the halite floor beam thickness);

- The interburden thickness between US and LS;
- Presence of clay partings in the roof and floor of the excavations; and
- Presence of water sitting above the anhydrite.

The results of the sensitivity analyses are presented in Table 14.3.1.2.

Table 14.3.1.2: Results of Modelling Sensitivity Analyses

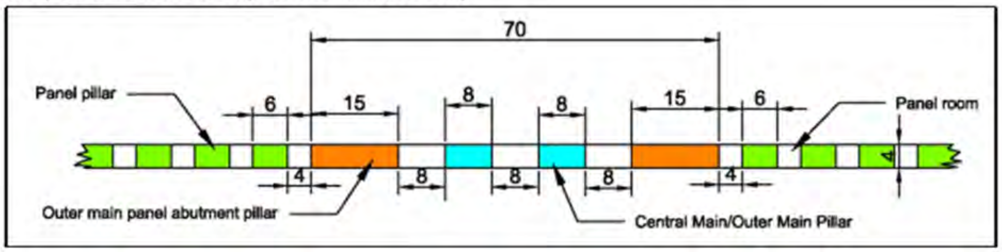
Variable	Sensitivity	Impact
Panel pillar width and pillar height.	High sensitivity	When the pillar width is less than the orebody height failure of all pillars occurs. When the pillar width is the same or greater than the orebody thickness the pillars are generally stable.
Thickness of halite roof beam	Moderate sensitivity	Some deterioration in mine stability or deformation of halite/anhydrite contact as halite thickness reduces.
Thickness of halite floor beam.	Low to Moderate sensitivity – Mains and sub-mains.	As the halite floor beam thickness reduces to 2 m there is increase in room floor heave (although no indication of pillar foundation instability). Largest amount of floor heave occurs in the central main excavations. Slight reduction in this when the mains pillar size is increased.
	Low sensitivity – Panel rooms.	Due to small span of rooms and relatively high W:H ratio of pillars.
Presence of carnallitite layer in floor	Moderate sensitivity – mains and sub-mains.	Increase in floor heave and reduction in pillar strength factor in the mains and sub-mains excavations and pillars.
	Low sensitivity – panel rooms.	Due to small span of rooms and relatively high W:H ratio of pillars.
Presence of clay layer in roof	Low sensitivity	Low impact due to relatively narrow room width.
Presence of groundwater aquifer.	No sensitivity	For the same mining geometry the inclusion of ground water as a dead load above the mine does not change the stability of the mine over a dry condition.

The mine design parameters resulting from the sensitivity analyses undertaken are presented in Table 14.3.1.3. Note that the excavation spans have been determined by excavation equipment selection criteria as well as geotechnical criteria. The check analyses undertaken on four cross sections through the deposit indicated that the mine design parameters developed from the sensitivity analyses were appropriate. Modelling confirmed that no rock mass instability was generated, and that the design factors of safety for all mine elements were achieved or exceeded. This suggested that negligible surface subsidence can be expected.

Table 14.3.1.3: Geotechnical Parameters

Mains and Sub-Mains	Width (m)	Height (m)
Central Mains, Outer Mains and Sub Mains	8	4
Central Main/Outer Main Pillar	8	
Outer Main Panel Abutment Pillar	15	
In-Panel Development		
Panel Entry Drives	8	Variable
Panel Entry Drive Pillar	8	
Panel Rooms	4	Variable
Panel Pillars	Variable	

Main/Sub-Mains and Panel Layout for a 4m thick orebody



Panels and Panel Pillars	Orebody Height (m)	Pillar Width (m)	Panel Room Width (m)	Panel ER (%)
	2	4	4	53%
	3	5	4	49%
	4	6	4	45%
	5	7	4	43%
	6	8	4	40%
	7	9	4	38%
	8	10	4	37%
These heights are applicable only if Upper and Lower Seams and interburden halite are mined together	10	12	4	34%
	12	14	4	32%
	14	15	4	30%
	16	17	4	28%

Thickness of Halite Roof Beam above Roof of US	Total Mining	Halite Beam Thickness
	Up to 8m	15
	10	17.5
	12	20
	14	22.5
	16	25
	18	27.5
	20	30

Minimum Interburden Thickness between Upper and Lower Seam	5m

14.3.1 Other Geotechnical Design Parameters

Beam between USS and LSS.

In such areas where the USS and LSS will both be mined a stable beam must be left in-place between the seams. The minimum thickness of the beam, regardless of mining heights, is 5 m.

Shaft Pillars

Two shafts will be developed to access the ore. It is preferable that the shafts be located near each other to share surface facilities and limit sterilization of mineralization. In order to maintain stability a 300 m radius pillar will be left in place around the shaft. Separation of the shafts should be 100 m. Shaft pillar orientation is shown in Figure 14-2.

Drillhole Pillars

Drillhole pillars are pillars left around existing exploration drillholes that pierced through the anhydrite/gypsum sequence and have the potential to conduct water from overlying horizons to the mining horizon. Although these drillholes will be grouted and sealed, a 30 m radius pillar is required around existing drillholes to protect from this potential conductivity.

Disturbance Zone Pillars

Disturbance zones refer to areas where anomalies have been detected on close spaced seismic lines. The anomalies could be collapse feature, faults, etc. A 50 m buffer zone pillar is required around known disturbance zones.

Minimum Thickness of Halite Above Mining

The minimum halite thickness refers to the beam of halite from the top of mining to the base of anhydrite and potentially water bearing units above it. Based on measured properties as discussed in Volume IV of this PFS, the anhydrite sequence could be considered an aquitard, where it is present. However, because drilling has found it to be missing or anomalously thin in approximately 10 percent of drillholes, it is considered to be unreliable as an impermeable seal between the mine and the overlying aquifer. The halite sequence may be considered to be an aquitard as breaks in the halite would be healed over time.

A minimum thickness of 15 m between the roof of mining and the base of the anhydrite/gypsum sequence is required. The required thickness varies with total mining height as shown in Table 14.3.1.1.

Table 14.3.1.1: Thickness of Halite Roof Beam Above Mining

Total Mining Height (m)	Halite Beam Thickness (m)
Up to 8 m	15
10	17.5
12	20
14	22.5
16	25
18	27.5
20	30

<u>Ground Support</u>

<u>Mains:</u> The geotechnical design stipulates that the main entries through the mine are to be reinforced by pattern bolting with 1.8 m long 22 mm diameter resin grouted bolts. Additional ground support may be required as identified by underground operations personnel; and

<u>Panels:</u> Ground support in production panels will be installed where identified as necessary by underground mining personnel as part of routine inspections of working places and travelways. Anticipated conditions that may require ground control measures include, but are not limited to, variations in the seam mineralogy such as halite pockets in the roof and ribs, bedding separation on clay partings, carnallitite lenses and salt horses.

14.4 Mine Design

A single grade value was provided in the resource model for each x,y location for each seam, based on a 25 by 25 m x,y spacing. With this type of model selective mine planning (i.e. targeting higher grade) within a seam cannot occur. If the full height of the seam is not mined the average grade for the entire seam has to be assumed. Seams considered for mining include the USS and LSS as only sylvinite can be processed in the process plant described in Volume VI Metallurgy and Processing of the PFS prepared by AMEC. As all material in the resource model is above CoG, it was not necessary to exclude zones due to grade.

A multiple step process was developed to create the panel design. This process is outlined in the following sections.

14.4.1 Disturbance Areas

Disturbance areas were identified and excluded from the mineral resource by CSA, as discussed in Section 12.5.1. They represent locally developed depressions due to salt removal by dissolution akin to salt collapse structures in the Canadian potash district. A disturbance area has the potential to affect the laterally continuous character or preserved thickness of the potash-bearing beds and may represent a zone that is potentially unsuitable for underground mining.

Figure 14-3 shows the identified disturbance areas along with the classification outlines.

As the existence of mineralization and units above/below these disturbance zones is in question, these areas are considered sterile in order to minimize risk of ground instability and potential water inflows from overlying units. For the purposes of mine planning, a 50 m buffer pillar will be left in place around the disturbance zones.

The total disturbance area including buffer pillars was determined to be 15% in the Measured Resource. This factor was applied to the mine design within the Indicated Resources area.

Additionally, underground horizontal drilling will occur once mining commences to prove and test areas prior to mining. If a disturbance zone is encountered in underground drilling, it should be sterilized and treated with caution.

14.4.2 Shaft Location

Detailed geology in various areas of the deposit is unknown, and therefore shaft location was selected from information in existing drillholes. The shaft location was selected by process of eliminating unsuitable areas and then selecting the optimal location of suitable areas.

Shaft siting considered the following:

- Primary elimination Criteria:
  - Away from known disturbance areas;
  - Drillhole that intersects mineralization; and
  - Shaft and shaft bottom infrastructure should be in halite rather than carnallitite.
- Secondary elimination Criteria:
  - Total shaft depth;
  - Approximate center of mass of the mineralization;
  - Confidence in area (i.e. spacing from detailed seismic);
  - Surface information (i.e. near rivers); and
  - Minimize sterilization of ore in shaft pillar.

All exploration drillholes were evaluated against the primary and secondary categories. Final selected shaft location is near drillhole EK_05 at the coordinates shown in Table 14.4.2.1. Figure 14-4 shows the shaft location.

Table 14.4.2.1: Shaft Location Coordinates

Shaft	Easting (x)	Northing (y)
Production Shaft	798,852	9,545,113
Services Shaft	798,852	9,545,213

Prior to shaft sinking pilot drillholes should be drilled recovering core for the length of the drillhole to ensure geology is suitable and as expected.

14.4.3 Upper Seam Sylvinite (USS) Layout

Mining thickness contours were prepared for the USS. Only material classified as Measured and Indicated Resource was considered for mine design. A minimum mining height criteria of 1.8 m was then applied to determine mineable areas (i.e., areas less than 1.8 m thick were excluded from the design). These areas are shown in Figure 14-5a. From the mineable areas access mains were laid out from the shaft location to the extremities and will serve as access to the various panels. Mains avoided all disturbance areas and were kept as straight as possible to eliminate transfer points as conveyor belts will be installed in all mains. Figure 14-5b shows the mains layout.

From the mains, panel polygons were laid out as per the geotechnical parameters discussed in Section 2.4. Average panel length is approximately 1,000 m with maximum panel length of 1,300 m. A detailed layout within a panel was not completed at this time, however each panel was cut into separate pieces based on mining height. This allows reserve calculations to be run for each individual mining height and application of appropriate extraction ratios based on height. The panel layout is shown in Figure 14-5b.

14.4.4 Lower Seam Sylvinite (LSS) Mining

The LSS exists in some areas where the USS is mined and in some areas where it is not mined or does not exist. In areas where mining does not occur above the LSS, the full height of the LSS can be mined, provided it meets the minimum height and grade requirements.

In areas where USS mining has occurred, a 5 m beam must be left in situ between the seams for stability purposes. The beam can be ramped through to provide access to the LSS and then mining of the LSS commences similar to the USS. Pillar layout of the USS and LSS will be stacked to ensure load transfer and maintain integrity of the 5 m beam.

If 5 m of interburden does not exist between the seams then a portion of the LSS will be left in situ to maintain the 5 m requirement. In several places where interburden is thin, the LSS is thick enough that up to 3 m may be left in situ and still meet the minimum mining height requirement.

Figures 14-6a to 14-6c shows the identification of mineable areas for the LSS as well as the final design of the LSS. Each panel was cut into separate pieces based on mining height again allowing reserve calculations to be run for each individual mining height and application of appropriate extraction ratio based on height. Ramps between the USS and LSS have not been designed at this time. Maximum elevation difference between USS and LSS is approximately 15 m and therefore the expected ramp lengths will be short and their impact on grade dilution minimal.

14.4.5 Thickness to Anhydrite

A minimum thickness requirement between the roof of mining to the base of anhydrite of 15 m was specified in the geotechnical parameters outlined in Section 14.3. Figure 14-7 shows thickness as modeled between these surfaces. Areas of 15 m or less are eliminated from the design.

Seismic techniques are unable to delineate discrete potash beds due to insufficient contrast between the halite and sylvinite densities. However, this same density difference between halite and anhydrite allows seismic techniques to map the base of the anhydrite with some certainty. Contour maps of the potash beds were prepared by CSA strictly from drillhole data. Because the contours of the base of the anhydrite (top of salt) and contours of the potash beds were produced using two separate methods, the reader is cautioned that, although there is no real evidence indicating so, there may be areas of the deposit with thinner salt cover than the 15 m minimum that would result in a reduction in the mineral reserve tonnages. Conversely, some areas that have currently been removed could potentially be re-included in the design. For the purpose of the PFS the issue of salt back thickness has been identified as of being of particular importance and should be further understood and quantified in future underground exploration.

14.4.6 Dilution

There is little visible distinction between ore and waste in the sylvinite seams. It is expected 15 cm of dilution will be encountered at the top and bottom of mining with a continuous grade control program in place. As the grade of mineralization is not a hard boundary it is expected the dilution will carry some grade. The grade of dilution was determined from the drillholes. An average dilution grade of 4.15% K_2O was used for the USS and 4.45% used for the LSS. LSS dilution may have higher than average grade in some areas as mineralization is left in situ to provide required beam between seams and therefore a higher grade of diluted material is expected, but not accounted for in the current schedule for these areas.

14.4.7 Detailed Layout

Two levels of detail of pillar layouts were used in the mine design process, a detailed layout and a life of mine (LoM) layout. The detailed layout was prepared in a single area to show how the panels will be developed with multiple cutting faces for each machine. The pillars are rotated in such a way that the maximum turning angle is 45° for both the CMs and shuttle cars transporting the ore. Figure 14-8 shows the detailed layout.

The LoM layout was prepared for the entire mining area and included polygons outlining entire panels. Figure 14-9 shows the detailed layout in comparison to the LoM layout. For the next level of study a detailed layout should be designed for a portion of the mine.

14.5 Grade Evaluation

Grades were provided by CSA in the form of 50 x 50 blocks with K_2O and Mg grades for each block and for each seam. These values were imported into a pseudo-block model containing a single block in the z elevation for calculating grade/tonnage of the various designed mining areas.

An average in situ bulk density of 2.07 t/m^3 was assigned for the sylvinite mineralization by CSA. SRK used this density value for all reserve calculations.

Once grades are reported from the block model the following steps occur to calculate the reserve:

- Extraction ratios as per geotechnical requirements in Section 2.4 are applied to the panel areas based on mining thickness;
- Discount Indicated Resource material for disturbance zones and associated buffer pillars (15% - as discussed in Section 12.5.1);
- Apply a recovery factor of 90% in all areas to account for additional mining losses in the USS, and 85% factor in the LSS; and
- Add dilution of 15 cm on top and bottom of seam. An average of 4.15% K_2O was used for the USS and 4.45% used for the LSS.

A summary for each seam is given below in Table 14.5.1.

Table 14.5.1: Reserve Summary by Seam

Area	Ore Tonnes (Mt)	K_2O (%)	MG	Avg Tk	Contained MoP (Mt)	Recoverable* MoP (Mt)
USS	111.0	20.8	0.032	4.35	38.3	34.3
LSS	40.7	17.8	0.216	3.67	11.9	10.6
Total	**151.7**	**20.0**	**0.082**	**4.17**	**50.2**	**44.9**

*Recoverable MoP takes into consideration process plant recovery of 89.5%

Figures 14-10 to 14-12 show mining panel reserve K_2O grades, thicknesses and Mg content. The average Mg estimated in the reserve is well below the process plant upper limit of 0.25%.

14.5.1 Grade Control

Visual grade control is not considered to be possible for either the USS or the LSS as a result of the lack of distinct marker horizons within, above, or below the sylvinite horizon.

Grade control using K40 gamma emission is considered to be appropriate for the anticipated geological conditions at the Sintoukola Project. On a room/panel scale, grade control will be conducted by short (10 to 15 m) low angle probe drilling into the back and floor of the mining rooms ahead of the mining face. A gamma probe down hole tool, calibrated to K40 grade, is then manually inserted into these drillholes to determine the elevation of the top and bottom of the potash seam. Instructions are then issued to the CM operator to ensure that the ore horizon is followed. This technique has been used successfully at operating potash mines for many years.

The complexity of the geological controls on grade distribution at the Kola deposit are such that the simplistic approach used in the development of the resource model is not considered to be sufficient to allow operational mine planning. Variations in grade and thickness are likely on a local scale that are not evident in the current resource model. A geologist will be allocated to each working area of the mine to ensure proper grade control procedures are consistently followed.

On a regional scale (1 to 2 km from workings) horizontal core drilling can be used to better define the grade distribution within the mine foot print. This technique is used at several operating potash mines to improve the resource definition without the penalty of resource sterilization associated with surface exploration drillholes.

14.6 Production Schedule

In order to ramp up production as quickly as possible, initial development will occur in two directions from the shaft. A new CM will be added when headings become available. Once mains are developed to their extremities, panels will be mined using a primary, secondary, and tertiary mining methodology, as shown in Figure 14-13, to allow for even settling/movement of large areas.

The grade to the process plant was considered a design parameter and consequently, the higher grade, southern portion of the deposit is developed ahead of other areas. As the grade differences are low between the areas and many miners are contributing to mill feed, this is the extent of grade optimization within the design. The scheduling is focused on mining large areas completely prior to moving into new areas to eliminate the need for ventilating southern and northern extremities at once.

Scheduling was undertaken with the goal of providing 6.85 Mtpa of Run of Mine (RoM) to the process plant, or approximately 2 Mt of recovered MoP per year.

In order to achieve this target the production rate stays fairly constant for the life of the mine and the number of CMs in use, once full production is reached, will stay constant.

14.6.1 Productivity

The assumed CM productivity is shown in Table 14.6.1.1. These rates combine manufacture (Sandvik) information with experience from other current potash operations utilizing similar equipment.

Table 14.6.1.1: Continuous Miner Productivity (MC470)

Parameter	Unit	Value
Theoretical Productivity	tpd	5,000
Availability	%	90%
Utilization	%	90%
Operational Utilization	%	64%
Productivity/day*	tpd	2,600
Productivity/annum	tpa	950,000

*based on 365days/year

For scheduling purposes one type of CM has been assumed, whereas in reality a mixed fleet will be used with smaller machines for lower mining height areas. This is accounted for in the economic model on a percentage basis.

Each day will consist of three 9-hour shifts (hot-change overlap between shifts) and it is assumed that the CMs will on average attain an overall utilization of 52%. This includes:

- Travel time;
- Safety meetings;
- Operator checks;
- Planned maintenance;
- Breakdowns;
- Rebuilds; and
- Other miscellaneous delays affecting the mining operation.

This 52% overall utilization results in a CM productivity of 2,600 tpd and 950,000 tonnes per annum (tpa).

All other equipment was scheduled to meet the production of the CM and is detailed in the following sections.

14.6.2 Development and Production Schedule

All scheduling work was done using Minemax iGantt software. The following parameters were used when creating the schedule:

- Process plant capacity is approximately 570 kt of RoM per month;
- Mine one area completely prior to moving to another (southeast first, then northwest) to minimize ventilation and services requirements;
- Mine on retreat where possible;
- Mine USS prior to LSS mining in any one area;
- Five day panel change provision was included for each machine as it is moved from one panel to another; and
- A rate of 2,600 tpd was used for each CM. For production ramp up 50% of this rate was used for the first 6 months, 75% next six months, and full production rate thereafter.

Based on the shaft sinking methodology, the shaft bottom layout and areas adjacent to the shafts are developed in conjunction with shaft sinking.

Mining begins with two CMs accessing the south portion of the orebody. The CMs work in unison to develop the main access ways to the extent of the design. Once the extents are reached the C's will begin work in the panels off the main roadway. One CM is assigned per panel and utilizes the multiple cutting faces available to it. As mains are developed and additional equipment is available submains are developed from the mains to allow panel access. Mains and submains are developed using the same methodology with the difference between the two being permanent (LoM) vs. temporary (several years) conveyor installations.

Table 14.6.2.1 shows a summarized annual production schedule for underground ore production. This will vary slightly from material being processed in the plant due to the surface stockpile and the timing of transporting the material to the process plant.

Table 14.6.2.1: Yearly Production Schedule

Year	Mining Ore Tonnes (kt)	K_2O Grade (%)	Avg Mining Thickness (m)	Ore Tonnes to Process Plant (kt)	Process Plant MoP Tonnes (kt)
1	2,369	20.71	4.6	1,538	467
2	6,342	20.45	4.08	6,775	2,053
3	6,871	21.19	4.16	6,840	2,151
4	6,874	21.25	4.17	6,840	2,149
5	6,871	21.21	4.14	6,859	2,158
6	6,875	20.42	4.31	6,840	2,078
7	6,874	19.48	4.34	6,840	1,977
8	6,877	19.78	4.45	6,840	1,993
9	6,861	19.83	4.58	6,859	2,012
10	6,856	19.43	4.74	6,840	1,963
11	6,878	19.81	4.87	6,840	2,005
12	6,882	19.62	4.89	6,840	1,987
13	6,880	19.74	5.00	6,859	1,996
14	6,881	19.33	4.52	6,840	1,974
15	6,874	19.17	4.75	6,840	1,926
16	6,880	19.53	4.37	6,840	1,974
17	6,872	19.63	4.42	6,859	1,990
18	6,862	20.23	4.32	6,840	2,041
19	6,858	19.90	4.20	6,840	2,021
20	6,883	19.93	4.34	6,840	2,008
21	6,817	20.63	4.82	6,859	2,086
22	6,598	20.13	4.76	6,840	2,046
23	5,242	19.22	5.17	5,870	1,678
24	661	20.42	6.43	294	88

Note that all development is in ore and therefore no waste is reported in the production schedule.

A color coded map showing the annual mining schedule is shown in Figure 14-14.

14.7 Shaft Sinking

A tradeoff study was completed comparing decline access and vertical shaft access and a decision was made to use vertical shaft access due to schedule, shaft pillar considerations, ground conditions and the presence of the aquifer. Two shafts will be required for ventilation and for secondary egress from the mine and their development is discussed in the following sections.

Initially a traditional shaft sinking approach was undertaken by G.L Tiley and Associates (Tiley). This work included development of a methodology, schedule, and cost based on information available specific to the Sintoukola Project. The ground conditions and high water table in the overburden units require a ground freeze to excavate the shafts. Due to current lead times on sinking equipment, necessary engineering work for this type of shaft and freezing requirement, the schedule commissioning of the shaft occurred in 2016 which is not in line with ELM's development plan. Alan Auld Engineering (AAE) was therefore engaged to look at alternative shaft sinking options mainly focusing on civil type construction earthworks. The depth to ore is considered to be shallow for traditional shaft access mine and more appropriate for civil works projects, however, both traditional sinking and civil type methods can be applicable.

AAE was able to identify an alternate means of shaft sinking that allows the commencement of production in 2015, which is in line with ELM's development plan. The alternative approach uses civil engineering shaft sinking principles that involves less front end work and does not require the procurement of special hoisting equipment. Material is removed from the shaft and mine using a high angle conveyor (HAC) and industrial lifts are used for equipment access. A ground freeze is still utilized for shaft development.

The AAE approach to shaft construction was selected for the purpose of the PFS.

14.7.1 Shaft Design

A civil excavation method allows for alternative materials handling techniques to be used such as HAC and industrial lifts. With these systems long skip overruns and skip loading pockets are not required and overall shaft length can be reduced. Using the civil/HAC approach, the shaft length can be shortened to 235 m.

Using this cost effective approach, two large diameter shafts (12 m diameter) will be sunk to allow for efficient machine and personnel movement and effective ventilation flow. Two adjacent, auxiliary shafts (8 m diameter) will be sunk to a depth of 100 m and will be used to assist the shaft sinking and mucking process. Each auxiliary shaft will be connected to the main shaft on this sub-level. Upon completion, the auxiliary shafts will be equipped as plenums for the mine ventilation system. Figure 14-15a shows the spacing and layout of the shafts.

14.7.2 Shaft Sinking Method

A summary of the shaft sinking methodology includes the following items:

- **Equipment -** sinking a civil type shaft involves using a large crawler or road crane and a skip, with a 360 degree excavator working on the shaft floor to fill the skips;
- **Ground Freeze** - the ground freeze drillholes will be stopped just above the halite at 200 m in depth to ensure that fracturing of the halite by the freeze process is minimized to prevent water inflow from overlying strata. A grouting operation will also be required at this level to seal off any fractures that may have formed before completing the shaft into the halite. A rectangular freeze pattern is proposed which encompasses each main shaft and its auxiliary shaft;
- **Sinking 0 to 20 m -** A collar structure will be built using secant piles constructed in a sequential interlocking fashion. The female piles are drilled and cast first at approximately 1 m in diameter on a hit and miss basis. These piles use a low strength concrete and are not reinforced. Then the missing piles are drilled and cast using a stronger concrete mix and steel reinforcing. The piles are arranged in an overlapping fashion, so that the male piles cut

through the female piles resulting in an overlap. This arrangement allows the finished structure to act in compression and does not need any additional bracing;

- **Sinking 20 to 100 m -** The first 100 m will be sunk using a crane and or gantry hoist winding to the surface. Below the secant piles a diaphragm walling will be used as a support system in the near surface material. A shotcrete reinforcing structural shell will be formed by spraying the shaft wall in 1 m advances and fixing the reinforcing as fabric mesh sheets with dowels for continuity. Four shafts will be sunk to this depth as shown in Figure 14-15b;
- **Sinking 100 to 220 m -** To speed up the cycle time the wind will be split into two parts by utilizing the auxiliary shafts. At the base of the two adjacent shafts at -100 m the intermediate level a short length of tunnel will be constructed to connect each auxiliary shaft to each associated main shaft. A long term temporary access arrangement and spoil winding hoist will be set up in each auxiliary shaft. This arrangement will then allow a second gantry hoist to be set up at the base of each main shaft at the 100 m level. This hoist will then be used to sink each main shaft the remainder of the depth. The spoil will be wound to this intermediate level and then transferred by trolley to the auxiliary shaft;
- **Lining -** The permanent lining will be placed once the final shaft depth has been reached. It is proposed that a slip form will be set up at the bottom of each shaft and the concrete lining will be placed at 600 mm thickness from -229 m to -100 m levels and 300 mm thickness from -100 m to collar level. This lining will be placed continuously at a steady rate until the surface is reached. A waterproof membrane may be placed behind the concrete. Concrete will be placed using a skip to give greater control over quality; and
- **Disposal of Waste From the Shaft -** It is estimated that approximately 60,000 m^3 of waste rock and 40,000 m^3 of salt waste will be generated during shaft sinking. The waste rock will be transferred to a temporary stockpile, where it will be reclaimed and used in road and platform construction activities. The salt waste will be placed in a separate area of the lined RoM stockpile pad and will later be hauled to the coast and disposed into the ocean using the waste dispersion system at the process plant during times of plant maintenance and downtime.

14.7.3 Surface Buildings

The top of the shaft will be housed within an enclosure. A steel portal frame shed structure will be built over the shaft site to cover the muck/ore transfer and handling equipment. During construction the building will also house the materials and plant necessary for the supply of the shaft lining (concrete and shotcrete). These structures will be built before shaft sinking starts to keep the entire shaft working area dry as the area is tropical and suffers from heavy and regular rainfall. A concrete hardstand and access road will be constructed outside the shed to ensure that spoil and material delivery vehicles do not become bogged down.

14.7.4 Personnel Lift

Day to day access to the mine for personnel and materials will be via a single compartment lift located in one half of the 12.0 m diameter services shaft. The complete lift and counter weight will be designed and supplied by companies such as Thyssen Krupp. This lift is a form of multi rope friction winder with a counter weight. It generally operates at speeds of 150 metres per minute (mpm), however geared lifts can operate at 60 mpm making travel time down the shaft approximately 4 minutes.

The lift car size has been selected to accommodate the majority of the parts of a CM, a double cab tipper truck, or 150 men. Loads greater than 11.5 t and items larger than the lift internal dimension will be lowered down the shaft using the gantry crane. The use of a large high capacity lift installation would allow most equipment to be driven into the car and lowered to the shaft bottom in one piece. The Winsford salt mine in Cheshire England uses such an industrial lift for a depth of 189 m and has been in operation since 1963.

14.7.5 Vertical Conveying

Material will be removed from the mine using a HAC. The HAC uses the concept where the product to be conveyed is squeezed between two independent belts. The bottom or underbelt is shaped over the top of the product once it is placed on the belt, with normal rollers and specially designed tensioning rollers that conforms the top belt over the product. The edges of the top belt lie against the bottom belt creating the seal. The HAC systems have been used to transfer numerous products including potash for many years.

The HAC system will use a full vertical lift of 235 m at 90 degrees, with the top and bottom (1,600 mm width) belts driven by two 550 kw drives. Two HAC systems, each capable of conveying 1,000 tonnes per hour (tph), have been costed and are included in this PFS to allow for constant vertical conveying ability from the mine to surface. If one conveyor is down or on maintenance, the other will be capable of handling all production from the mine. A full stairway system with landings will be installed adjacent to the HACs. This will allow easy access for any maintenance on the belt as well as provide an alternate means of egress from the mine.

Figure 14-16 shows the HAC proposed for the Sintoukola Project.

14.7.6 Shaft Bottom Layout

The majority of the underground infrastructure will be located on the periphery of the shaft pillars to minimize the quantity of openings near the shaft helping to maintain long-term stability of the shaft pillars. Locating shops farther away from shaft location translates to longer manual development time (i.e using drill and blast techniques) prior to being able to build a CM underground. Minimal openings can be excavated (two rather than three mains) to the shop area and then once a machine is built the access ways can be enlarged with the CM and a third access could be added.

The shaft bottom layout is shown in Figure 14-17. All potential flammable stores/equipment (e.g. diesel/lube bay, electrical substation) are placed between intake and return in a central area (against returns) to minimize risk to the rest of the mine should a fire occur. Three 1200 m^3 storage bins are located on the return side servicing the upcast shaft. Geology sections will need to be checked to ensure that there is sufficient halite thickness to construct the bins.

Using the HAC for transporting ore to surface, a loading pocket is not necessary and ore will be fed from the RoM bunker directly to the HAC.

14.8 Description of Mining Method

Mining will be conducted using a room and pillar mining method. This method allows for the size of the pillars to be adjusted based on ground conditions and the height of the orebody. Mining will be conducted using Sandvik MC470 CMs for both development into the mine panels and for panel extraction. Additionally a smaller machine, the MC250 which mines down to a height of 1.8 m, will assist in low height areas. For this PFS the production schedule was generated using MC470 productivity rates and the economic model assumed a percentage (20%) of the overall production would be mined by the smaller machines with reduced productivity and different operating costs. Proportions of large vs. small machines at the mine should be further evaluated during the FS.

Development mining will be performed as three parallel headings so that separate intake ventilation, conveyor and exhaust ventilation headings can be maintained. The development headings are sized at 4 m height and 8 m wide and will be systematically bolted.

Thus, all main, submain and panel development and panel mining will be done by cutting rather than by drilling and blasting methods. This will lead to good rockmass stability and provide good sizing of delivered ore for processing.

14.8.1 Primary Mining Equipment

CM mining is performed by a series of activities that allow the cutting head to be used for cutting and the gathering arm loader portion of the machine for removing the cut rock and loading it into electric-powered shuttle cars.

The Sandvik MC470 CMs can operate in cutting heights as low as 2.4 m and as high as 5 m. The MC250 has a minimum cutting height of 1.8 m and maximum of 3.6 m. Mining within individual panels will be performed at the desired height as defined by ground and ore control requirements. In ore zones thicker than 5 m the top portion of the ore will be cut first and then the ore remaining in the floor will be mined. The CMs can thus provide highly selective cutting and material separation for ore control and will also provide relatively smooth floor, wall and roof conditions.

Within mains, where bolting is required, the CMs can be advanced into the desired operational face as far as the operator’s cab before the machine will need to be backed out so that rock bolting can be performed. No access between the operators cab and the face should be allowed either while the cutterhead is in operation or beyond ground support. If cutter bits need to be replaced, then the CM needs to be backed up so that this operation can be performed under good ventilation and ground control conditions.

The primary underground mobile equipment fleet consists of CMs, shuttle cars, and feeder breakers. The primary equipment is listed in Table 14.8.1.1.

Table 14.8.1.1: Primary Underground Equipment

Equipment	Type	Quantity
Sandvik MC 470	CM	6
Sandvik MC 250	CM	3
Sandvik TC 790	Shuttle Car	18
	Feeder Breaker	9

14.8.2 Underground Haulage

Ore haulage from the CM at the face to the feeder breakers and conveyors, will be via electrical powered shuttle cars with a capacity of 18 t. Shuttle cars such as the Sandvik TC 790 have a power cable reel with a maximum capacity of 280 m of cable. The on board ore storage capacity of the Sandvik CM MC 470 will allow the use of two shuttle cars per miner. The CM and two shuttle car scenario will be capable of achieving the desired production rate of 2600 tpd per CM.

14.8.3 Ancillary Underground Mine Equipment

The primary underground mobile equipment fleet consists of CMs, shuttle cars, and feeder breakers. There will be one primary operator and one helper assigned to each CM. Two shuttle cars with one operator each are assigned to each CM as well as one feeder breaker. There is one operator assigned to each feeder breaker.

In addition to the primary production equipment there is a fleet of ancillary equipment that is required to support the production operations for the underground mine. Table 14.8.3.1 summarizes the proposed ancillary equipment.

Table 14.8.3.1: Ancillary Underground Equipment

Equipment	Type	Quantity
Sandvik DS 311	Rock Bolter	2
Terrapro Mantrip	Mantrip	9
Getman A-64 Lube / Mech.	Fuel / Lube Truck	5
Getman A-64 Crane	Utility	2
Terrapro Utility	General Purpose	9
Getman Roadbuilder	UG Grader	2
CAT 272D	Skidsteer	4
Sandvik LH 514	UG Loader	4
Getman A-64 Water Truck	Water Truck	2
Sandvik DE130	UG Core Drill	3

One of the DS 311 roofbolters services each of the two main areas of the mine. Only the development mains are systematically bolted.

The RDG 1504 motor grader is used to maintain the main roadways. The water truck is a water tank and water spray system mounted on the Getman A64 carrier. A limited amount of water will be used due to the soluble nature of potash ore. The fuel/lube vehicle, utility, power cable reel handler, and utility crane configurations are all installed on the Getman A-64 carrier. This allows for a significant reduction in spare parts required due to the interchangeability of using the same carrier.

14.8.4 Surface Mobile Equipment and Shops

Surface facilities at the mine site are described in detail in Volume VIII (EGIS, 2012). Main components on surface include ore loading, logistics and transfer, maintenance areas, and administration offices.

Surface machinery at the mine site will consist of the following:

- Loader – CAT 998;
- Ambulance;
- Forklift (maintenance);
- Skid Steer loader (bobcat style);
- 5 t Carry Deck; and
- 20 t rubber tire mobile crane.

The CAT998 will be used to move material out of the stockpile as necessary. Some months require material movements of approximately 1,500 tpd to meet process plant requirements. This can be further optimized at the FS stage and a smaller loader may be suitable.

14.8.5 Underground Conveying

Underground belt conveyors will be used for ore transportation in all areas of the mine. The belts are distributed in the mains and submains and ultimately in the working panels near the CM working face. As previously described, the ore is placed on the belts from the feeder breakers supplied by the shuttle cars. The conveyors will carry ore loaded by the feeder breakers to the storage and transfer facilities located near the production shaft to be conveyed to surface. The material handling rate required is based on the calculated rate of 2,600 tpd for each CM with an overall production requirement of 570 kt per month. The conveyor belts are installed as required based on the development and production advance. The belts are identified and summarized in Table 14.8.5.1 and shown graphically in Figure 14-18.

Table 14.8.5.1: Conveyor Belt Summary

Belt Type	Area	Belt ID	Length (m)	Expected Capacity (tph)	Expected Operating kW	Maximum Capacity (tph)
Mains	North West	C1	830	390	37	1,200
		C1-1	3,190	390	149	1,200
		C1-2	2,300	390	112	1,200
		C1-3	370	390	22	1,200
		C1-4	2,530	200	112	1,200
		C1-5	690	200	37	1,200
		C1-6	890	200	37	1,200
		C1-7	1,815	200	93	1,200
		C1-8	3,440	200	149	1,200
		C1-9	870	200	37	1,200
		C1-10	600	200	37	1,200
		C1-11	1,740	200	75	1,200
	West	C2	1,800	390	75	1,200
		C2-1	2,520	390	112	1,200
		C2-2	2,560	390	112	1,200
	South East	C3	3,680	390	149	1,200
		C3-1	1,750	390	75	1,200
		C3-2	3,980	390	149	1,200
		C3-3	3,980	390	149	1,200
		C3-4	1,380	390	56	1,200
Total Mains			**40,915**		**1,775**	
Panel	9 –1 per CM		5,000	390	280	800
All Belts Total			**45,955**		**2,055**	

The belt width selected, 1,200 mm, can support the production requirements at modest belt speeds of 1.0 metres per second (m/s). At this speed the nominal belt capacity is 800 tph which is twice the requirement. The main conveyors could be run at 1.5 m/s to provide a maximum capacity of 1,200 tph as shown in Table 14.8.3.1. If additional capacity is needed, the belt speed could be modified, however with belt speeds higher than 1.5 m/s dust generation may become an issue.

The expandable panel belts and feeder breakers are installed near the production or development face to allow a minimum distance for the shuttle cars to operate from the CM to the feeder breaker. The conveyor belt is stored in a specifically designed storage cluster and fed out as required. The panel belt feeds the ore onto the sub main belt which feeds the ore to the main belt and ultimately to the ore storage / transfer facility at the bottom of the shaft.

14.8.6 Surface Stockpile and Haulage

Since the mine development starts six months before the process plant has been commissioned, a lined surface stockpile area was designed to store up to 3 Mt of RoM material. The base of the pad is required to be level, and a 2 m-high perimeter bund is incorporated to temporarily contain any runoff from the RoM material. The perimeter bund is also required to anchor the geotextile and High Density Polyethylene (HDPE) liners. The levels of the pad designs have been designed to balance the cut to fill material to avoid bringing in additional material.

A runoff storage facility has been sized to an area of 22,500 m^2 (150 x 150 m) to store the runoff from the RoM stockpile. The facility is required to be lined to prevent seepage of the contact water. Estimated total water in the storage facility over three years is approximately 452,000 m^3.

Timing of feeding material from the stockpile to the process plant will need to be optimized once more accurate construction times are established. If the process plant is not on-line before the mine is in full production the material in the stockpile may be left to the end of mine life before reclaiming. If however the process plant is ready before full mining ramp up is completed then the stockpile material can be fed to the process plant in the early years. Additionally, material from the production schedule can be balanced with stockpile feed to provide the process plant with a constant feed tonnage.

With current mine production and process plant operating schedules, the maximum stockpile size necessary is 1 Mt of RoM, and therefore the stockpile costs will be factored in the economic model, without the designs being updated at this time. It is planned to reclaim material from the RoM stockpile using a CAT988 front end loader.

During standard production operations, RoM ore conveyed to surface is conveyed from the headframe bin to a 16,000 t stockpile which is located in a domed structure. The ore is reclaimed from the ore stockpile and conveyed to a truck loading surge bin. From the truck loading surge bin the ore is loaded onto road trains that are transported to the process plant located approximately 36 km away. If there is no production from the shaft surface stockpile material can be loaded onto road trains until such time as the mine production recommences.

The production schedule shown in Section 14.7.2 does not model the stockpile discussed here. This is accounted for and tracked in the economic model.

14.8.7 Mining Risk Categories

As the deposit is presently unmined with widespaced drilling, more operational scale resolution of the mineralisation will be achieve through underground exploration. There are several factors/items that have been identified in areas that give more uncertainty, and therefore risk, to a particular area. In order to address these areas of potential concern SRK has classified the areas of the design into risk categories:

- Distance to anhydrite: Though a 15 m buffer has been left, there is uncertainty associated with how these surfaces were interpreted. Either anhydrite, mining surface, or both could be several meters off in either direction therefore making the buffer from mining to anhydrite thinner or thicker than expected. Areas where thickness of the buffer is greater, have less risk as they can tolerate several meters of error and still provide a sufficient buffer;

- Thick mining areas: Potential displacement or deformation of units above the mining horizon is a function of mining height. Typically one can expect to see ½ the mining height displacement in 10 years after mining. This includes mining on both the LS and US, and in areas where both are mined the thickness of USS and LSS mining must be summed to determine full mining height. The geotechnical design parameters do adopt a low extraction ratio that reduces displacement. Areas where mining height is thicker are associated with more risk; and
- Disturbance areas: Though individual disturbance areas have been identified, there is no known correlation of disturbance across and between seismic lines. With current data an assumption has been made that areas with more identified disturbances carry a higher risk.

Based on the above categories, the design was classified into low, medium, and high risk areas as shown in Figures 14-19a through 14-19b. For purposes of reserve estimation and cashflow modelling the risk categories are treated the same. When creating the production schedule an effort was made to schedule mining in lower risk areas first.

Encountering unexpected carnallitite is another risk which will be minimized by advanced drilling underground.

There is little that can be done at this time to alleviate these risks, however, underground drilling in advance of mining and ensuring extraction ratios and mine plans are adhered to will lower these risks.

14.8.8 Access to Lower Seam Mining

Mining from the USS to the LSS will be done using the same equipment and methods as mining the USS and LSS. The vertical distance between the USS and LSS is approximately 5 m and therefore horizontal development distance through halite will be approximately 34 m assuming a ramp grade of 15%. This waste material can be gobbed (stored) underground if it is convenient or can join the ore on conveyors, as it is a minimal amount and the process plant will not see a notable difference in grade due to this material.

It is expected that this initial decline to the LSS will be slow to develop as a brow would need to be created. The brow would be approximately 3 to 4 m in vertical height, so mining of the floor will take place to such a point. Then the brow would be appropriately supported prior to continuing mining. Additionally, areas before the ramp will need to be supported as well to help ensure long term stability.

Once on the LSS level, mining activities will be similar to those undertaken on the USS with panels being mined on retreat using primary/secondary/tertiary methods.

14.9 Mine Ventilation

Ventilation and climatic modelling was completed by Mine Ventilation Services (MVS) engineers for the Sintoukola Project. Both a ventilation model using VnetPC Pro+ package and a climate model using CLIMSIM were completed.

Ventilation model work used two shafts, one that serves as intake (services shaft) and one that serves as exhaust (production shaft).

14.9.1 Airflow Requirements

The minimum recommended air velocity for an area with workers in a drift is 0.30 m/s. Based on the minimum velocity requirements and drift dimensions, the minimum quantity in a drift should be 12 m^3/s. With the long distances from the shaft to the face and back it was found to be difficult to have more than three parallel entries in the mains. Therefore, to maintain minimum velocities, a three main system was used. One entry is designed to be a single intake drift. The other entries are designed to have dual common return with one of the entries containing a conveyor belt.

The ventilation system is planned as an exhausting system. With the fan located at the top of the return shaft, it will ensure that no air in the mine will be recirculated due to leakage. This design also removes the significant heat-load from the main fan/motor installation from impacting conditions underground.

14.9.2 Time Phase Modelling

Four staged models were completed for this study for production years 1, 5, 10, and 20. The models were completed to accommodate nine working panels.

<u>Year 1 of Production</u>

In the first year of production mining is rapidly making progress away from the shafts with an emphasis on mains development. The total fan pressure at the shaft collar is estimated to be 1.41 kPa with a total airflow of 258 m^3/s.

<u>Year 5 of Production</u>

In the fifth year of production mining is focused in the panels, rather than mains development. The total fan pressure at the shaft collar is estimated to be 1.40 kPa with a total airflow of 257 m^3/s.

<u>Year 10 of Production</u>

In the tenth year, production in the panels has spread much further away from the shafts. This results in a large increase in the total fan pressure and quantity from the Year 1, and Year 5 models. This is due to the long distances that the air must travel and from the additional airflow leakage that occurs. The total fan pressure at the shaft collar is estimated to be 2.95 kPa with a total airflow of 328 m^3/s. The pressure on the bulkheads can reach up to 1.4 kPa, therefore it is recommended that the mine uses block stoppings that are capable of handling higher pressures and typically provide a higher total resistance resulting in less leakage.

<u>Year 20 of Production</u>

In the 20th year of production, mining continues with an emphasis on the panels in the northern part of the mine, while mining of panels in the southern part of the mine is being completed closer to the shafts. It is assumed that areas where panels have been completed are removed from the ventilation circuit (sealed). For these reasons the operating pressure of the fan is similar to the Year 10 model. The total fan pressure at the collar is estimated to be 2.95 kPa with a total airflow of 274 m^3/s.

14.9.3 Fans

A comparison of the fan duties, installed motor powers, and annual costs for each modeled development stage is provided in Table 14.9.3.1. The installed power(s) were calculated with an assumed fan efficiency of 75%.

Table 14.9.3.1: Fan Operating Points

Location	Pressure kPa	Quantity m3/s	Air Power (kW)	Motor Power* (kW)
Year 1	1.41	258	364	485
Year 5	1.40	257	360	480
Year 10	2.95	328	968	1,290
Year 20 (LoM)	2.95	274	808	1,078

*Assuming 75% Fan Efficiency

The fan at the mine will need to be able to operate at a wide range of pressures because of the rate at which the distance of the faces from the shafts expand. For this reason MVS engineers recommend the use of VFD motors. This will increase flexibility as the ventilation requirements change over time. The lowest operating pressure from the models is in year 5 with 1.40 kPa and a highest pressure at 2.95 kPa in year 10.

14.9.4 Climatic Studies

With the relatively warm ambient and virgin rock temperatures a climatic study was completed to estimate temperatures throughout the mine to ensure safe working conditions. A value of 28.9ºC was used as a basic rock temperature based on the average temperatures collected from drillhole samples from the site. The geothermal gradient was estimated to be 1 °C per 50 m for this study. It was determined that mine cooling would be required.

14.9.5 Mine Refrigeration

There are two methods of cooling the mine. The first involves cooling the water supply which is used at the faces and the second is bulk and/or localized air cooling. Cooling of water used underground is more efficient than air cooling, however with relatively low water amount in use for potash operations a combination of the systems will be necessary. A water limit of 1l/s per machine was applied to underground water cooling based on operating experience at other potash mines.

Spot coolers near working areas of the mine can also be utilized to cool air in a local area. These can be moved when necessary to follow equipment/working areas as the mine progresses. A power allotment of 4 MW has been provisioned for cooling purposes in the mining cost model.

With the possibility of larger shaft diameters using the civil type approach, the ventilation design should be re-evaluated and optimized for the next level of study to minimize the need for cooling.

Additionally if the mining areas could be scheduled closer together that will minimize ventilation requirements and provide less area for leakage.

14.9.6 Face Ventilation

In each working area, it was determined that there would be the need for two auxiliary fans underground. A small fan, approximately 10 kW (9 m^3/s), will be used to direct fresh airflow into dead-end headings where roof bolting or other activities are being performed prior to a ventilation breakthrough or connection. This fan is sized to provide the volume demanded by the diesel engine used to move the rock bolter even though the actual bolting will be done with electric/hydraulic units. It is assumed that this fan would be placed in the fresh air supply and that vent ducting would be advanced into each heading requiring temporary ventilation.

CMs will generate significant dust, even with water sprays. Production panel fan sizing was based on the assumption that when a CM is operating, two electric shuttle cars, and the feeder breaker would also be in that heading which drove the calculation of the required volume.

The concept uses a 56 kW fan pushing air from the mains via air duct tubing to the mining face. Air is pushed across the face and exhaust air is blown out of the entry.

Intake and exhaust airways are separated by stoppings, overcasts, underpasses, regulators and other control mechanisms. These structures are used to direct fresh air to the working area and exhaust air that contains dust away from the working areas.

Half of the required air will be forced from the face fan while the other half is generated from the surface-mounted exhaust fans. Leakage through the stoppings installed in each mining panel will report to the return air ways. Using this method, access roadways can be kept in intake air and provide for a better environment for the employees and equipment.

14.10 Mine Staffing

The mine operations staffing requirements are summarized in Table 14.10.1. Numbers shown here include a 10% labor factor to account for absenteeism. The numbers reflect the mine staff after 3 years of operation where initial expatriate personnel are replaced by hourly personnel.

Table 14.10.1: Underground Mining – Total Personnel

Labor Category	Positions	Required Staff*
Salaried / Expatriate		
Mine Management	17	17
Hourly/Local		
Underground/Surface Operations	367	489
Underground/Surface Mine Maintenance	86	115
Total Hourly/Local	**453**	**604**
Total Labor	**470**	**621**

*Required staff is based on 4 crews to fill 3 shifts per day

The Sintoukola Project will require well trained underground miners, maintenance personnel, support staff and a technical management group that is familiar with all aspects of underground soft rock room and pillar mining techniques using CMs, conveyor haulage systems and vertical haulage systems.

An extensive on-the-job training program using experienced expatriate management teams will be utilized to design and implement the proper educational and training programs for the local in-country workers. These programs will develop trained and experienced miners that will be required for the long term, underground mine operations. A rigorous safety training program will also be implemented in order to support the continuous improvement program requirements for a modern mining operation.

Once mining operations are well established, approximately 80% of the salaried positions will be replaced by local hourly personnel. Personnel will need to be highly trained in order to ensure continued streamlined operations. The personnel switch from expatriate to local personnel is accounted for in the mining cost model after four years of operations.

14.10.1 Salaried/Expatriate Labour

The direct management group for the underground operation is summarized in Table 14.10.1.1. A qualified and experienced management and technical group will be required to complete the final design, implementation and manage the underground mine development and production plans to meet the ore production requirement. Ore grade control will be an integral part of this plan and the ore control/geological group will be an integral part of this team.

Table 14.10.1.1: Underground Mining - Salaried/Expatriate Personnel

Labor Category	Positions: First Three Years	Positions: Remaining Years
Mine General Management Labor		
Mine Manager	2	1
Mine Superintendent	2	1
Mine General Foreman	4	2
Mine Supervisor	12	0
Mine Trainer	4	2
Safety Officer	4	2
Engineering & Geology		
Chief Engineer	2	1
Ventilation Engineer	4	2
Rock Mechanics	4	2
Senior Mine Planner	2	0
Senior Surveyor	4	0
Chief Geologist	2	0
Mine Maintenance		
Maint. Superintendent	2	1
Maint. General Foreman	4	2
Maint. Supervisors	10	0
Maintenance Planner	2	1
Total Salaried/ Expatriate Personnel	**64**	**17**

Once mining operations are well established, approximately two thirds of the salaried positions will be replaced by local hourly personnel. Personnel will need to be highly trained in order to ensure continued streamlined operations. The personnel switch is accounted for in the mining cost model.

14.10.2 Hourly/Local Labour

The hourly crew schedule is based on a total of four crews with three 9-hour shifts per day with seven days per week operating coverage. The effective hours for each shift will allow for a hot change at the working face that will allow maximum utilization of the equipment, especially the CM operations. The nine hour shifts include allowances for lunch, breaks, and travel time to the working faces. The CMs will be staffed with a primary operator and a helper to allow for continuous operations during lunch and other breaks. A 28 day rotating schedule with a total of four crews has been applied. Each crew will work seven days on shift (i.e., days) and have two days off. Each respective crew will report back to work for the next successive shift (i.e., swing) and work seven days with two days off. Each crew will report back to work at the next successive shift (i.e., nights) and work seven days and then have three days off. This example crew will then report back to work on the day shift. There is a complete rotation of work and days off every 28 days. Hourly staffing will vary over time as a function of the planned production tonnages with a ramp up in production over the first year. Staffing and job titles are summarized in Table 14.10.2.1 for the maximum hourly staffing requirements at full production.

Table 14.10.2.1: Underground Mining – Production Crew Summary

Labor Category	Positions Per Day: First Three Years	Positions Per Day
Mine General Management Labor		
Local Mine Manager	0	1
Local Mine Superintendent	0	1
Local Mine General Foreman	0	2
Local Mine Supervisor	0	12
Local Mine Trainer	0	2
Local Safety Officer	0	2
Engineering & Geology		
Local Chief Engineer	0	1
Local Ventilation Engineer	0	2
Local Rock Mechanics	0	2
Local Senior Mine Planner	0	2
Local Senior Surveyor	0	4
Local Chief Geologist	0	2
Mine Planning Tech.	6	6
Geologist Tech.	8	8
Surveyor Tech.	10	10
Production Labor		
Production Operator	3	3
Skip Tender	0	0
CM Operator	30	31
Shuttle Car Drivers	59	59
Bolter Operators	7	7
Helpers	40	41
UG Support Labor		
Lube / Mech. Truck Operator	17	17
Power Cable Truck Operator	7	7
Utility Truck Operator	30	30
UG Grader Operator	7	7
Skidsteer Operator	14	14
UG Loader Operator	18	18
UG Water Truck Operator	7	7
UG Drill Operator	10	10
Helpers	27	27
UG Services Labor		
Construction	12	12
Belt Operator	12	12
Rehabilitation	4	4
Tool/parts room	4	4
Mine Maintenance		
Local Maint. Superintendent	0	1
Local Maint. General Foreman	0	2
Local Maint. Supervisors	0	10
Local Maintenance Planner	0	1
Mechanic - Shop	16	16
Mechanic - Field	8	8
Mechanic - Production Lift	8	8
Electrician -Shop	8	8
Electrician - Field	16	16
Helpers	16	16
Total Hourly Labor	**404**	**453**

Note: Numbers here reflect production after 3 years of operation where expatriate personnel have been replaced by local hourly labor. All hourly positions require 4/3 as many total people due to the requirement of 4 crews to fully staff 3 shift per day. This has been accounted for in the mining costs.

The average hourly pay rate including burden and shift differential allocations for the various job descriptions and responsibilities is summarized as follows:

- Experienced equipment operators for the CMs, bolter operators, LHD operator, road grader, and hoist men are paid; US$8.95 per hour;
- The shuttle car operators, feeder breaker operators, conveyor operator, mine services, hoist – oilers, and similar skill sets are paid; US$5.38 per hour;
- The helpers, operators in training, and others are paid; US$2.95 per hour;
- Experienced lead mechanics, electricians, welders, and machinists for the underground and surface positions; US$8.95 per hour;
- The 2nd mechanics, 2nd electricians and similar skill sets is; US$5.38 per hour;
- The helpers, operators in training, and others is; US$ 2.95 per hour; and
- Senior office based staff are paid US$20.60 per hour.

14.11 Support Services

14.11.1 Mine Dewatering System

A provision has been made to pump water from the mine. Two sources of water have been identified:

- Standard mine water: wash water, ground water, and meteoric water seeps from the mine shafts; and
- Emergency inflow water: inflows from the overlying aquifers resulting from a potential breach of the salt back.

Water from the shafts is collected in the shaft bottom sumps and pumped periodically as needed to keep sump water levels at an acceptable level. Water from the shaft sumps would be pumped using a submersible pump to a nominally sized holding tank at the main level. A 250 HP centrifugal pump will then pump the water to surface in a 2" line installed in the shaft to surface water management and treatment facilities.

Should an inflow occur from a breach in the salt back, water will need to be collected as close to the source of the inflow as possible. This may prove to be problematic if an inflow occurs in an area of the mine which is not currently active. An inflow in an abandoned panel could go undiscovered and therefore regular patrols throughout the abandoned areas are planned.

Water/brine would be collected in local sumps and then pumped to the main sump and pumping station in the central area of the mine. A second 250 HP pump will be used in addition to the standard water pump to move water out of the mine via a 6 inch diameter line pre-installed in the upcast shaft. This gives the capacity to move 18 m^3/min. If additional capacity is needed in an emergency situation another line and pump would need to be installed.

Brine pumped out of the mine will be stored in the lined surface impoundment constructed for the RoM stockpile. This provides several days storage to allow for alternate brine disposal solutions to be implemented. Options under investigation for brine solution storage in an emergency situation are:

- Brine injection into the Coco beach formation; and
- Pumping via surface pipeline to the coast for marine disposal.

Brine disposal into the Coco beach formation is being investigated as part of the FS level studies.

14.11.2 Underground Electrical Systems

Electrical power will be used for CMs, shuttle cars, feeder breakers, ventilation fans, pumps and conveyor belts. A backup cable will also be installed down the production shaft. It is estimated that the underground portion of the mine will have a total connected load of 15.1 MW and an estimated operating load of 10.4 MW as shown in Table 14.11.2.1.

Table 14.11.2.1: Underground Electrical Power Requirements

Description	Unit Hp	Units	Total Hp	Total kw	Load Factor	Load kw	Utilization Factor	Kwhr Per Month
Vent Fans	25	9	225	168	0.95	159	95%	109,100
CM MC-470	1,250	8	10,000	7,460	0.70	5,220	70%	2,631,900
CM MC-250	830	1	828	618	0.70	430	70%	217,900
Shuttle Cars	197	18	3,550	2,650	0.70	1,850	70%	933,300
Feeder Breaker	175	9	1,575	1,180	0.70	820	50%	297,000
Panel Conveyor	75	9	675	500	0.80	400	50%	145,000
Main Conveyors	2,755	1	2,755	2,055	0.60	1,233	50%	443,930
Dewatering Pumps	250	2	500	370	0.50	187	0.5%	670
Other	200	1	200	150	0.75	112	90%	72,480
Estimated Total			**20,300**	**15,150**		**10,420**		**4,850,000**

The mine power allocation for the surface high voltage distribution will deliver power to the HAC system, surface conveyors, men and material lift, Alimak lift, chiller unit and main ventilation fans at the respective locations on the surface. The total connected load for these surface facilities is estimated to be 6.9 MW with an estimated operating load of 6.5 MW as shown in Table 14.11.2.2.

Table 14.11.2.2: Surface Electrical Power Requirements – Allocated to the Mine

Description	Unit Hp	Units	Total Hp	Total kw	Load Factor	Load kw	UtilizationFactor	Kwhr Per Month
Main Vent Fans	1730	1	1730	1,290	0.95	1,230	100%	882,360
HAC System	740	2	1,475	1,100	1.00	1,100	95%	752,400
Men / Material Lift	420	1	420	310	0.80	250	60%	107,000
Emergency Hoist	40	1	40	30	0.80	24	5%	860
Surface Ore Handling	20	2	40	30	0.80	24	75%	12,960
Chiller Unit	5,360	1	5,360	4,000	0.95	3,800	75%	2,052,000
Other surface	150	1	150	110	0.85	95	95%	65,000
Estimated Total			**9,217**	**6,870**		**6,517**		**3,873,000**

The mine electrical power load from the surface facilities and the underground facilities have an estimated total connected load of 22.0 MW and an estimated total operating load of 16.9 MW. The use of soft start equipment on the larger electrical motors as well as the use of mine automation software will allow for the more efficient use of power to reduce operating costs.

Power distribution from the main underground substation will be stepped down to 4,160 volts, 3 phase and 50 hertz for the underground mine distribution to the nine mobile power centers (MPC) strategically located near the production operations. There are a total of nine mobile power centers (MPC) with each MPC designed to operate the CM, two shuttle cars, feeder breaker, panel belt and associated auxiliary ventilation fan for each production unit. Each mobile power center will also allow additional section feeder cables to be connected through the MPC to allow a power distribution grid to be established and expanded as the mine production areas are worked. The electrical load for each MPC of 1,525 kW was estimated by combining all the electrical equipment associated with each MPC. The skid mounted MPC contains the required transformers, circuit breakers and cable connectors to supply each section's estimated power load

Main and sub main conveyors, additional ventilation fans, and other stationary equipment, such as pumps, ore feeders, ore loading facilities at the bottom of the shaft, as well as the underground shop, will be powered from transformers and motor control stations located near the main transformer and substation at the bottom of the shaft as well as the conveyor substations located throughout the mine.

14.11.3 Communications

An underground leaky feeder communication system will be installed and advanced in all areas with the development and production headings. It is recommended that communication is available in all areas of the mine, which are accessed by underground personnel. The communication system will also tie to the operations and safety office on the surface and serve to improve the efficiency of the operation and as an important emergency contact and notification system.

The system will also be tied to the individual hard hats as part of the personal protection equipment (PPE) for all underground workers. Stationary hard wired phones will also be installed in strategic locations throughout the mine as well in the lunchrooms and refuge chambers.

14.11.4 Workshops

Extensive underground workshop capacity and mobile maintenance capacity will be provided in order to repair breakdowns, carry out regular planned maintenance as well as major equipment overhauls as required. The underground workshop area will be fitted with an overhead bridge crane with a capacity of 50 t. Numerous well lighted work bays will be provided to allow the completion of the required maintenance in a timely and efficient manner. A small stores warehouse will also have routine spare parts in inventory. Larger, less frequently used repair items will be stored at the surface warehouse to be supplied as required. Office, lavatory and lunchroom facilities will also be constructed as part of the underground shop facility. Underground storage of hydrocarbon based fluids (diesel, lubricants, hydraulic fluid, etc.) will be stored in large specifically designed totes that will be lowered using the service hoist / cage as required. Spill containment will also be set up in a central location where the totes are stored.

14.11.5 Mine Automation Systems

The extensive and complex nature of the underground working will require a sophisticated level of operations management to allow for an efficient low cost operation. The significant number of operators and mechanics as well as the CMs, shuttle cars, feeder breakers, conveyors, production hoisting, ore grade control, worker and supply hoist schedules and the required predictive and breakdown maintenance schedules for the equipment will require a detailed monitoring and management plan. New auto mine hardware and software products are available to be used as a tool to facilitate the management of the underground mine operations.

The effective use of the AutoMine system may also lead to the testing and installation of systems that could lead to an autonomous production system whereby the CM and conveyor loading systems are not manned except at a centrally located control room with a dispatch system. It is suggested that additional review and discussions with manufactures (Sandvik) and other mine operations be reviewed to allow for an independent analysis of the available alternatives.

For the purpose of PFS this has not been costed but noted for investigation during the FS stage.

14.12 Health and Safety Considerations

Health and safety considerations for the underground mine go beyond the production, ground control and ventilation requirements discussed in prior sections. The workplace expectation is that using industry good management practice system to manage a productive, safe and healthy work environment will be a priority for the entire mining group. The system will require strong management participation to insure the proper work environment from the top management down through to the work force is implemented. Proper operator training as well as continuous improvement programs will be an integral part of the system.

14.12.1 Refuge Stations

Mine refuge chambers are priced for inclusion in each working area with a total of three mobile chambers plus a permanent station constructed near the main warehouse / shop area at the bottom of the shaft. The mobile chambers are designed to hold a total of 25 workers each for a total of over 75 workers that could be accommodated. These semi-portable units (2.4 m height by 2.3 m wide by 9.7 m long with a weight of 7400 kg) are designed to provide miners with shelter, air, food, water and communications in the event of a mine emergency such as a fire, flooding, ground fall, earthquake or other disaster.

The units consist of a prefabricated skid or wheel mounted chamber equipped with first aid, air filtering and oxygen replacement equipment, seating, food and water, toilet facilities and communications and are typically designed to operate for a minimum of 96 hours of emergency use.

The permanent facility near the shaft station is designed to house 75 miners with the same facilities as the mobile chambers. A small mined out area will be bulk headed off with a sealable door. The inside of the chamber is sprayed with shotcrete and set up with the provision requirements, phones, power and auxiliary air supply. The total capacity of 150 miners housed in refuge chambers underground is sufficient for the potential maximum number of miners that could be underground at any one time which would be at shift change.

14.12.2 Emergency Planning

The mine operators will develop an Incident Command Plan including procedures for the management of:

- Cave-ins;
- Rock falls;
- Fires;
- Floods;
- Accidents on roads; and

- Other incidents that cause or can cause danger to personnel, and/or installations.

Objective of plan:

- Save lives and avoid injuries;
- Reduce the damage to crews and installations;
- Establish an effective system of information exchange and internal/external communication;
- Re-establish normal conditions to the affected area; and
- Assure the continuity of the operations.

14.12.3 Mine Emergency Warning Systems

The use of “leaky feeder” wire antennas and communications systems will be used for an early warning system. These systems utilize a mine wide network of radio cables that allow for the signal to leak out of the cable so that it can be picked up anywhere within line of site of that cable. It allows for communications from a surface or underground hub and allows for normal radio communications between people, equipment, and also allows for remote monitoring and control of equipment such as fans, pumps, air and water monitoring devices, etc. The system will also have a notification component associated with each miner’s hard hat.

14.13 Mine Infrastructure

14.13.1 Lunch Room

Each section will have a central area with bottled drinking water, eating areas, communications station, first aid supplies, small tools and meeting area.

14.13.2 Sanitary Waste

Chemical toilets will be provided underground for sanitary waste. A contractor will service these units.

14.13.3 Water Distribution

Industrial quality water will be used underground for dust control at the CM operating faces, on the roof bolter, on the feeder-breaker, for roadways and on conveyor transfer points. A 50 m^3 head water tank is included at the shaft area on the surface. Each CM operating section has the same distribution system that delivers water to the CM, feeder breaker, conveyor transfer points and the roof bolter. The CM has an onboard booster pump that connects into the integral cutter head spray system.

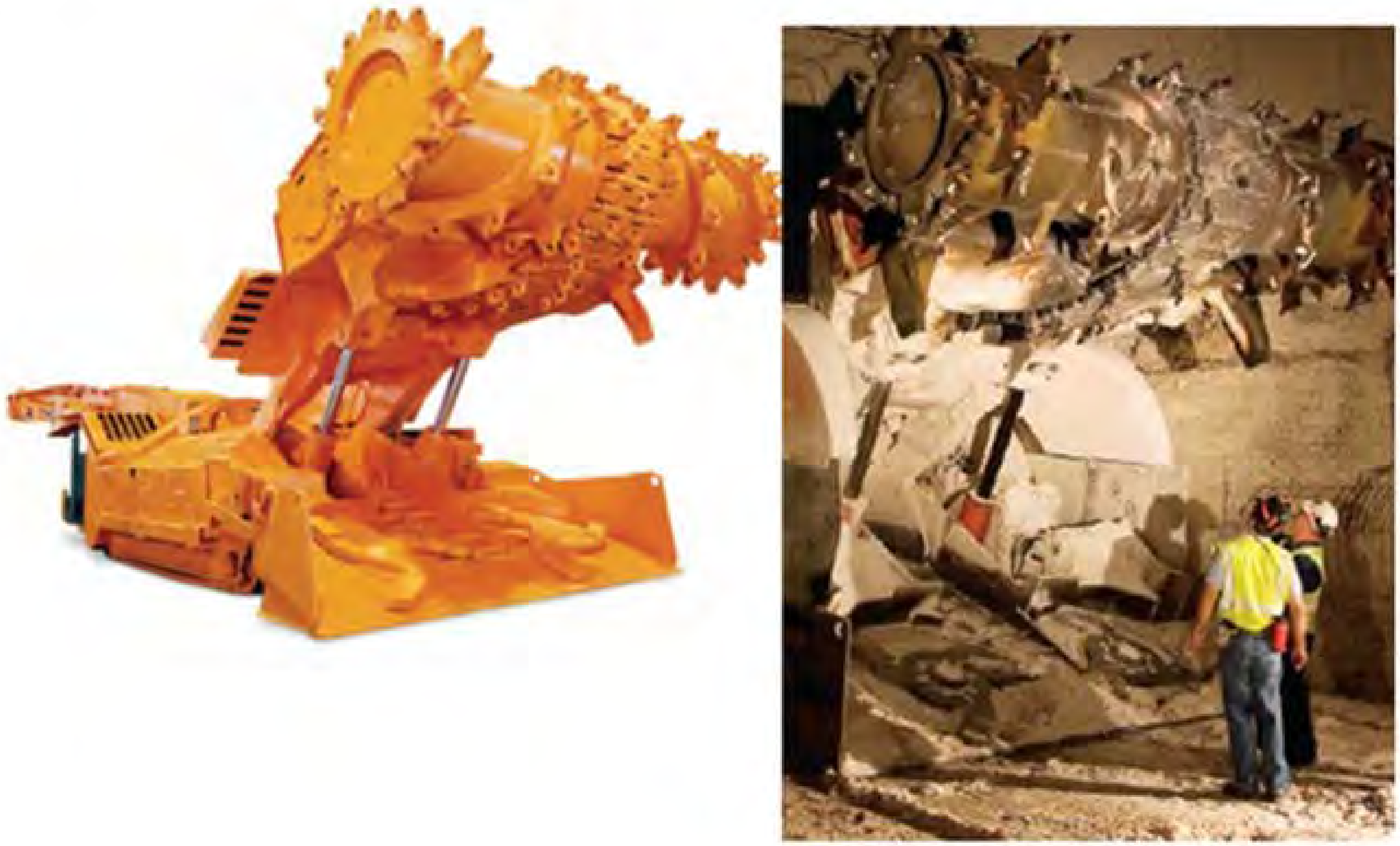
Once water is used it will evaporate or collect in a sump near the shaft and be pumped to surface. It is expected that the majority of the water will evaporate prior to reaching the sump area.

14.13.4 Fire System

Fire fighting stations are located at the underground shop, near all conveyor transfer points and at each operating face. Firefighting equipment will include valve manifolds, lay-flat hose and spray nozzles. Automatic dry chemical fire suppression systems will be supplied with each CM, shuttlecar and feeder breaker. Dry hand held fire extinguishers will also be provided on the mobile equipment as well as at strategic locations throughout the mine for electrical or other remote requirements.

14.13.5 Underground Supplies

Supplies will be brought underground from the surface warehouse to the underground warehouse located near the shop area. Normally consumables include roof bolts, grout, cutter bits, ventilation duct, and water hose. Diesel, lubrication oil, and hydraulic oils will be shipped to the mine in specially designed totes that allow for easy transfer with forklifts from shipping trucks to the supply cage and lowered to the underground shop area. The underground shop area has a specified area for the totes with a full containment system installed. The respective totes are hooked up to a manifold system to allow easy transfer of the contents to the required areas in the shop as well as the mobile lubrication trucks for distribution throughout the mine as required. Material will be transported using Terra Pro underground utility vehicles, crane trucks or LHD's depending on size and weight.



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Republic of Congo

Source SRK, 2012

Figure 14-1

CM and Conveyor System

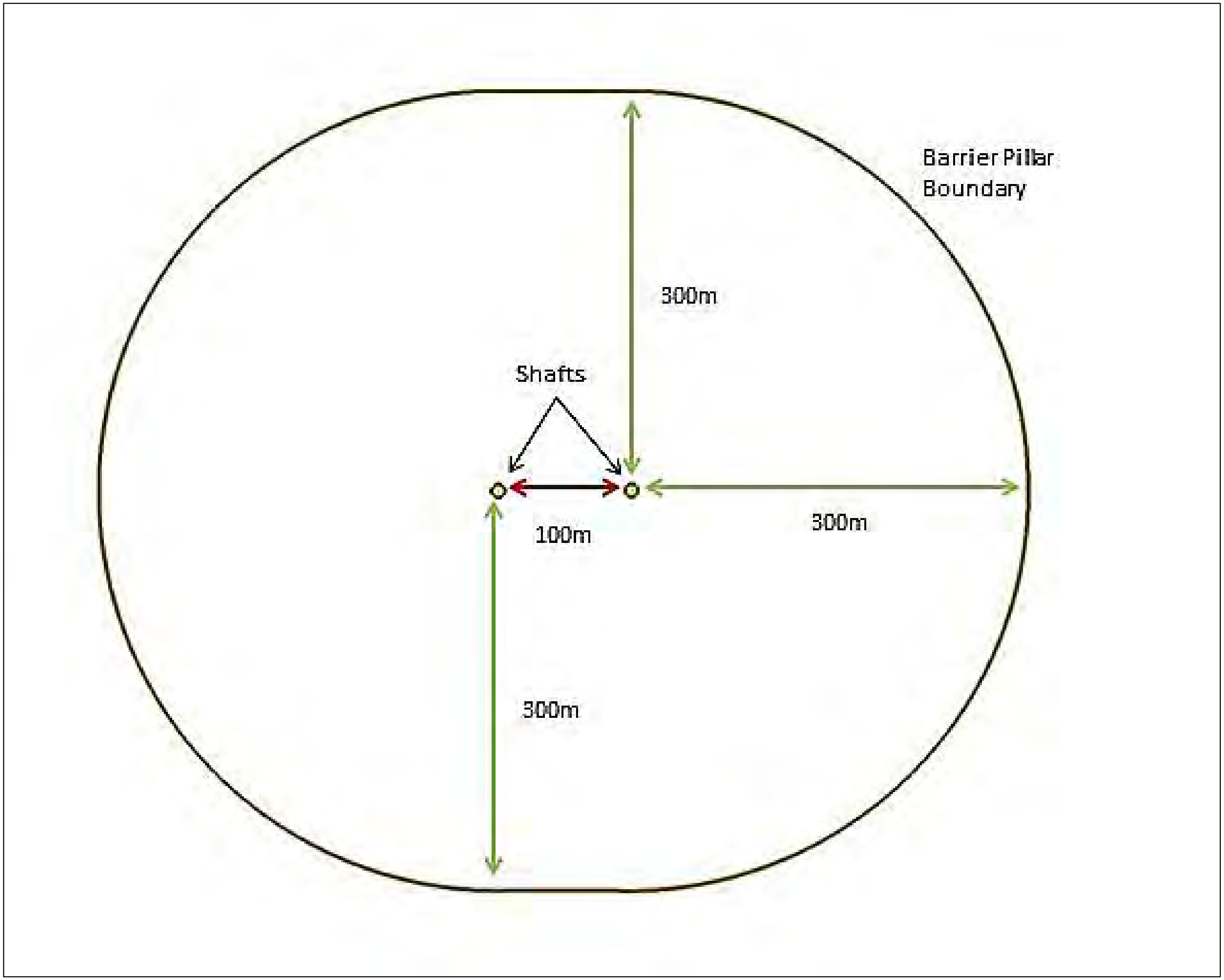

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Figure 14-2

Shaft Pillar Orientation

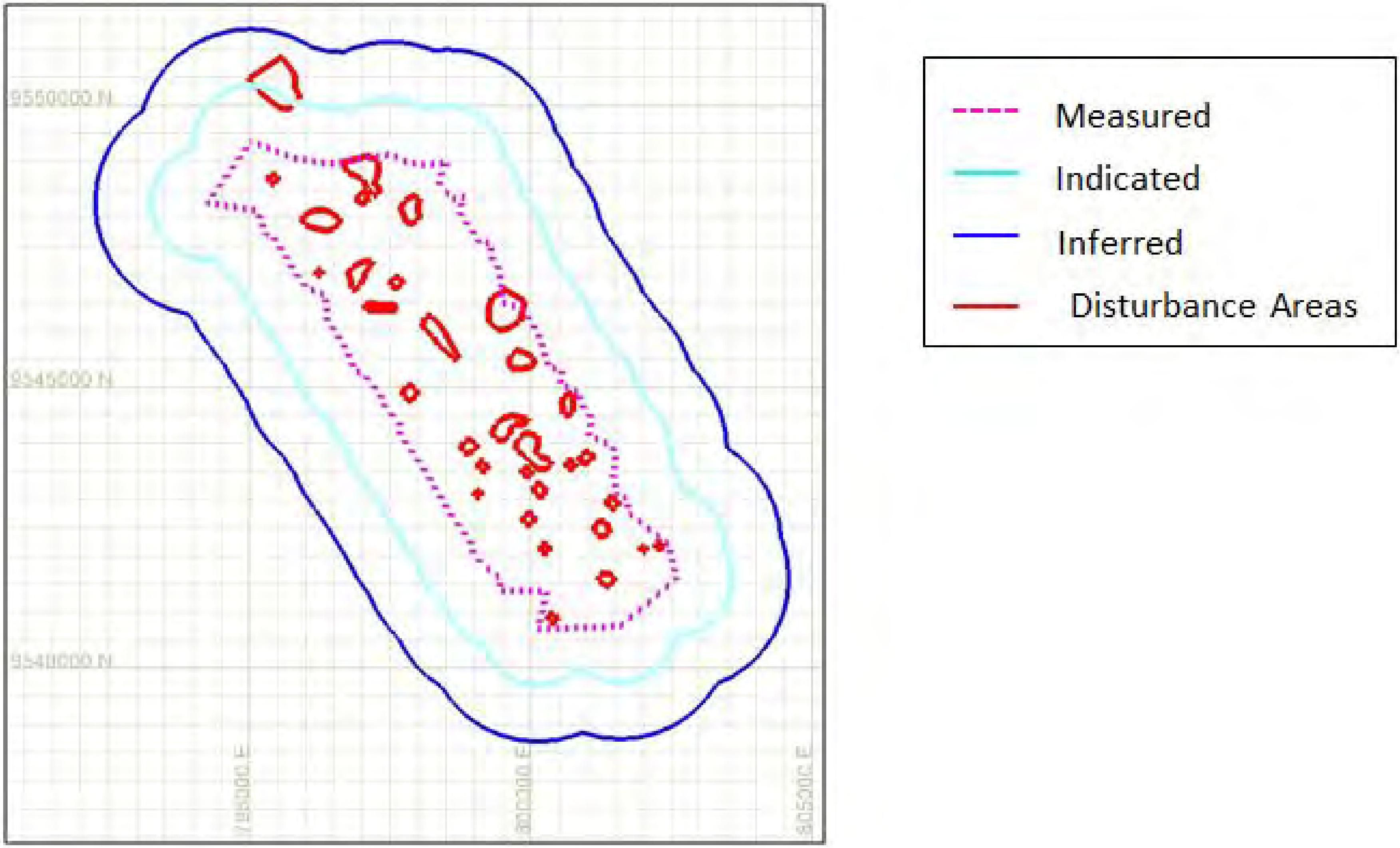

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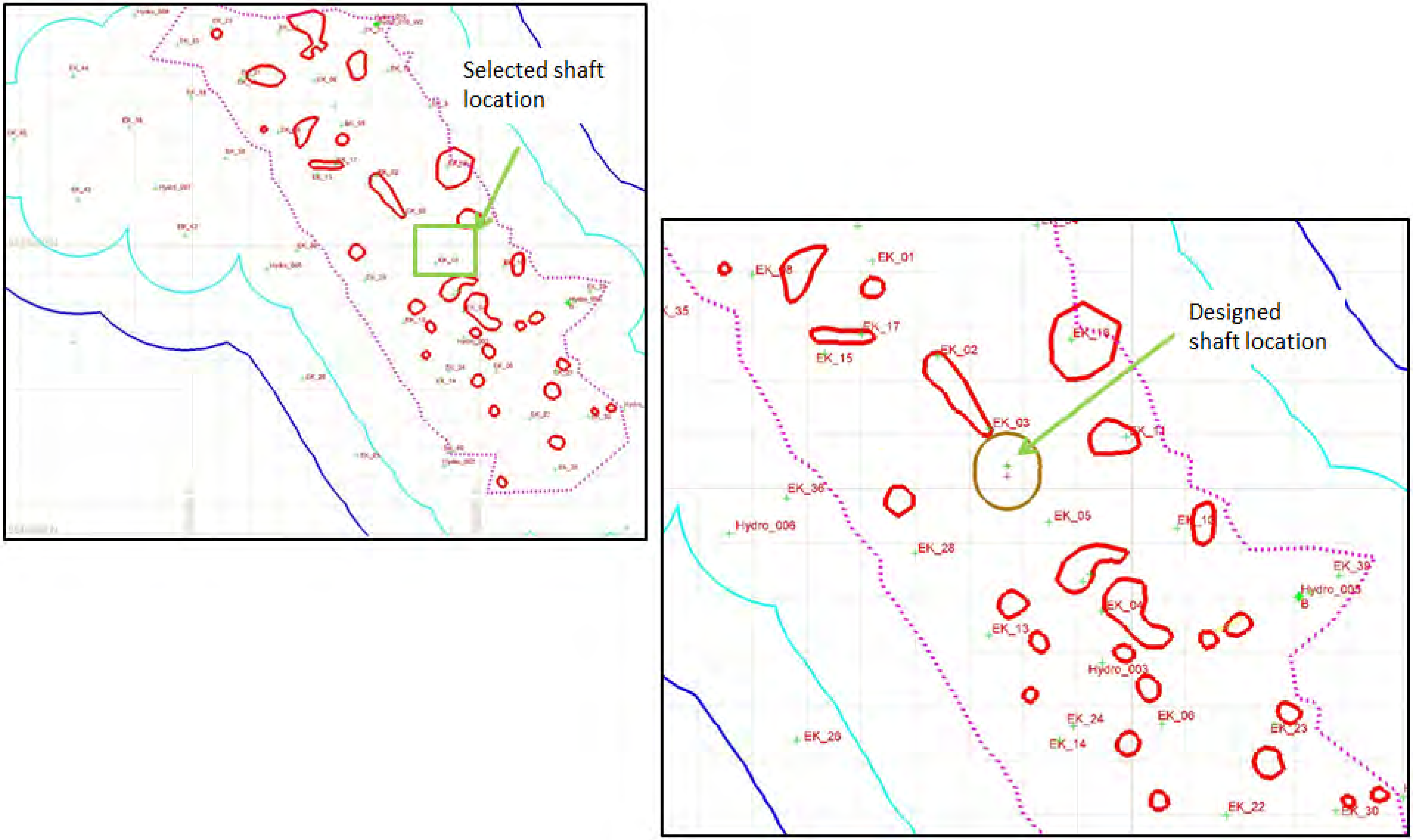
Figure 14-3

Disturbance Areas with Classification Outlines



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Source SRK, 2012

Figure 14-4

Shaft Location

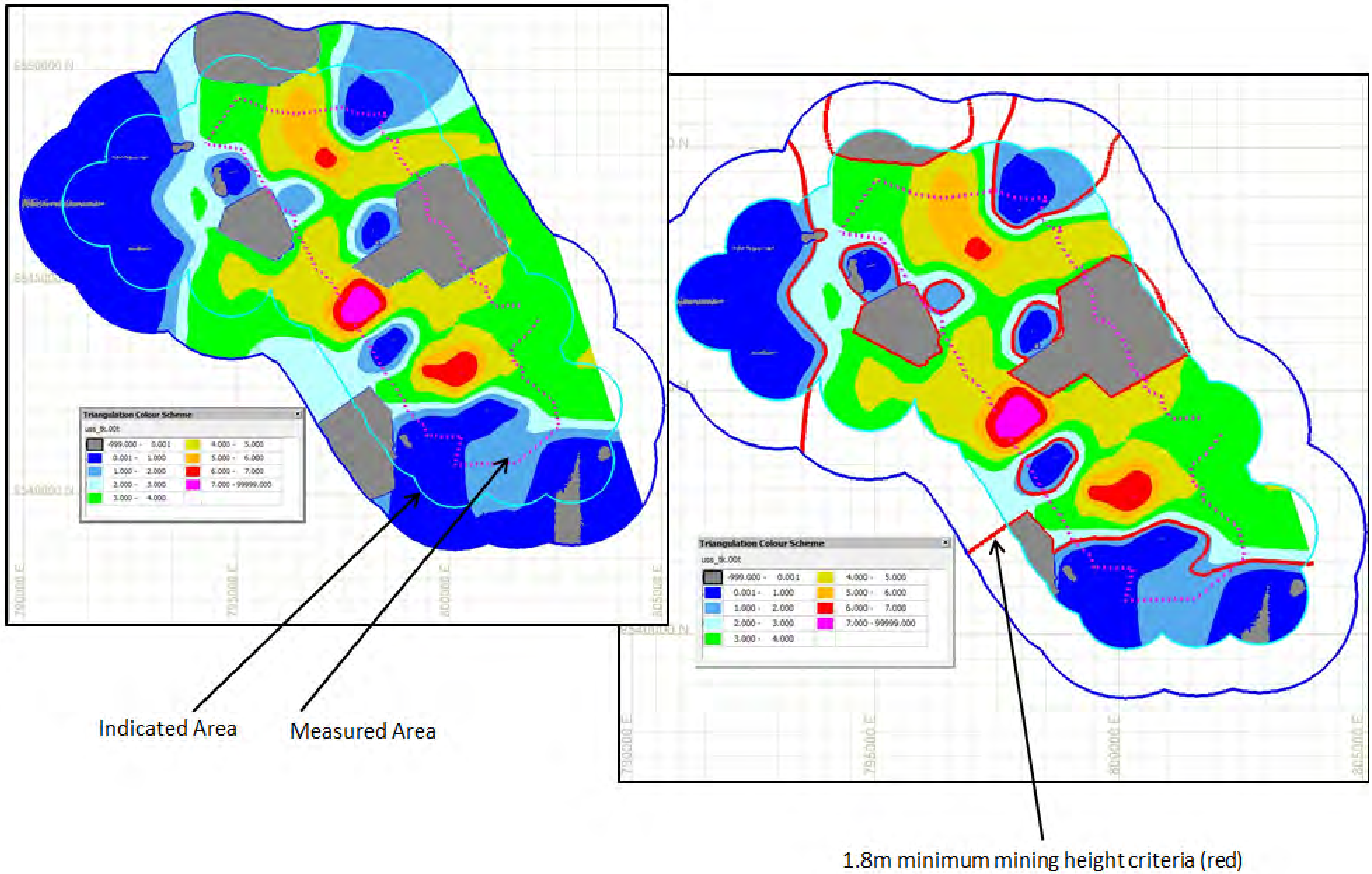

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Figure 14-5a

Mineable Areas for the USS

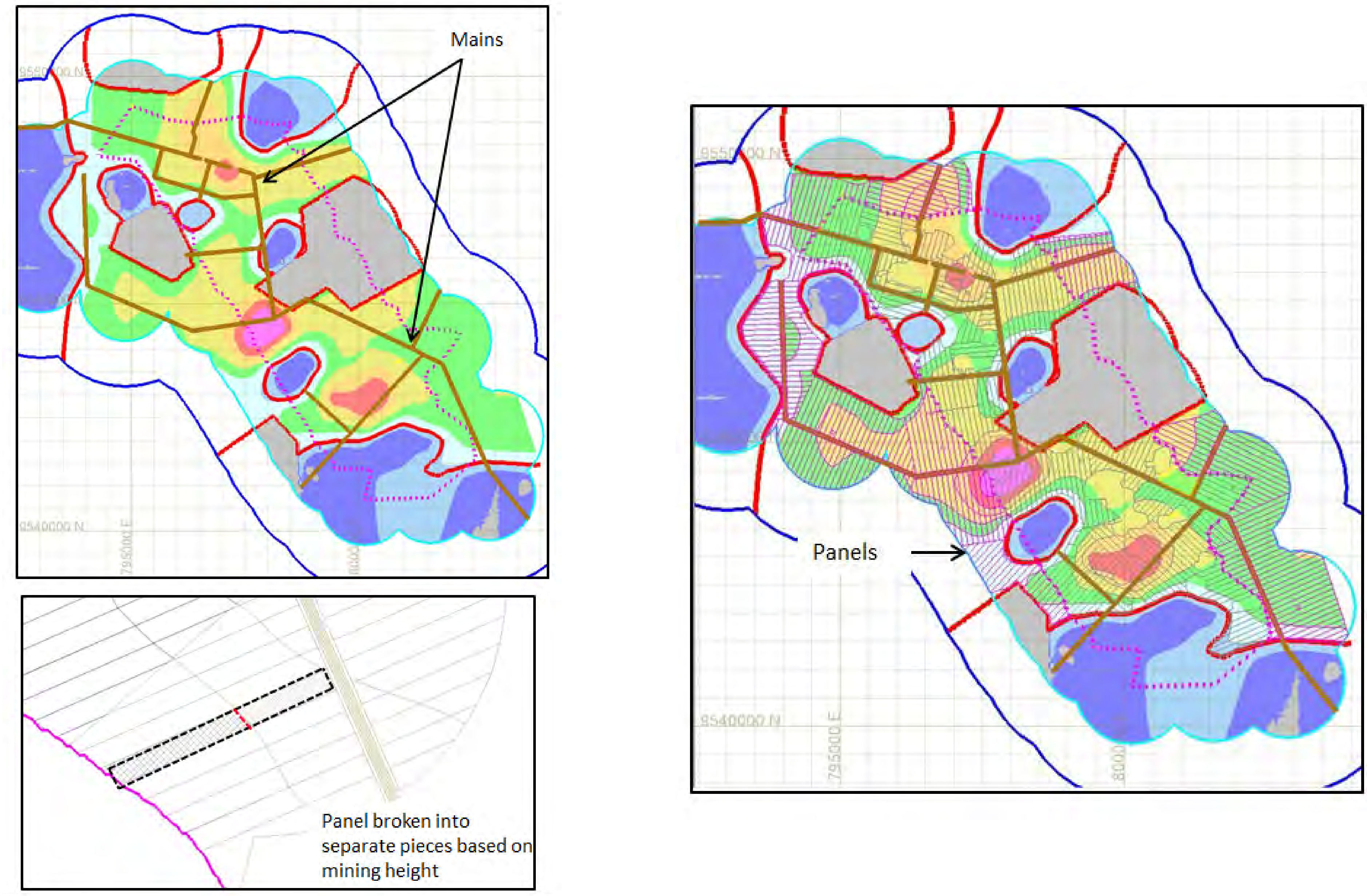

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Figure 14-5b

USS Mining Areas (continued)

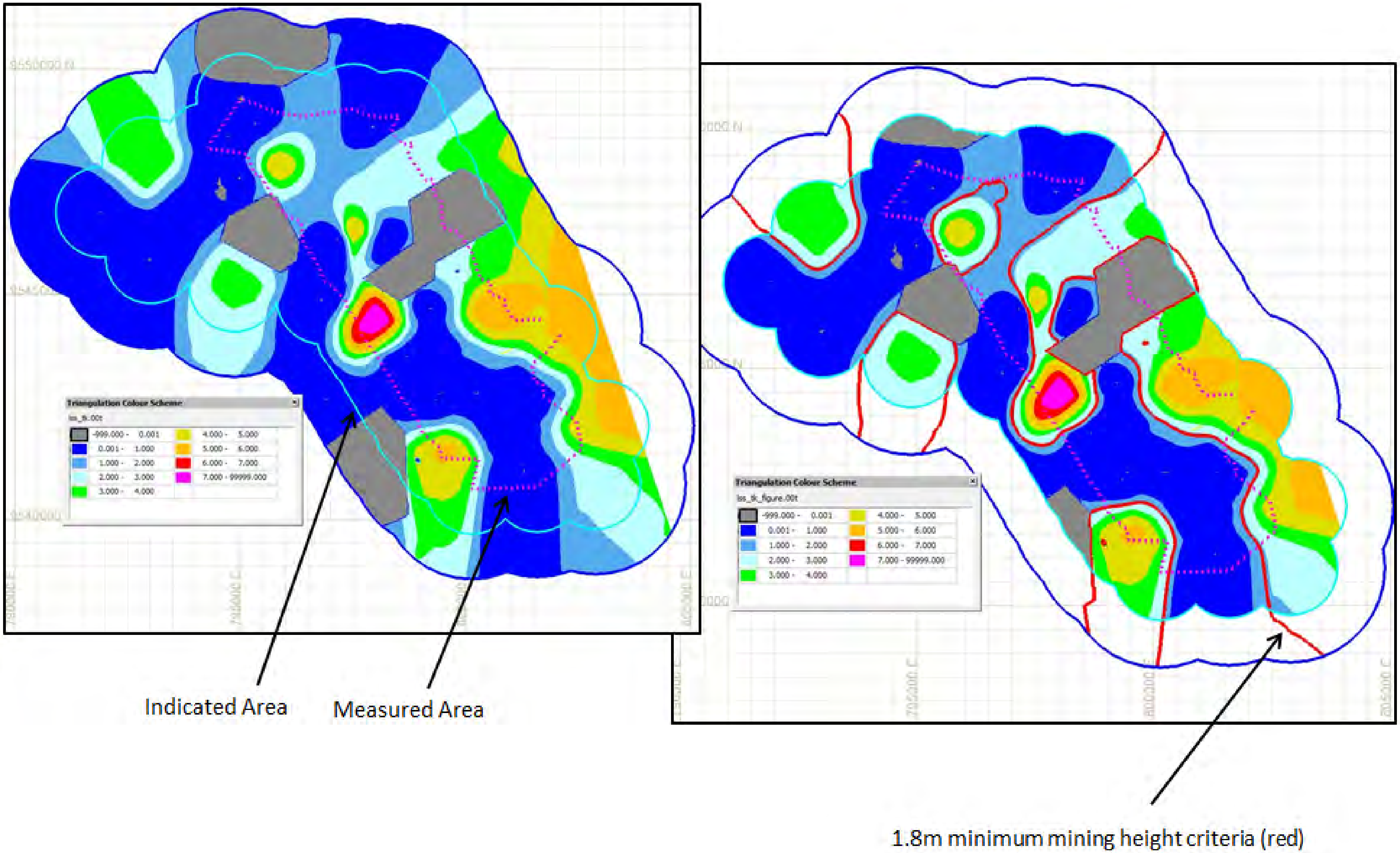

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Figure 14-6a

Mineable Areas for the LSS

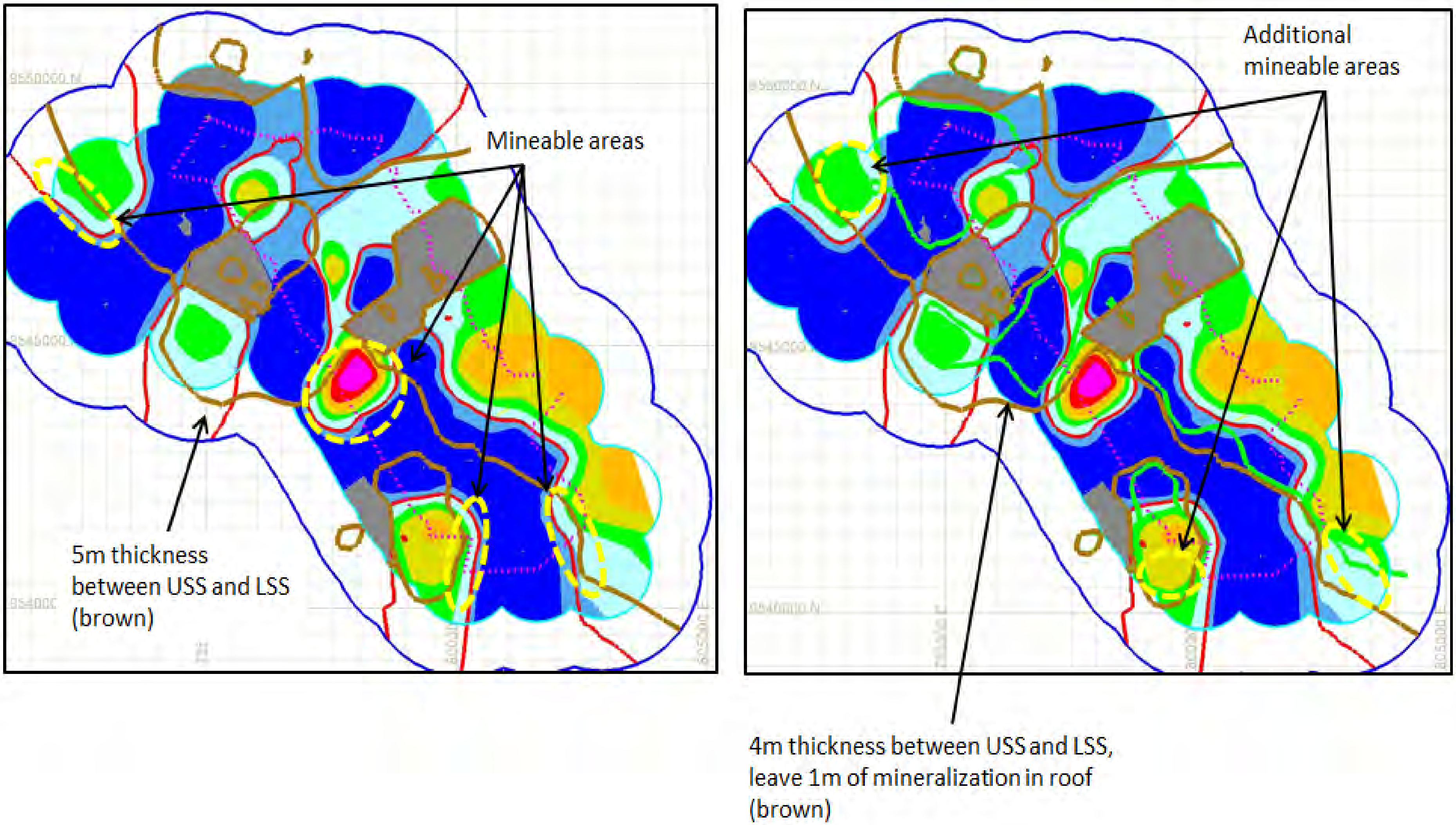

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Figure 14-6b

Mineable Areas for the LSS (continued)

USS shown as polygons with LSS solid shaded to show LSS mining locations with respect to the USS

Mining Heights

Full LSS

1m LSS left in-situ

2m LSS left in-situ

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Figure 14-6c

Mineable Areas for the LSS (continued)

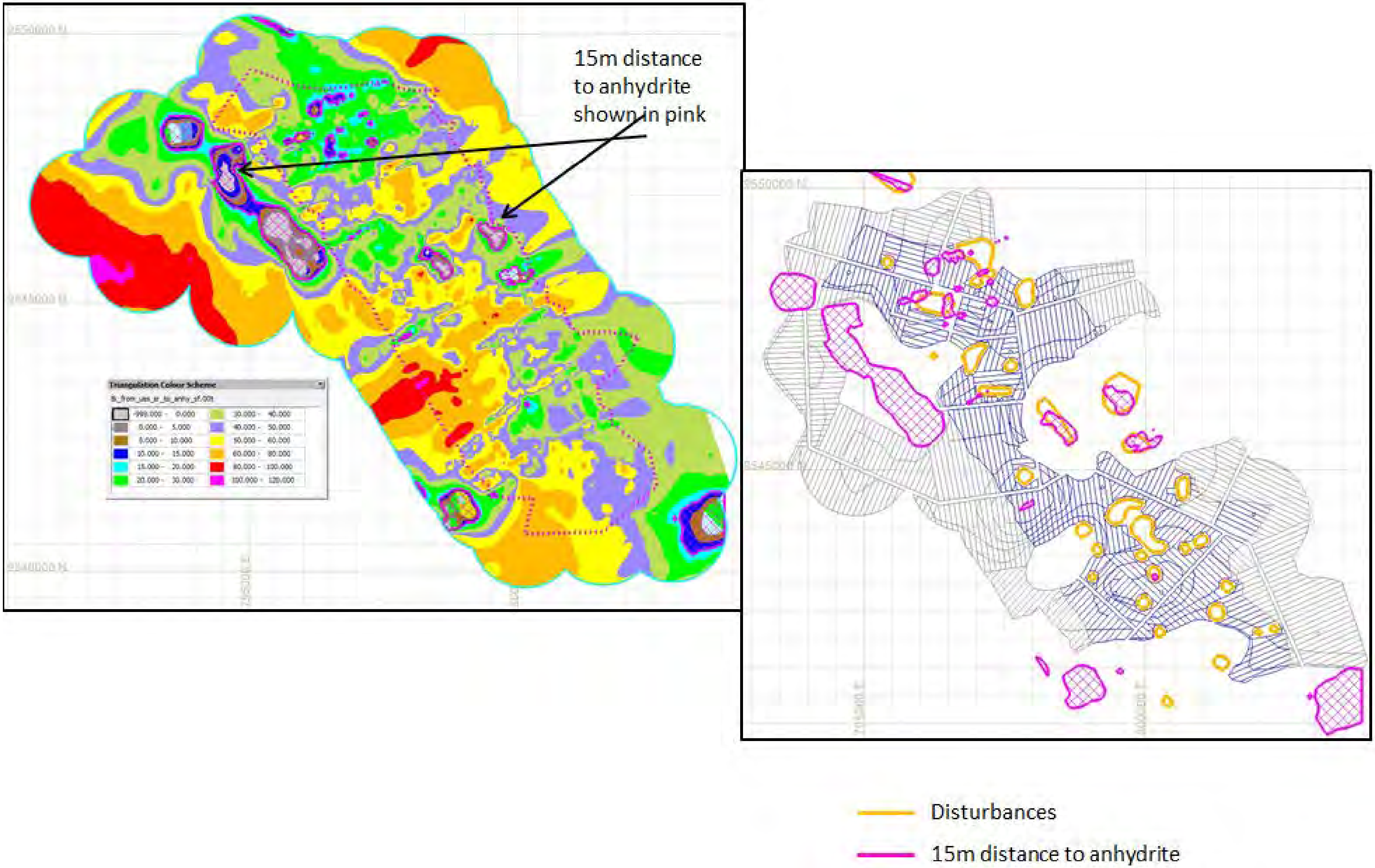

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Figure 14-7

Thickness of Halite between USS Roof and Base of Anhydrite

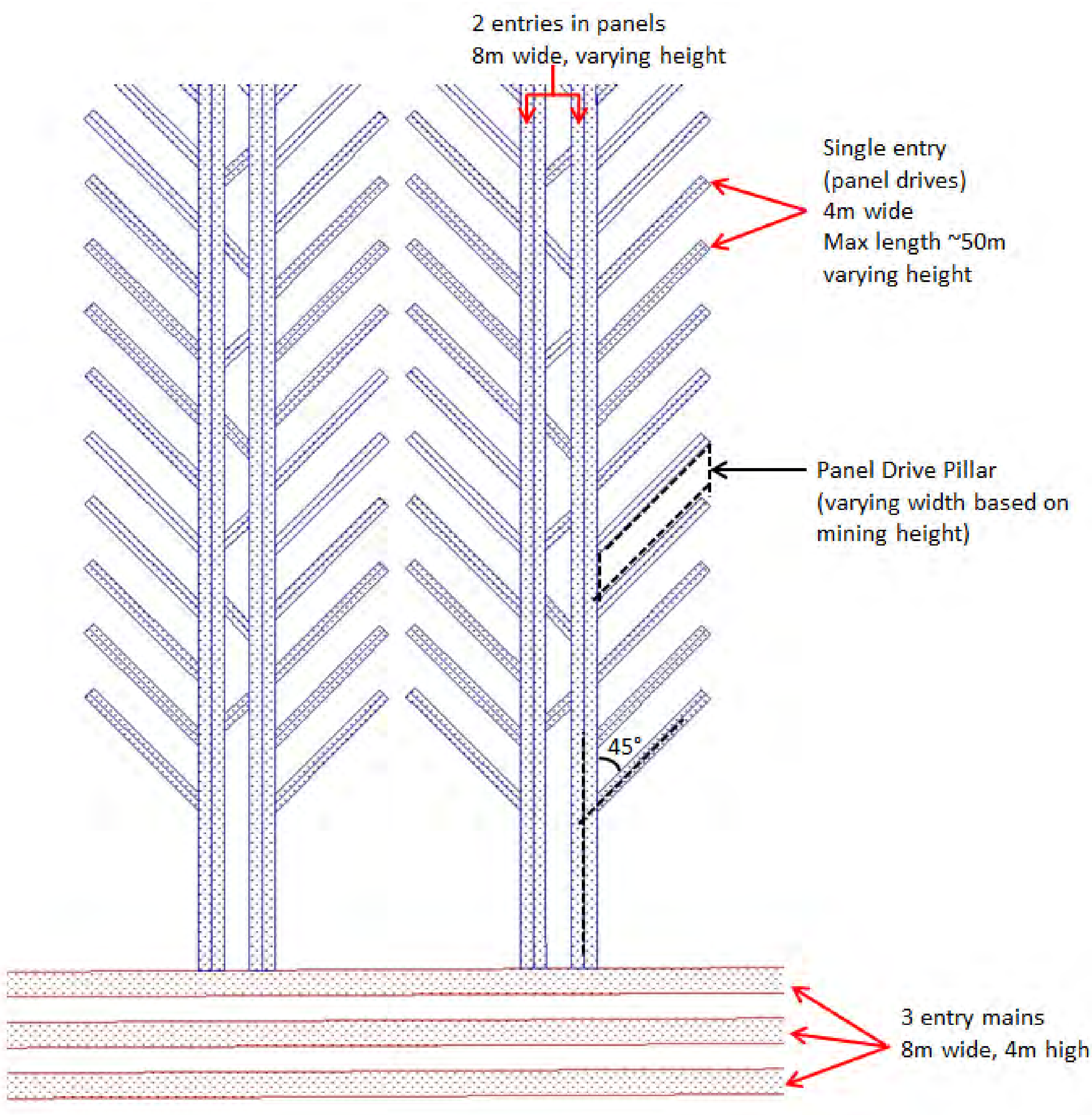

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Figure 14-8

Panel Layout

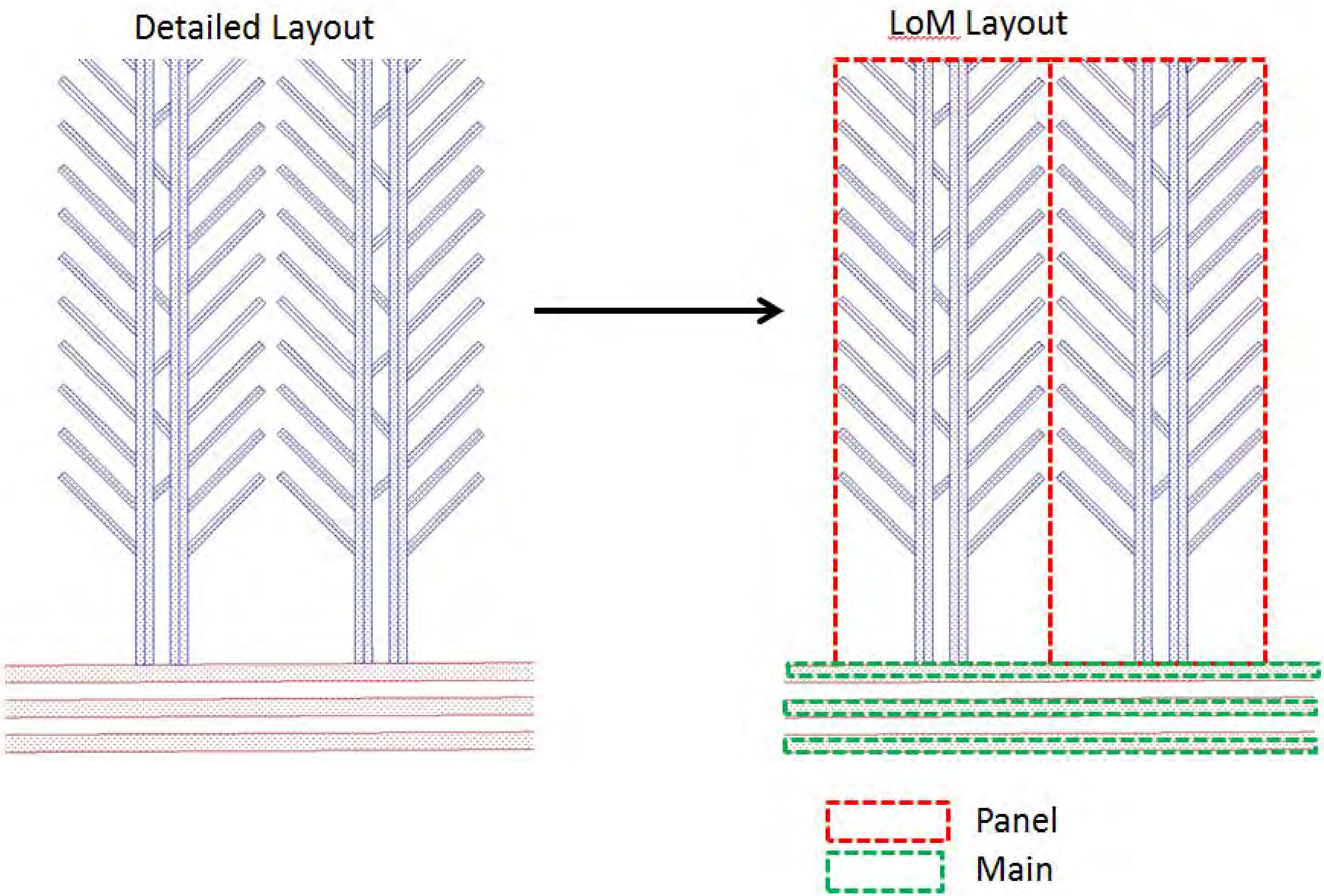

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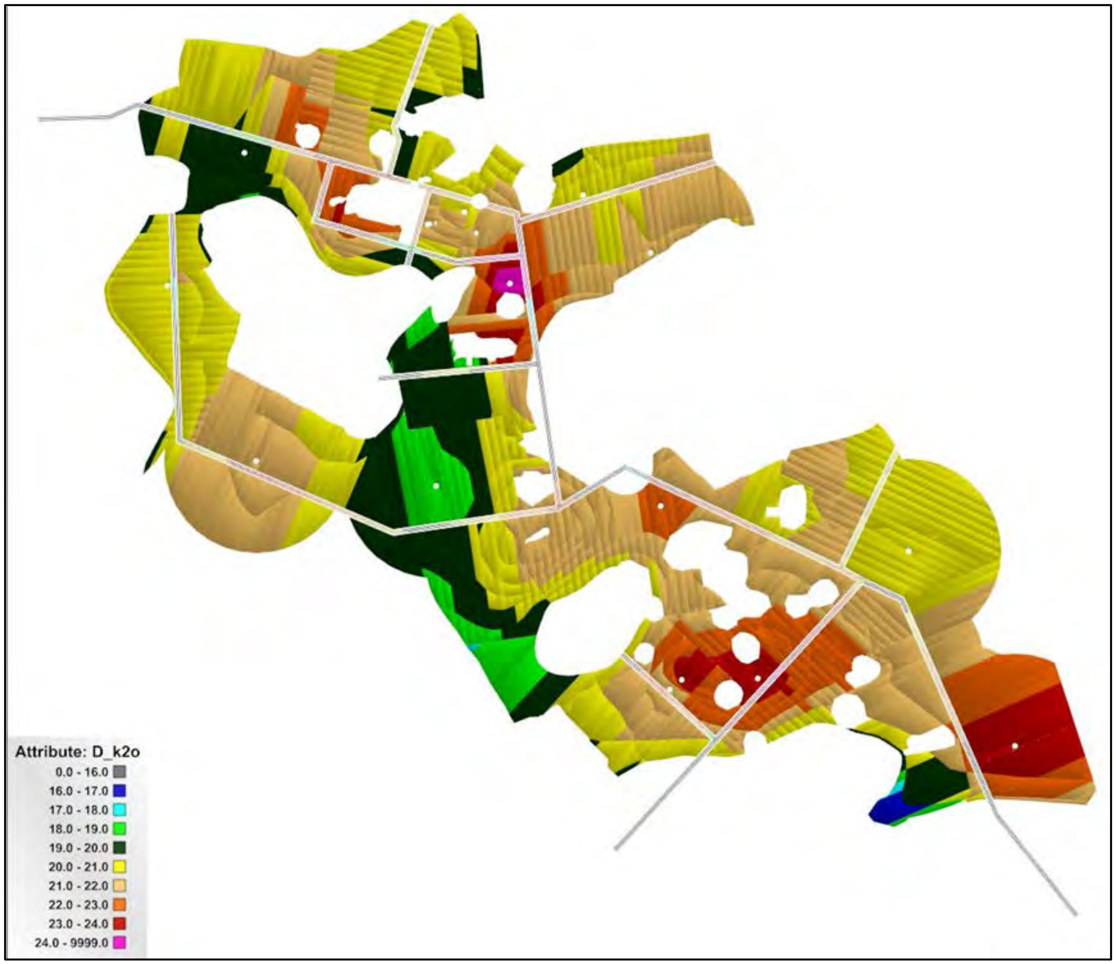
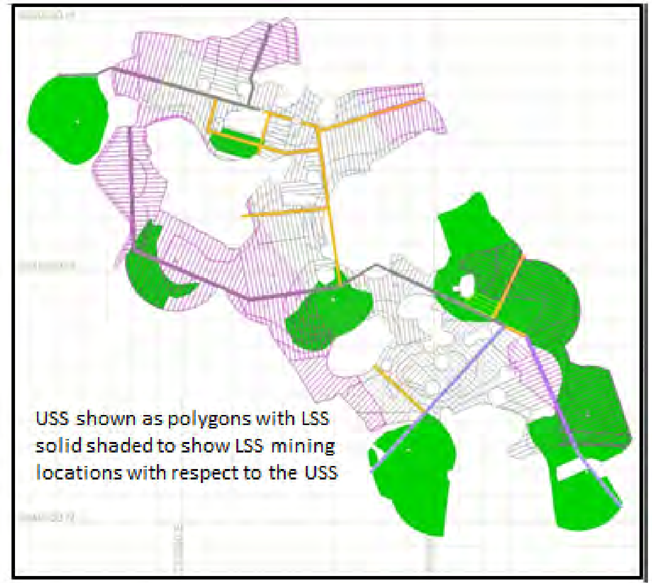
Figure 14-9

Detailed Layout in Comparison to LoM Layout





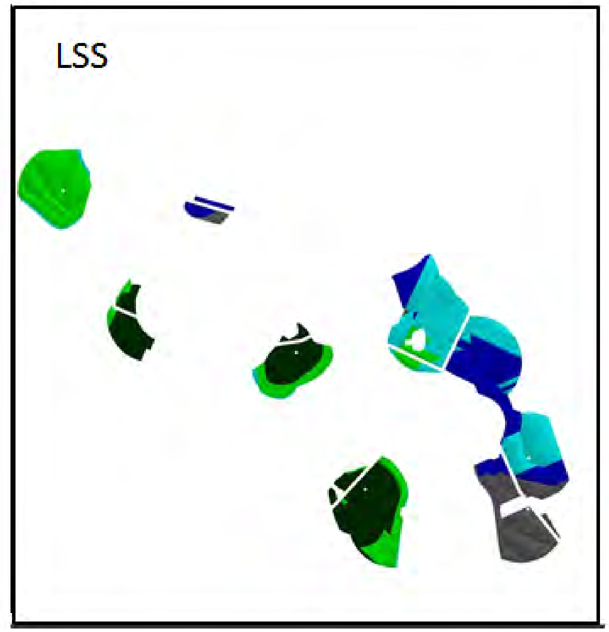


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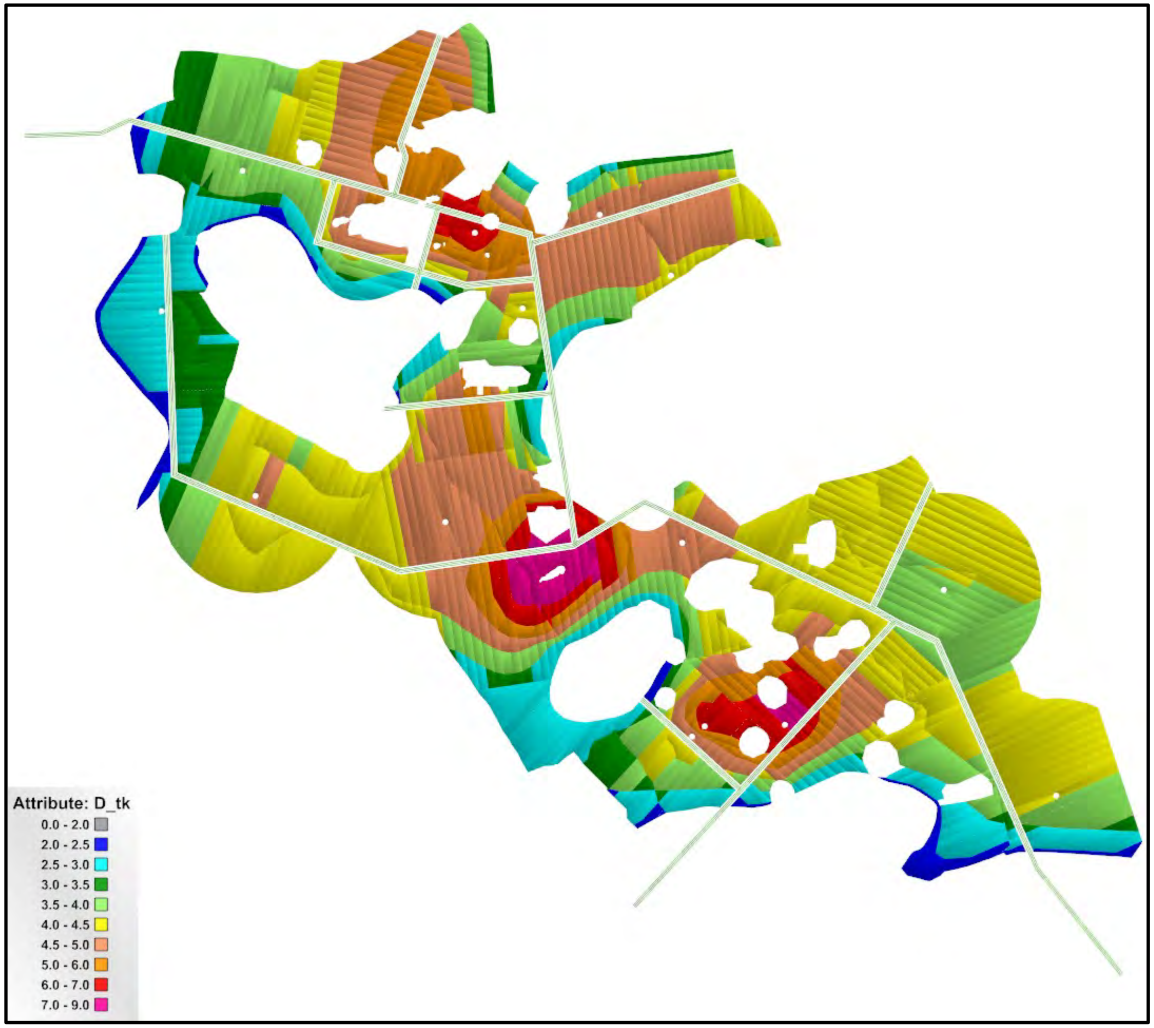
Figure 14-10

Mining Panel Reserve K_2O Grades



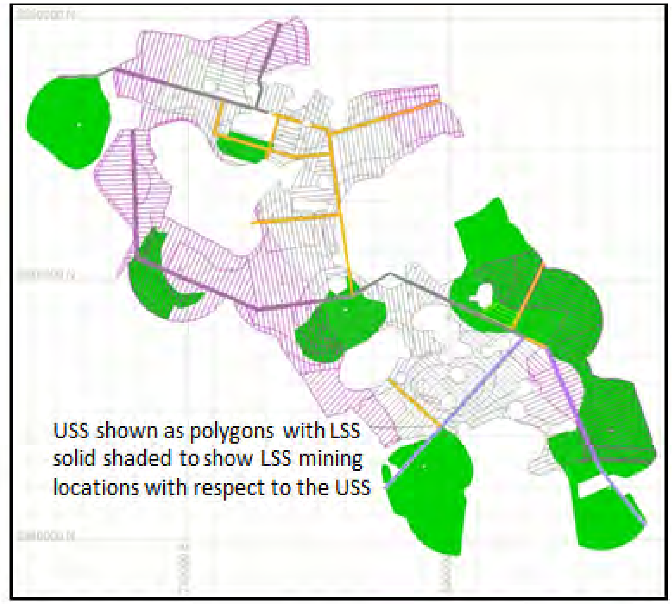


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Figure 14-11

Mining Panel Reserve Thickness

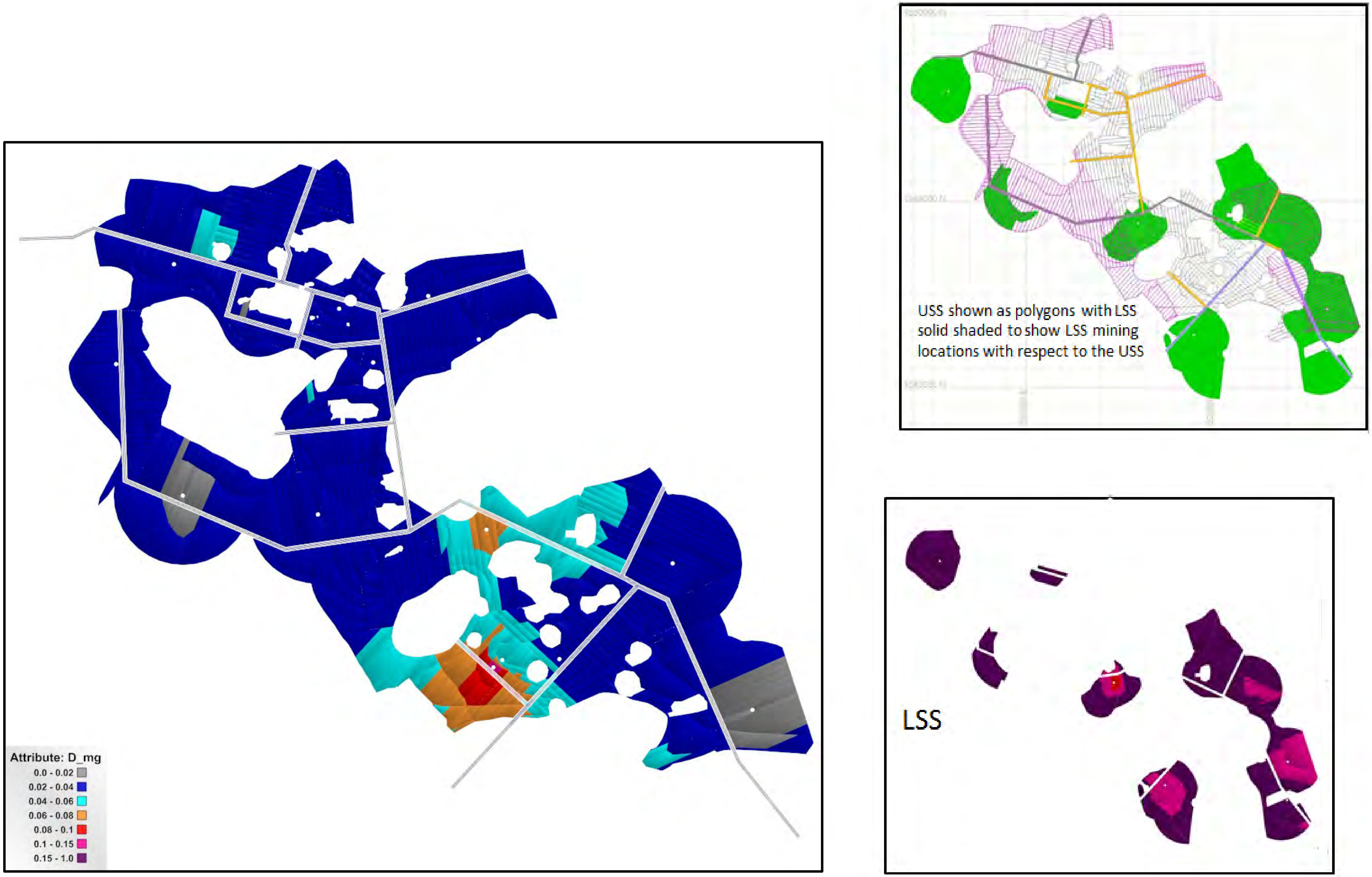

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Figure 14-12

Mining Panel Reserve Mg Grades

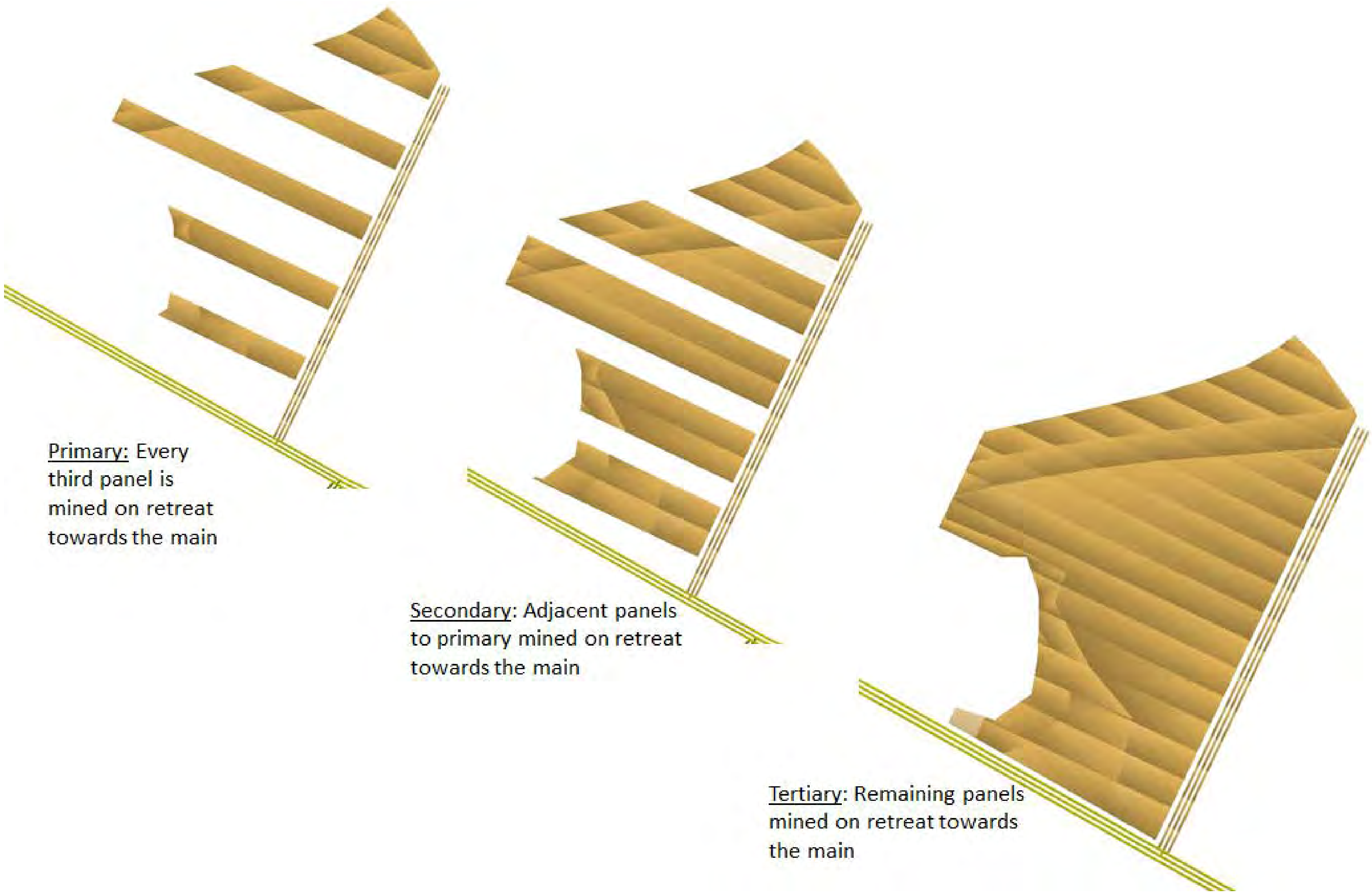

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Figure 14-13

Primary, Secondary, and Tertiary Mining

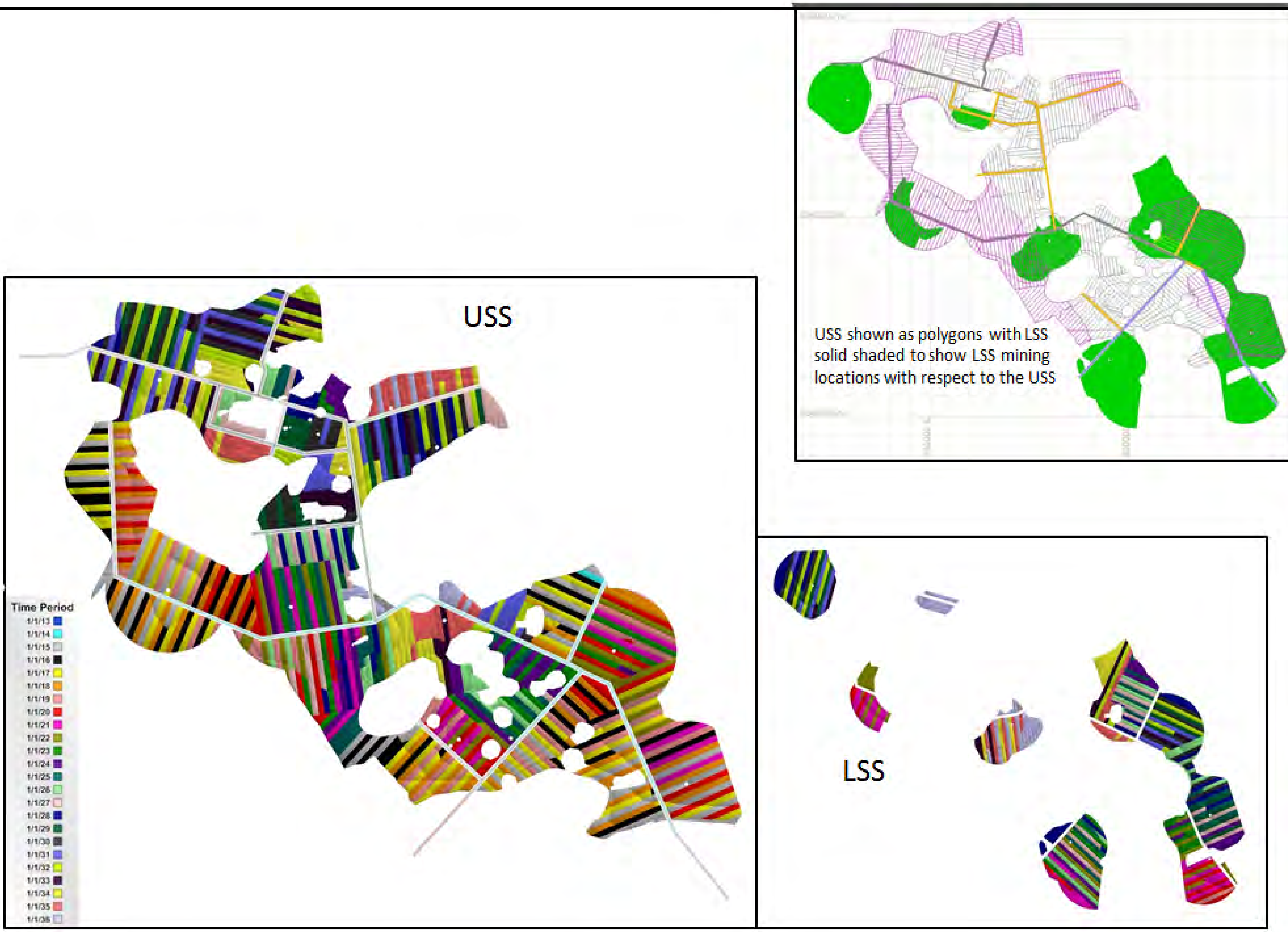

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Figure 14-14

Annual Mining Schedule

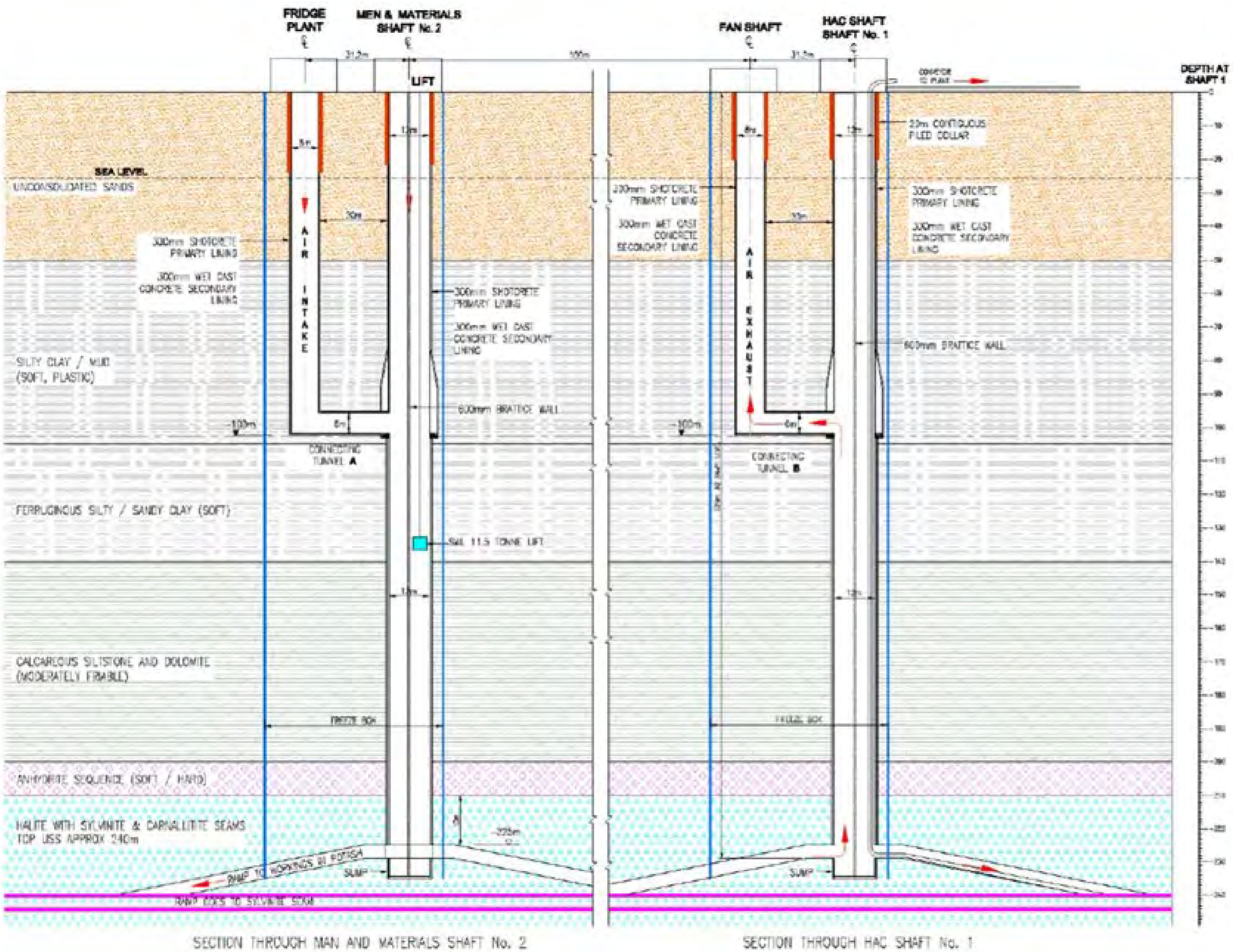

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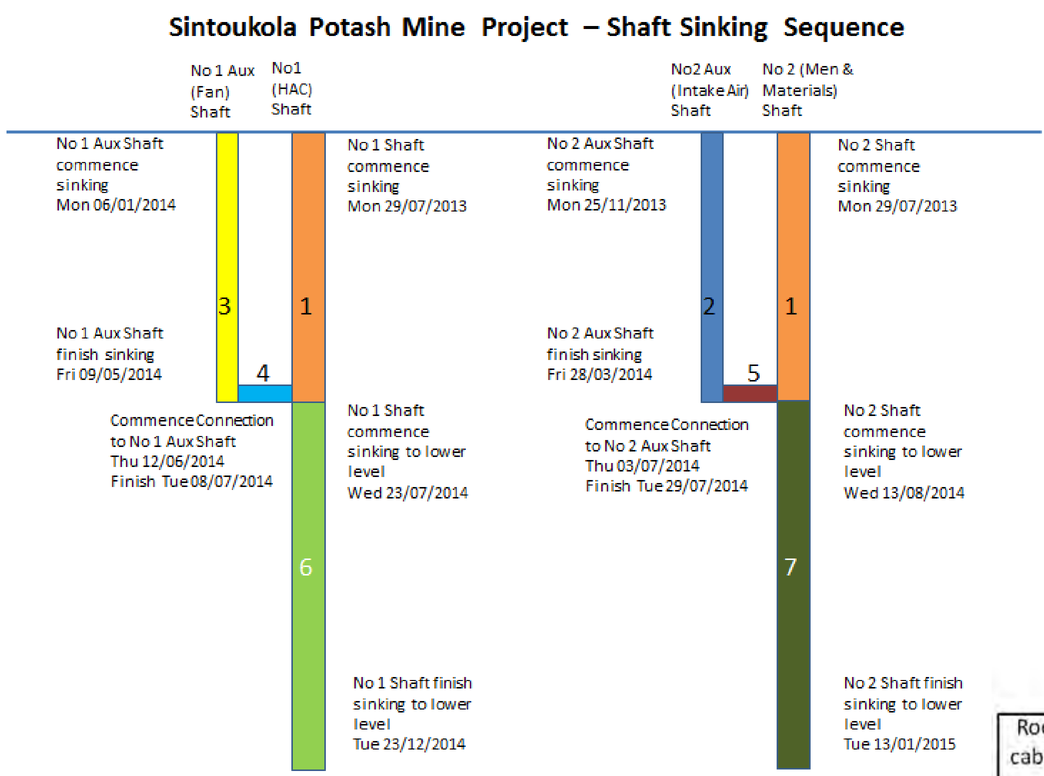
Figure 14-15a

Spacing and Layout of Shafts



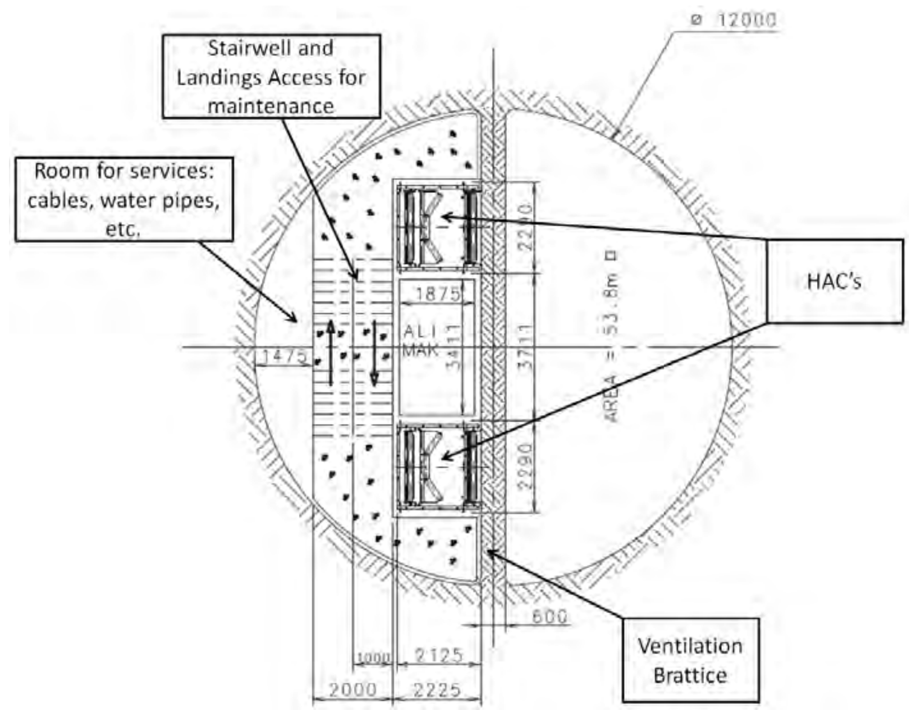


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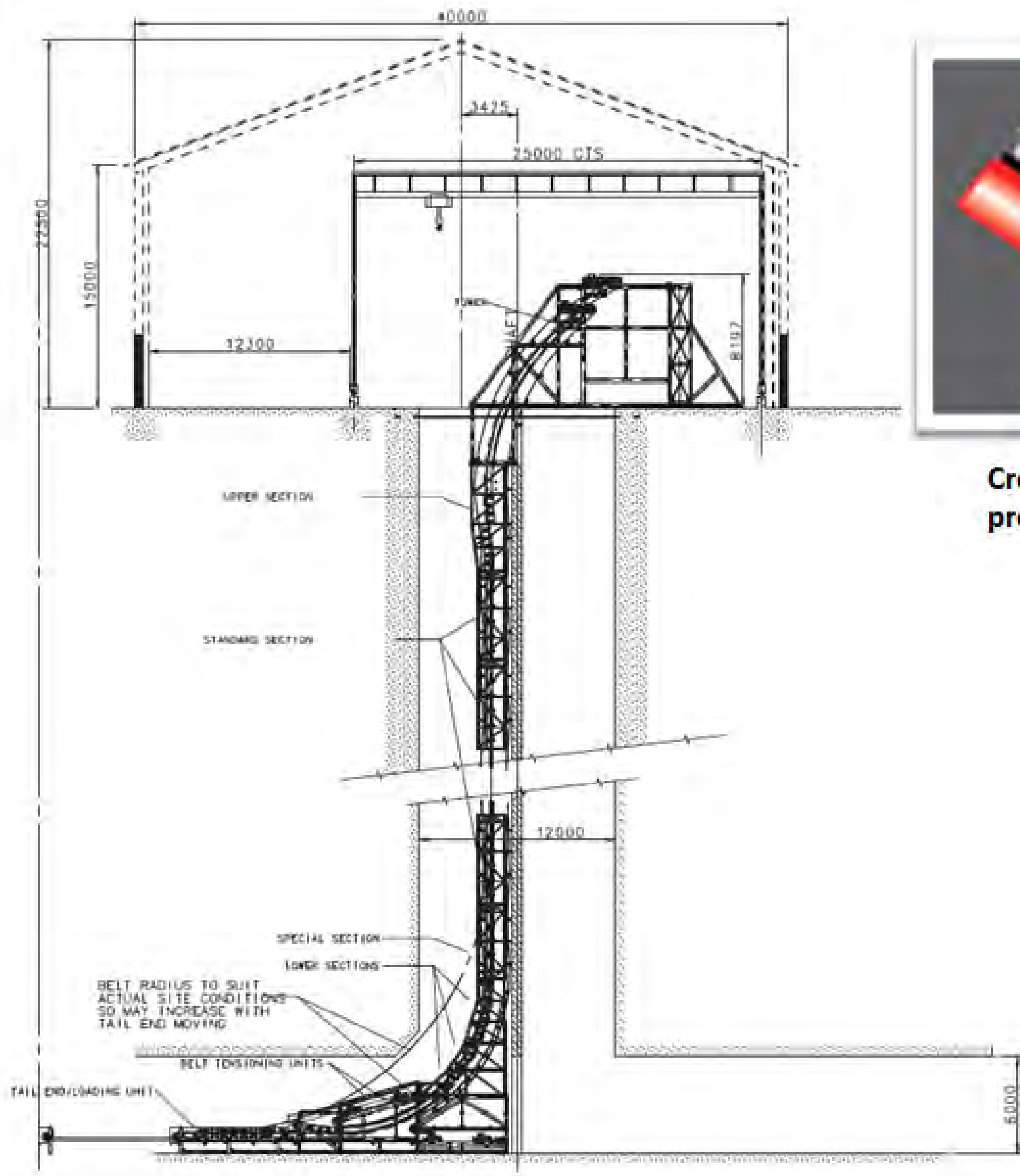
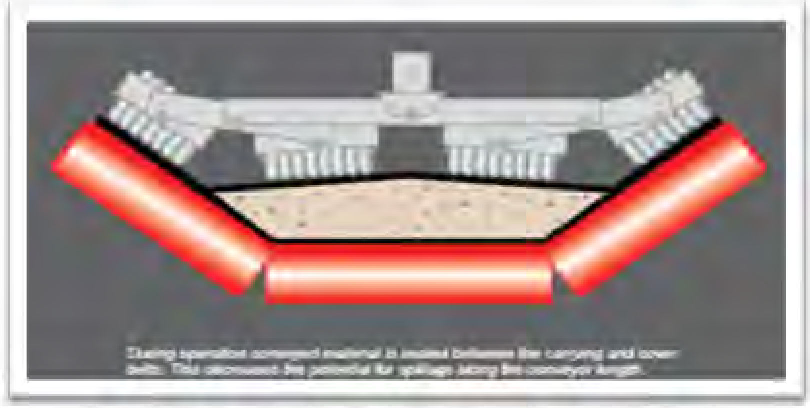
Figure 14-15b

Spacing and Layout of Shafts, continued





Cross Section of the two belts in black with the product squeezed by the top compression rollers.

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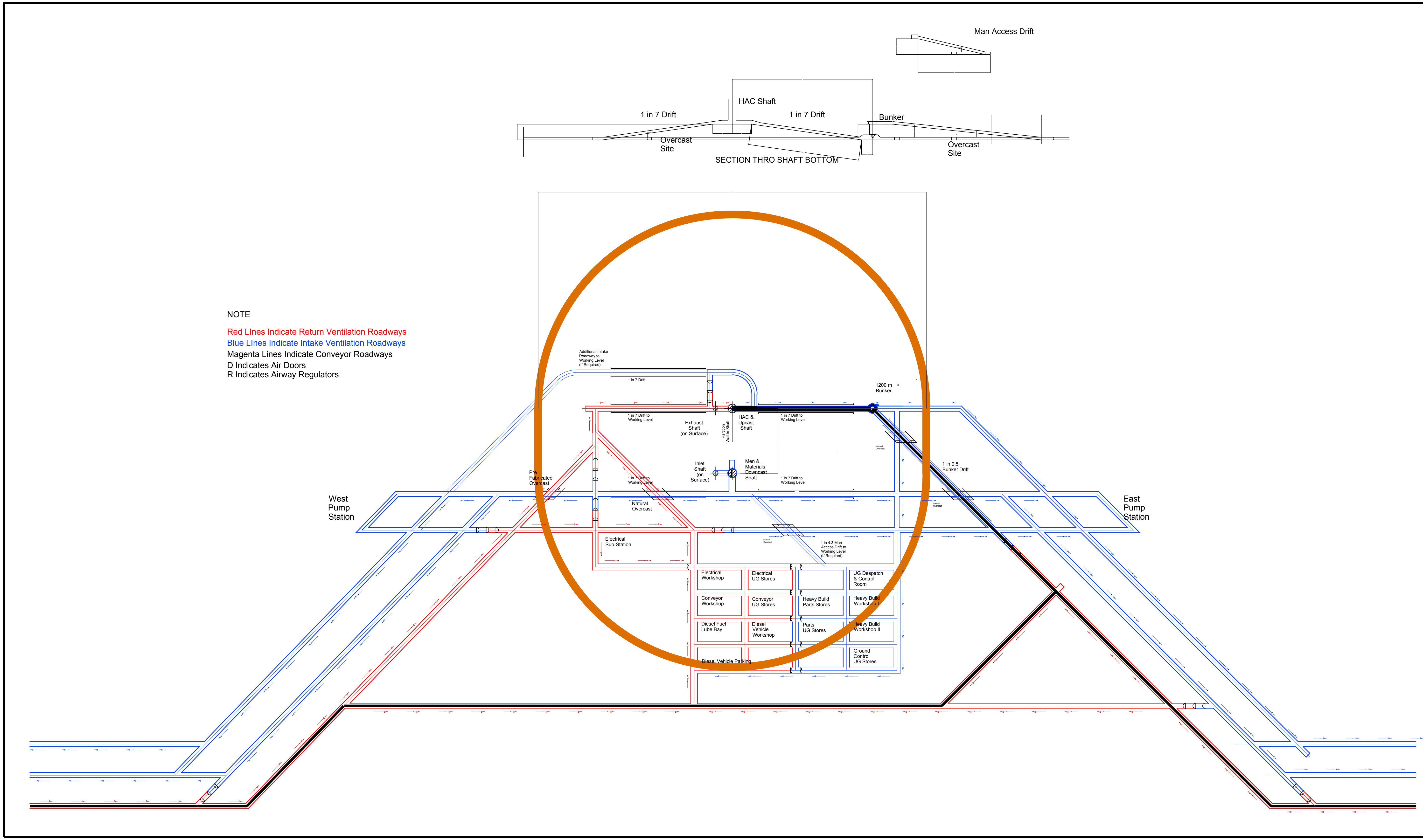
Figure 14-16

High Angle Conveyor (HAC)



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SRK JOB NO.: 340100.020 / TASK 910 / 04

FILE NAME: 340100.020.Rev.A.Fig.14-17.Shaft.Bottom.Layout.2012-09-05.dwg

SINTOUKOLA PROJECT

PRE-FEASIBILITY REPORT

SHAFT BOTTOM LAYOUT

DATE:	APPROVED:	FIGURE:	REVISION NO.
SEPT. 2012	DY/JP	14-17	A

S:\Elemental Minerals 3401\340100.020_Sintoukola_FS_Active\900_AutoCAD\Figures.2012-09\340100.020.Rev.A.Fig.14-17.Shaft.Bottom.Layout.2012-09-05.dwg

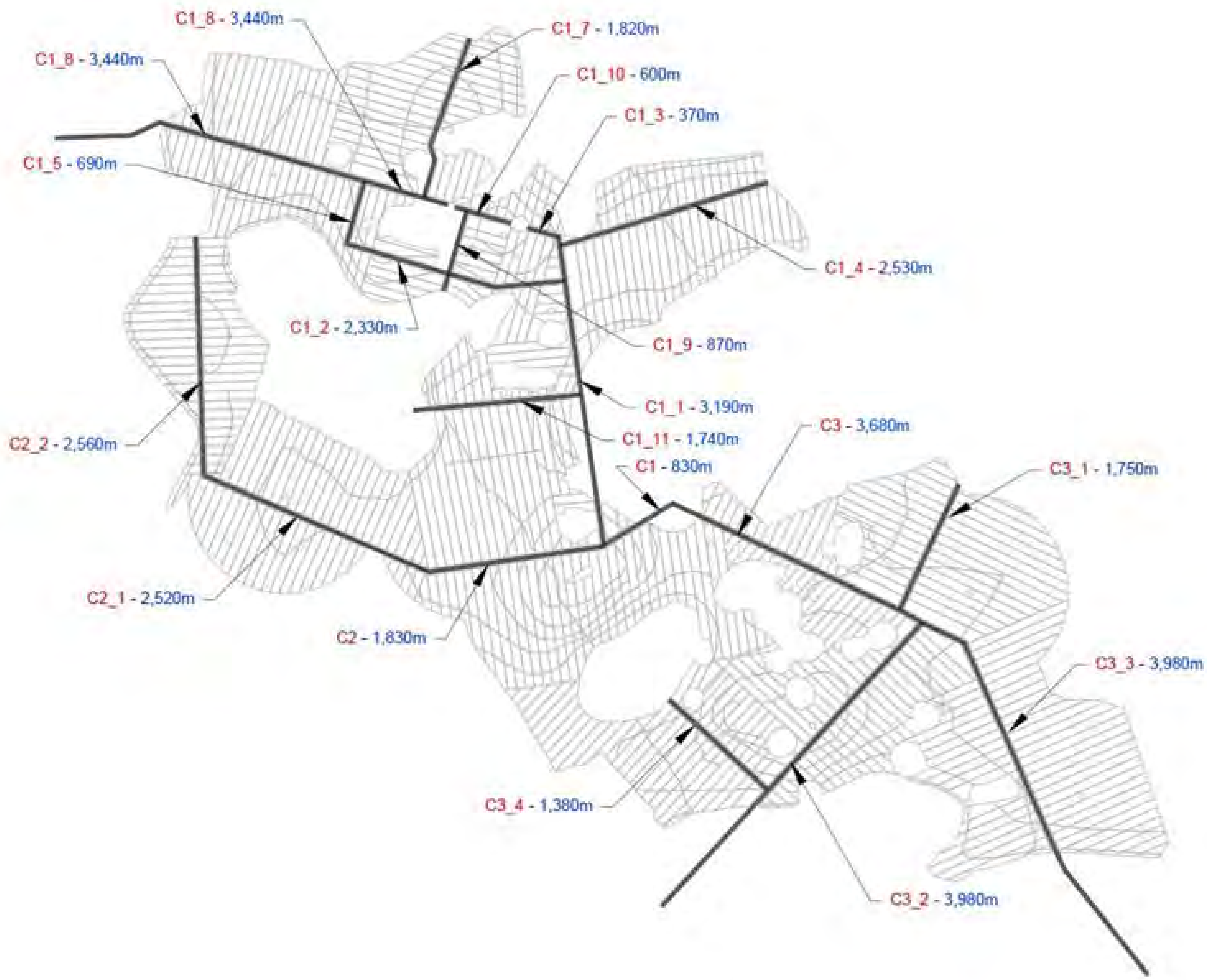

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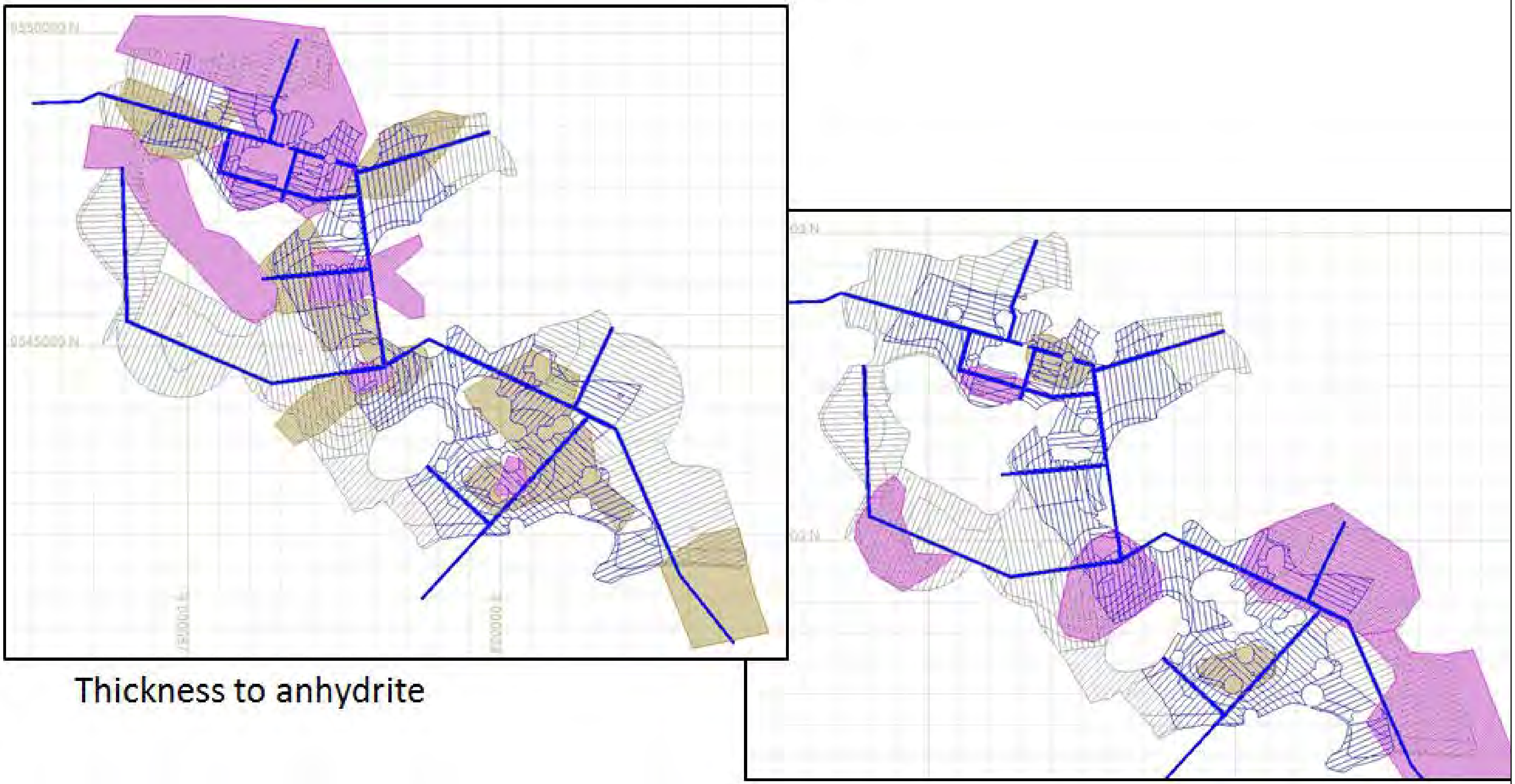
Figure 14-18

Conveyor Belt Locations



Thickness to anhydrite

Mining Height

High - Pink
Medium - brown
No shading - low

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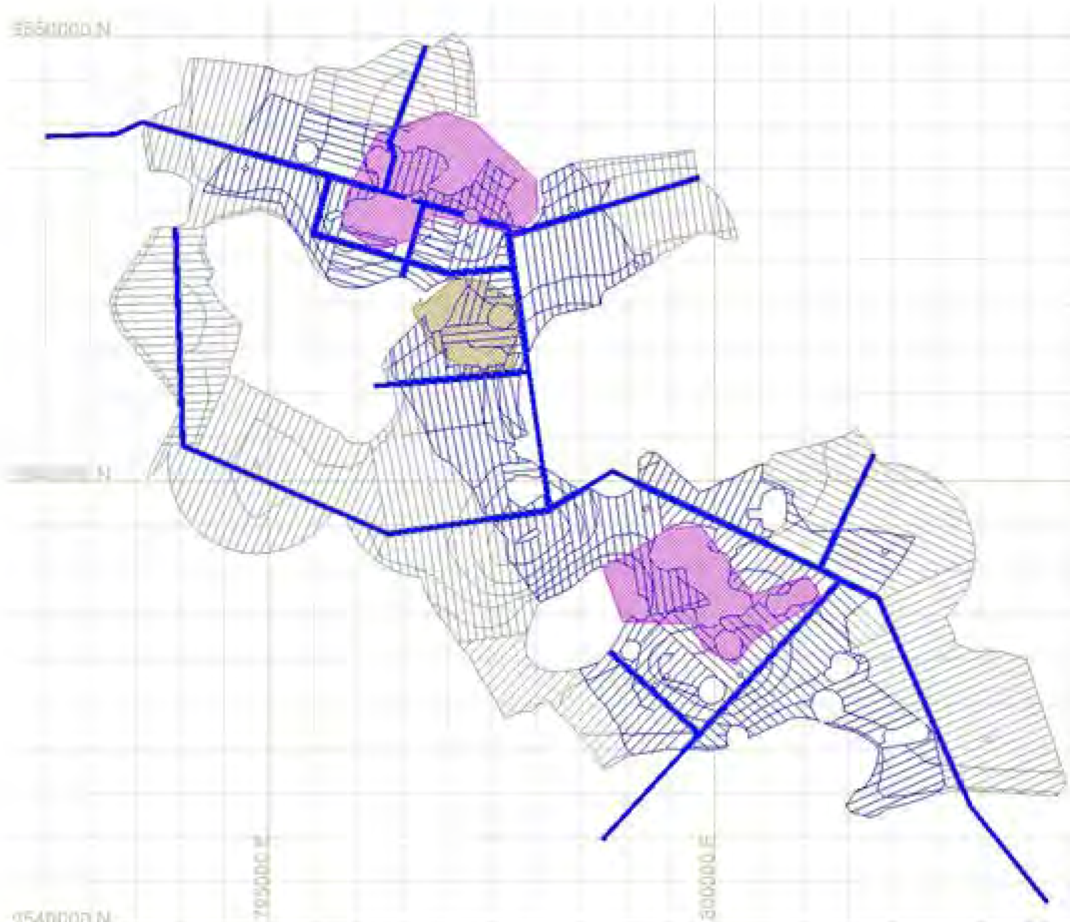
Figure 14-19a

Risk Areas



Note that disturbance areas were only available for the measured area

Disturbances

High - Pink
Medium - brown
No shading - low

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Figure 14-19b

Risk Areas

15 Recovery Methods (Item 17)

The process facility is designed to produce 2.0 Mtpa of potash from the processing of Sintoukola ore. The process plant will produce fertilizer grade potash. The product split is 53:47 between granular product (coarse grained potash with a mean particle size of approximately 2.85 mm and is suitable for direct application and in bulk blending with other fertilizer nutrients) and standard product (fine grained potash with a mean particle size of approximately 1 mm and is suitable for direct application and for use in various NPK formulations). Provisions have been made for the future addition of two compactors and associated equipment which will change the product split to 89:11.

The process plant will process a nominal 900 tonnes per hour (6.84 Mtpa) of RoM ore, based on a RoM grade of 19.8% K_2O (31.3% KCl), with an overall potash recovery of 89.5% and a concentrate grade of 60.5% K_2O (95.8% KCl). After the ore is slurried in the fine ore slurry tank, the mill will operate as two twin streams through to regrind flotation. This allows for a greater range of operating capacity during commissioning and allows for maintenance on one stream while the other stream continues to operate. For the twin streams, pumpboxes are equipped with a single pump with no installed spare. Complete spare pump assemblies should be stocked on site to change out the pump when required.

The process plant is scheduled to operate 326 days annually (one, 2 week shutdown and the equivalent of 25 shutdown days for maintenance) with an availability of 97% after planned shutdowns. The 97% is achievable because the process plant has been designed with twin streams that allow for 1 stream to be taken down for maintenance while still operating at partial rates. Total operating hours are estimated at 7,600 annually.

The process plant will start up at reduced rates and will ramp up to full production over several months of operating. A typical ramp up scenario is shown in Figure 15-1. Typically a process plant such as this would ramp up to full production in six to ten months.

The overall process stages are shown in Figure 15-2. The stages of the process are described in more detail below.

15.1 Mine Site Ore Handling

RoM ore is conveyed from the headframe to a 16,000 t stockpile which is located in a domed structure. The ore is reclaimed from the ore stockpile and is conveyed to a truck loading surge bin. From the truck loading surge bin the ore is loaded onto road trains in a covered load out station to be transported to the process plant.

15.2 Plant Ore Handling

The road trains offload into a truck unloading hopper in a covered offloading facility. From the unloading hopper the ore is conveyed to the raw ore feed crusher, then conveyed to the 6,000 t raw ore storage bin. If required, the raw ore storage bin can be bypassed and the ore can be conveyed to a 16,000 t supplementary raw ore stockpile. Ore can be reclaimed from the supplementary raw ore stockpile with mechanical equipment and fed into the truck unloading hopper.

15.3 Crushing

Ore is reclaimed from the raw ore storage bin and fed to two roll crushers. The roll crushers operate in an open circuit and feed into the fine ore bin. From the fine ore bin the ore is conveyed to the fine ore slurry tank where it is slurried to 65% solids using process brine. The discharge from the fine ore slurry tank is split between two banks of attrition scrubbers. The purpose of the attrition scrubbers is to liberate insoluble material from the potash particles.

The discharge from each attrition scrubber is pumped to two vibrating wet crushing screens. Oversize from the wet crushing screens is fed to a cagepaktor style crusher. The discharge from the cagepaktor is returned to the fine ore slurry tank. Undersize from the wet crushing screens is pumped to the deslime circuit.

15.4 Desliming

The insoluble material (or slimes) is very fine and can inhibit flotation performance if it is not removed from the potash particles. In addition, the liberated insoluble material must be removed from the slurry prior to flotation otherwise it will absorb flotation reagents causing increased consumptions. The insoluble material is removed by screening followed by hydrocyclone classification.

The undersize from each set of wet crushing screens is pumped to a set of two deslime screens. The deslime screens separate the fine material (minus 1 mm) from the coarse material. The fines (screen undersize) are pumped to a pack of deslime hydrocyclones. The coarse (screen oversize) is fed to the coarse conditioning drum via a drag conveyor. The overflow of the cyclones contains the majority of the fine insoluble material. The deslime cyclone overflow will be combined with the salt centrifuge centrate cyclones underflow and be fed to the hydroclassifier. The underflow of the deslime cyclones is fed to the fines conditioning launder.

In the hydroclassifier the fine insoluble material (slimes) reports to the overflow while the coarser solids report to the underflow. The hydroclassifier underflow is fed to the scavenger flotation circuit. The majority of the insoluble material reports to the hydroclassifier overflow, this will be combined with the salt tails cyclone overflow, the discharge from the crusher dust scrubber and the discharge from the product dryer scrubber and fed to the slimes thickener. The slimes thickener underflow is pumped to the RSF. Clear brine from the slimes thickener overflow flows to the process brine distribution tank.

15.5 Flotation

The deslime cyclone underflow is reagentized in the fines conditioning launder with the addition of the required dosage of flotation reagents. The fines conditioning launder flows to the rougher flotation feed pumpbox.

The coarse material is reagentized in the coarse conditioning drum with the addition of the required dosage of flotation reagents. The discharge from the coarse conditioning drum flows to the rougher flotation feed pumpbox. Process brine is added to the rougher flotation feed pumpbox to maintain the slurry density of the rougher flotation feed at 35% solids. Separate fines and coarse reagentizing ensures that each fraction receives the required dosage of flotation reagents without the risk of the fine particles absorbing reagents away from the coarser fractions.

Reagentized ore slurry from the rougher flotation feed pumpbox is pumped to two banks of rougher flotation machines in parallel. Design is based on a minimum of 5 minutes of retention time. Rougher flotation concentrate is screened using sieve bend style screens which provide a separation size of approximately 500 µm. The oversize (+500 µm) from the screens is sent to the product centrifuge feed pumpbox. The undersize (-500 µm) fraction flows to the cleaner flotation circuit.

The rougher flotation tails flow to a vibrating regrind screen. The oversize from the regrind screen (+1.19 mm) is sent to a regrind cagepaktor crusher. The discharge of the regrind crusher is reagentized in the regrind launder with the addition of the required dosage of flotation reagents. The underflow of the regrind launder flows to the regrind flotation feed pumpbox.

Reagentized ore slurry from the regrind flotation feed pumpbox is pumped to two banks of regrind flotation cells in parallel. Design is based on a minimum of five minutes of retention time. Regrind flotation concentrate is screened using sieve bend style screens which provide a separation size of approximately 500 µm. The oversize (+500 µm) fraction from the screens is sent to the product centrifuge feed pumpbox. The undersize (-500 µm) fraction flows to the regrind cleaner flotation feed pumpbox.

The tails from regrind flotation are recirculated to the regrind screen resulting in a closed loop regrind circuit. The undersize from both regrind screens flows to the salt cyclone feed pumpbox.

The undersize from the rougher concentrate sieve bends and the undersize from the regrind concentrate sieve bend is combined together and fed to four cleaner flotation columns in parallel. Concentrate from cleaner flotation is sent to the product centrifuge feed pumpbox. The tails from cleaner flotation are recirculated back to the rougher flotation feed pumpbox.

15.6 Scavenger Flotation and Slimes Thickening

The hydroclassifier underflow is pumped to the scavenger flotation feed launder where it is reagentized with the required dosage of flotation reagents. The reagentized slurry is fed to a single scavenger flotation column. Concentrate from the scavenger flotation column is sent to the product centrifuge feed tank pumpbox. Underflow from the scavenger flotation column are sent to the salt tails centrifuge feed pumpbox.

The overflow from the slimes thickener flows to the process brine distribution tank. The underflow from the slimes thickener is pumped to the RSF. The insolubles are contained in the RSF and the decanted brine is pumped back to the process brine distribution tank.

15.7 Salt Handling

The salt cyclones are used to thicken the salt slurry prior to centrifuging. The salt cyclones are fed from the salt cyclone feed pumpbox. The underflow from the salt cyclones is sent to the salt centrifuge feed tank. The overflow for the salt cyclones is fed to the slimes thickener to clean the brine prior to the brine being recycled to the process brine distribution tank.

The salt slurry is then fed to six screen bowl centrifuges operating in parallel. The centrifuges will debrine the salt to 95% solids. The salt cake from the centrifuges is fed to two salt centrifuge launders where they are re-slurried with sea water to 40% solids. The salt slurry then flows to the salt disposal tank. As the sea water is under-saturated, it will start to dissolve the solid salt in the disposal tank and in the pipeline. This salt slurry at 20 to 25% solids is pumped to the brine dilution tank, which is part of the salt brine disposal system being designed by EGIS.

15.8 Product Centrifuging and Drying

Concentrates from the flotation and scavenging processes are debrined in four screen bowl centrifuges. The centrifuges debrine the slurry to a level of >95% solids. The debrined solids are fed to two fluid bed dryers. Product is discharged from the product dryer at less than 0.1% moisture and approximately 180°C. The off gases from the product dryer are fed to the product dryer dust cyclones. The cyclone underflows are combined with the respective dryer discharge. The product dryer dust cyclone overflows are fed to the product dryer scrubbers. The discharge from the product dryer scrubbers is pumped to the slimes thickener.

15.9 Product Screening

Dried product from the product dryers is fed to four multi-deck product screens. The product is separated into three size fractions, namely standard product, oversize and undersize. The standard product is fed to the standard product cooler before being conveyed to product storage. The oversize and undersize fractions, and a portion of the standard fraction (the portion of standard will vary depending on market conditions), are combined and fed to the compaction plant.

15.10 Compaction and Product Treatment

The compaction circuit generates granular product through compaction, flake breaking and screening. Initially the compaction circuit will have three compactors. Room will be left to add two compactors for future expansion. Each compactor is operated in a circuit with a force feeder, a double roll flake breaker, two multi deck screens and a cagepaktor crusher.

Each compaction bucket elevator feeds two multi deck vibrating compaction screens. The oversized material is crushed in a cagepaktor and circulated back to the compaction screens. The on-spec granular product is sent to the glazing circuit. The undersized material is combined with the fresh feed from product screening and the material from the surge bin and is fed to a compactor. Each compactor has a force feeder to aid in getting the feed distributed across the compactor rolls. The compactors create dense, competent, solid flake. The flake is reduced in size in a double roll flake breaker before being sent to the compaction screens.

Glazing of the granular product produces a hard surface on the granules to help prevent product breakage and dust production in post treatment handling and dispatch. In the glazing circuit, water is sprayed through nozzles onto the product in a conditioning drum before entering a fluid bed dryer/cooler. The water addition to the conditioning drum is regulated to maintain 1.5 to 2% product moisture by weight. The conditioning drum discharges directly to the dryer/cooler. The dryer section uses heated fluidizing air to heat the product to 160°C. This drives off excess moisture which leaves the particles with a hardened glazed surface. The cooler section uses plant air (utilizing a separate fluidizing fan) to cool the product to 90°C before dispatch.

The product is then sent to two multi-deck glazing screens (room is left for the addition of a third glazing screen for future expansion) for removal of the fines and oversize. The screens will produce three size fractions. The oversize is fed to the glazing cagepaktor before being recirculated to the glazing screens. The undersize is returned to the compaction circuit. The on-spec granular product is conveyed to product storage.

15.11 Loadout and Product Storage

Standard product is discharged from the standard cooler onto the standard stockpile belt conveyor, while granular product is discharged from the granular product belt conveyor onto the granular storage product belt conveyor. Due to the high humidity at the process plant site, both standard and granular product will be treated with an anticaking agent (a mixture of de-dust oil and anticake amine) to prevent product setup in the storage building. Standard and granular products are transferred to separate storage areas within the product storage building.

From product storage, either standard or granular product can be reclaimed via four product reclaim belt conveyors (two for standard product and two for granular product) at a rate of 1,525 tonnes per hour (tph). Standard and granular products are screened to remove oversize and undersize material prior to being transferred to ship loading. Oversize from the granular product screens is sent to the glazing oversize cagepaktor. Undersize from the granular screens and oversize from the standard screens is sent to the glazing dryer/cooler. A final dose of anticaking agent is applied to both standard and granular products prior to being dispatched to the transfer belt conveyor.

15.12 Water and Seawater

Process water will be delivered to the process plant from the infrastructure developed by EGIS. Process water will be used for centrifuge screen washing, reagent mixing, gland water, grade control water, glazing water, column cell water, and equipment cleanup.

Seawater will be delivered to the process plant from the infrastructure developed by EGIS. Seawater will be used for wet scrubber scrubbing liquid, salt cake re-slurrying and dilution of the slimes thickener underflow.

15.13 Brine Management

With the principal components of potash ore being halite ($NaCl$) and sylvite (KCl), a double saturated $NaCl$-KCl brine system is formed when ore is mixed with water. At the beginning of operation water will be used to slurry the ore. Once a sufficient volume of double saturated $NaCl$-KCl brine is produced transportation and unit processes are performed in saturated brine solutions to avoid large recovery losses. Once in the process plant, the saturated brine solution (process brine) is recirculated many times and fresh water addition is minimized.

Before the salt is discharged into the ocean the salt slurry is centrifuged to remove as much of the brine as possible (the centrifuge cake is debrined to 95 - 96% solids by weight). The brine recovered in the centrifuges is cycloned to remove any coarse solids and the cyclone overflow is pumped to the process brine distribution tank. The overflow from the slimes thickener is also sent to the process brine distribution tank. Brine from the RSF is also returned to the process.

Any excess brine is bled off to the salt disposal tank. Sometimes a small quantity of brine is needed to be bled out of the system to control the buildup of deleterious elements in solution. This brine bleed would also be sent to the salt disposal tank.

15.14 Reagents

Reagents used in the process are received, mixed where necessary, and stored in a purpose-built facility adjacent to the mill, from where they are dispatched to the various areas of the process as required. The facility contains the requisite unloading facilities, storage tanks, mixing tanks, mixing equipment, dispatch and metering pumps for the respective reagents in the process.

15.14.1 Flotation Amine

Amine is used in the flotation process to selectively promote the adherence of KCl particles to the air bubbles while inhibiting the adherence of NaCl particles. The KCl particles float to the surface with the air bubbles while the NaCl particles sink to the bottom. Amine needs to be maintained within a tight temperature band for optimum effectiveness; therefore, it is necessary to insulate the storage and mixing tanks and add heat provided by heating coils within the tanks. Amine is neutralized with acid, most likely hydrochloric, in the heated, insulated mixing tanks prior to being pumped to various areas of the process.

15.14.2 Flotation Oil

Flotation oil assists the flotation amine to make coarse particles of KCl more hydrophobic. Flotation oil is sometimes called an extender or promoter. Flotation oil is pumped and added to flotation feed slurries without dilution.

15.14.3 Frother

Frother is added to flotation feed slurry to increase the strength of the air bubbles in the flotation machines. The stronger bubbles allow the potash particles to stay at the surface longer, increasing the likelihood that they will be swept over the overflow as concentrate. Frother is pumped directly from the holding tank to the reagent addition points without dilution.

15.14.4 Depressant

Depressant is added to the flotation feed slurry to coat the trace amounts of insolubles remaining after desliming. Depressant reduces the absorption of flotation amine on the insolubles. This reagent reduces the quantity of amine required to float potash particles, and also reduces the quantity of insolubles floating with the potash particles.

Depressant is delivered as a powder to site in bulk bags. To mix, powdered depressant is added to water by eductors or other specialized wetting equipment. After wetting, the solution is mixed into brine in the mix tanks. Retention time and agitation are necessary to thicken the mixture. Positive displacement gear metering pumps deliver it to the flotation process as required.

15.14.5 Hydrochloric Acid

As noted above, hydrochloric acid is used as a component of the flotation amine process. Hydrochloric acid is delivered to site in liquid form and is pumped to the flotation amine mix tank.

15.14.6 Dispersant

Dispersant is added to the hydroclassifier feed to prevent fine particles from agglomerating so that they remain in the fines stream and can be separated from the coarser particles. Dispersant is delivered to site in bulk bags. The dispersant will be mixed with water in an agitated tank before being diluted with process brine and being stored in the holding tank.

15.14.7 Flocculant

Flocculant is used to coagulate fine particles in the slimes thickener in order to promote settling of the fine tails. Flocculant is delivered to site in bulk bags in powder form. The flocculant is added to water by eductors and mixed in an agitated tank. The flocculant is then transferred to a separate holding tank and is further diluted with brine. From here it is pumped to the various process points.

15.14.8 Anticake Amine

Anticake amine prevents product particles from bonding together after periods without movement. This enables product to flow easily when retrieved from product storage, and also after transport. Insufficient application of anticake amine can result in serious handling difficulties for the customer, depending on transport conditions and duration.

As with flotation amine, care must be taken to ensure the amine temperature is held at a temperature suitable for pumping.

15.14.9 Dedusting Oil

Similar in composition to lubricating oil, dedusting oil provides a surface for dust adherence, preventing dusty conditions as product is handled. The oil also helps the anticake amine to evenly coat the product particles.

Prior to application, anticake amine is mixed with dedusting oil in a heated mixing tank in the loadout area. The solution (called anticaking agent) is sprayed onto and mixed into product before it is transferred to product storage and just before it is conveyed to the jetty for ship loading.

Month No.		0	1	2	3	4	5	6	7	8	9	10	11	12
Process Plant Throughput	tph	0	400	450	500	600	700	800	900	900	900	900	900	900
Running Time	%	0	60.0%	75.0%	80.0%	85.0%	86.8%	86.8%	86.8%	86.8%	86.8%	86.8%	86.8%	86.8%
Commissioning Complete			Production Begins											

srk consulting

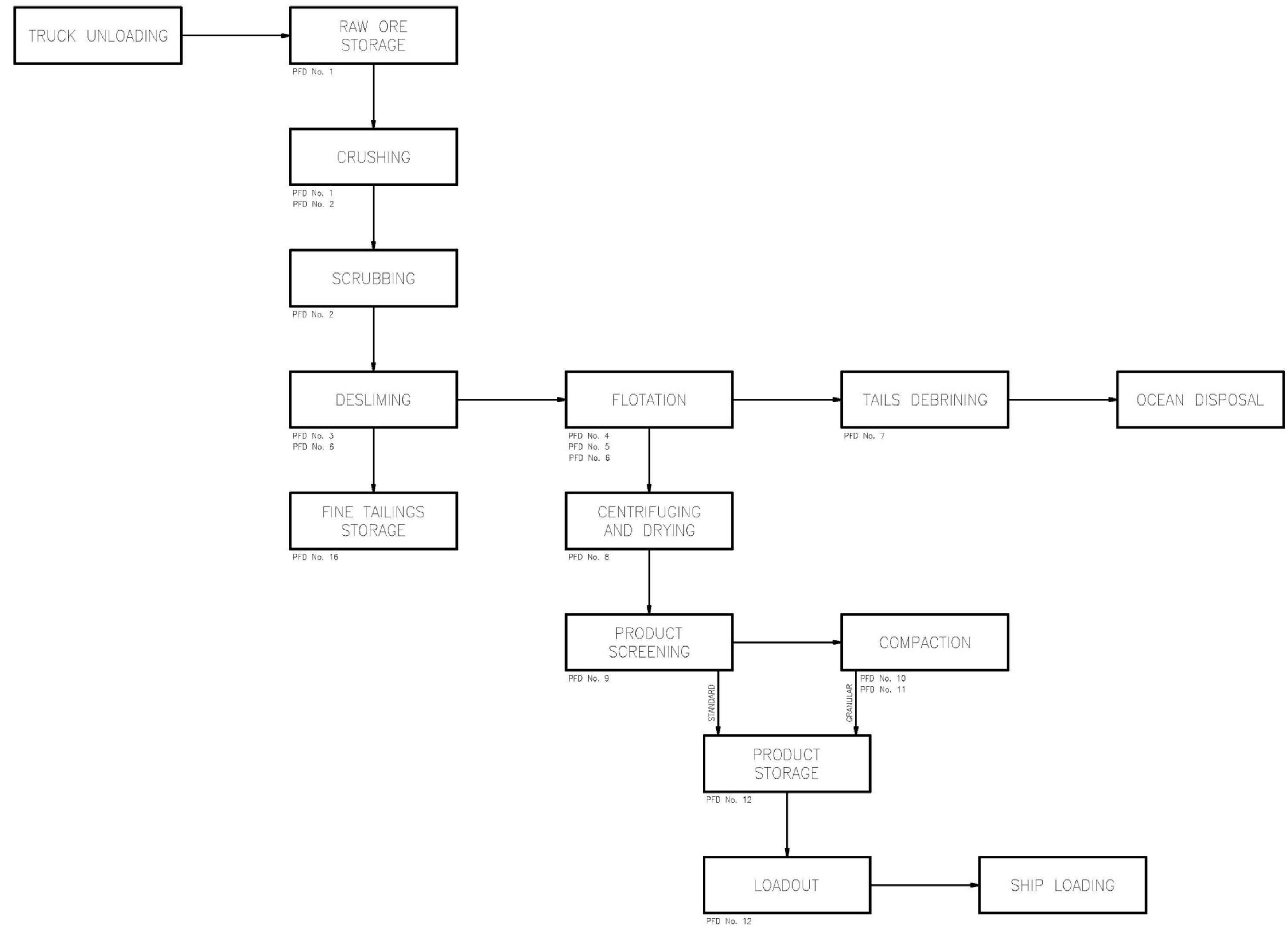
elemental minerals limited

Sintoukola Potash Project
Republic of Congo

Source: AMEC, 2012

Figure 15-1

Typical Ramp up



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Sintoukola Potash Project
Republic of Congo

Source: AMEC, 2012

Figure 15-2

Block Diagram of the Plant

16 Project Infrastructure (Item 18)

The project infrastructure facilities, shown in Figure 16-1, can be broken down into the following areas:

- Mine site facilities;
- Haul road and road trains;
- Process plant site facilities;
- Marine facilities;
- Employee facilities; and
- General infrastructure (power, natural gas, water and access roads).

Key design inputs include the following:

- A geotechnical and geophysical program (onshore and near shore), a topographical LIDAR survey including aerial orthophotography, two bathymetric surveys, a beach topographical survey and metocean survey, were performed during Q4 2011 to support the PFS design for the haul road, the mine, process plant and employee facility areas, the marine facilities and associated infrastructure. A detailed assessment of the site characteristics (geophysical, geotechnical, morphology) was obtained in order to develop site specific designs;
- Grading plans, platforms and surface water controls were incorporated into the design to minimize erosion; and
- A WBS was developed to confirm that the various elements are included in the Economic Model.

The results of infrastructure design and costing are presented in Volume VIII (EGIS, 2012).

16.1 Mine Site Facilities

The mine site is located at the Kola deposit, 36 km from the process plant consisting of the following key facilities:

- Platform: The overall area covered by the mine site platform is approximately 12 ha. Three platform section types were allowed for in the design and costing, which consists of the following:
  - Paved platform;
  - Unpaved platform; and
  - Building platforms.
- Buildings. The onsite buildings include administrative (including medical and security), training, waste treatment, logistics and transfer and mobile equipment maintenance facilities. The total floor area of the surface building facilities at the mine site is approximately 9,400 m^2. The buildings related to shaft access are addressed by SRK in Section 14.7.3 and buildings related to the process plant are addressed by AMEC in Sections 15.1 and 15.2. The surface buildings are generally of two types:
  - Office and associated buildings (mess halls, offices, training rooms, changing rooms) will be modular construction; and
  - Industrial buildings (workshops and storage buildings) will be conventional steel structures.

- Power distribution. From the primary medium voltage (MV) substation at the mine site, two power transmission lines will supply the mining operation, an additional 22 kV distribution loop will feed the MV/LV transformers installed for the office buildings;
- Water supply. Fresh water will be sourced from the groundwater, and a potable treatment plant will be used to supply potable water;
- Waste water treatment. Domestic wastewater will be collected and sent to a holding tank before treatment, which will be by biodegradation plus filtration, prior to being disposed in a leach field. The sludge will be removed for external treatment;
- Solid waste treatment. All recyclable materials (i.e., paper and cardboard, plastics and metals, etc.) will be collected separately and pressed into bales at the process plant site. Bales will be hauled by truck to their respective external recycling locations. Any organic matter will be composted, with the remaining solid waste sent to a public sanitation landfill; and
- Industrial waste management. Waste oil will be collected and stored before being removed for external treatment. Oily water will be treated by desander/de-oiling units before being transported to the industrial wastewater treatment plant (IWWTP) located at the process plant site.

16.2 Haul Road and Road Train

A trade-off study was performed at the beginning of the PFS to determine the optimal location for the process plant, with the following possibilities considered:

- Process plant at mine site: Hauling 2 Mtpa of final product along public roads; and
- Process plant at coast: Hauling 6.85 Mtpa of run of mine ore from the mine along a new, dedicated road called the "Haul Road".

As part of the PFS design and costing, the decision was taken to construct a dedicated 36km haul road and locate the process plant at the coast which required large tonnage trucks called "road trains" to transport the ore economically. The geometry (width of 12 meters, curves and slopes) of the haul road and its pavement structure have been designed to account for the size and type of haulage equipment used and the site specific geotechnical conditions.

The haul road alignment was based on the geotechnical and hydrological conditions along the haul road route. Therefore, a design was developed to minimize the erosion potential (embankment cut and fill slopes) and upgradient watersheds, and the road alignment was developed to follow ridgelines as much as possible to decrease the impact of water flow from watersheds. As the haul road crosses several existing tracks, seven grade-separated crossings were allowed for in the design to avoid any non-mining traffic on the road.

EGIS performed a study to identify the preferred road train configuration, and several different truck and trailer axle loading configurations were considered in order to optimize the hauling cost and pavement structure. The selected road train configuration will consist of a truck and two side dump trailers with a combined payload capacity of 152 t. The trucks will be loaded and offloaded in covered loadout facilities designed by AMEC. The road train fleet has been calculated to require approximately 21 road trains (spare units included) with a total of 140 round trips each day and an average hauling rate of 900 tph.

16.3 Process Plant Site Facilities

The process plant will be located at the coast, with the following key items:

- Platform. The overall area covered by the process plant site platform is approximately 20 ha. Due to the high water table expected at this site, EGIS has included gravel materials in the design and costing that will improve foundation conditions and enhance surface water drainage from platforms. Three types of platform section were allowed for in the design and costing, which consist of the following:
  - Paved platform;
  - Unpaved platform; and
  - Building platforms.
- Buildings. The onsite buildings include administrative, mess hall, medical and security, training rooms, water supply, waste treatment, workshops and storage buildings. The total floor area of the process plant facilities was estimated to be approximately 66,700 m^2, excluding the actual process plant, reagents storage and product storage areas that were part of AMEC's scope discussed in Sections 15.3 through 15.14. The surface facilities are of two types:
  - Office and associated buildings (mess halls, offices, training rooms, changing rooms, medical and security buildings)which will be modular construction; and
  - Industrial buildings (workshops, storage buildings, power distribution building, water supply building, waste treatment buildings)will be conventional steel structures.
- Maintenance and fueling facilities. Haul truck maintenance and refueling will generally be done at the process plant site, and allowances were made for appropriately sized facilities;
- Power distribution. From the primary MV substation at the process plant, two power transmission lines will supply the process plant, and an additional 22 kV distribution loop will feed the MV/LV transformers installed for the office buildings;
- Water supply. Fresh water will be sourced from groundwater wells and a potable treatment plant will be used to supply the potable water;
- Firefighting. A firefighting system was included at the process plant site, consisting of a gravity fed water storage tank and buried pipeline system with hydrants;
- Waste water treatment. Domestic wastewater will be collected and stored in a holding tank before treatment. Treatment will be by biodegradation plus either filtration or clarification, with co-disposal into the ocean with the salt brines. The sludge will be removed for external treatment;
- Solid waste treatment. Eighty percent of the waste produced at the sites will be handled by recycling and by the composting of organic matter. The remaining twenty percent will be sent to a public landfill. All recyclable materials (i.e., paper and cardboard, plastics and metals, etc.) will be collected separately and pressed into bales, which will then be hauled by truck to their respective external recycling location; and
- Industrial waste management. Waste oil will be collected and stored before being removed for external treatment. Oily water will be treated by desander/de-oiling units before being transferred to the IWWTP. The remaining sludge will be removed for external treatment.

16.4 Marine Facilities

The bathymetric conditions along the coast are not favourable for the direct loading of ocean-going vessels in the 35,000 deadweight tons (DWT) and larger category. Therefore, a transshipment solution has been selected, which involves loading the potash from a jetty terminal into barges; barges then travel to the ocean-going vessel moored in deep water (minimum draft of 15 m), and unload the potash directly into the holds of the vessel.

The jetty consists of a 750 m long steel structure supported by piles; the conveyor and feeder at the end of the jetty will be installed on this structure. The product will be conveyed at a rate of 1500 tph via the jetty onto two barges, each with a 5,000 t capacity. Mooring berths are provided on either side of the jetty. The loading of a 35,000 DWT vessel will require seven barge load rotations, which will take approximately 53 hours to fill. On average, one vessel will be loaded per week, which implies an overall utilization of the loading facilities of 35%, and allows sufficient excess capacity for adverse weather conditions, delays, etc. Barge operations will require the support of a tug/workboat and navigational aids.

The barge loading operation requires the construction of a 250 m long breakwater that will significantly reduce wave disturbance in its lee. Four breakwater options were considered and the selected breakwater option consists of twelve reinforced concrete caissons. This option improves both construction schedule and constructability, as the caissons can be pre-constructed at the port of Pointe Noire, towed to site and sunk on a bedding layer on the seabed with concrete heavy ballast.

16.5 Employee facilities

The employee facilities are located near the coast, with key items as follows:

- Buildings: The employee facilities, or base camp, covers an area of approximately 10 ha and is located at a distance of 5 km from the process plant site. These facilities are sized to accommodate 950 persons (both skilled and unskilled workers) and are composed of the following five major functional entities:
  - Mess hall;
  - Recreation;
  - Accommodations. Various levels of modular construction depending on worker category:
    - Collective housing units (eight room shared clusters), that will consist of modular units with covered passageways (concrete floor, steel structure and steel deck roof) linking the housing area together; and
    - Individual housing units that will consist of two person apartments, one person apartments and villas for management.
  - Technical area;
  - Reception; and
  - Logistics.
- Power distribution. The existing facilities and water facilities located near the process plant will be fed by 22 Kv line coming from the HV coastal sites substation.

16.6 General Infrastructure

16.6.1 Power

A trade off study was performed to investigate various onsite and offsite power supply options and the preferred option is discussed below.

Electric power will be supplied from the ROC national power grid, shown in Figures 16-2 and 16-3. From the Mongo Kamba II 220 kV substation in the suburbs of Pointe Noire, a new 220 kV transmission line 57 km in length will be constructed to supply power to the process plant and employee facility sites. From the HV coastal site substation, another new 220 kV transmission line 35 km in length will be constructed to supply power to the mine site.

The HV substations will be located at a distance of 2 km from the shaft or the process plant to avoid corrosion by salt and to allow for the installation of the Air Insulated Switchgear (AIS). From these power substations, multiple parallel 22 kV buried cables will be run to the primary MV substations located on each site.

Demand for electrical power is estimated at 24 MVA at the mine site and 32 MVA at the process plant and employee facility sites.

16.6.2 Natural Gas

AMEC estimated that 20 million cubic meters (Mm^3) of natural gas per annum will be needed for the product drying process. Therefore, EGIS sized a 150 mm nominal diameter pipeline that is 81 km in length, shown in Figures 16-2 and 16-3. This pipeline will run from the gas treatment plant at Cote-Mateve, located 10 km south of Pointe Noire, to the process plant.

16.6.3 Water

Two water sources will be required for the Project:

- Fresh water. Both the mine and process plant sites will be supplied by wells; and
- Sea water. Approximately 9,700 cubic metres per hour (m^3/hr) of seawater will be needed for process makeup water (1,085 m^3/hr) and for dissolution (8,550 m^3/hr) as part of the sea brine management. The seawater required will be supplied via a screened seawater intake installed approximately 400 m from the shoreline.

16.6.4 Site Water Balance

SRK developed a site wide water balance that summarized the PFS water balance calculations performed by SRK, EGIS and AMEC for the Sintoukola Project. The water balance requirements and associated flow rates were based on a production rate of 2.0 Mtpa MoP. The water balance summary confirmed that consistent design assumptions and criteria were used between the different project discipline battery limits. Refer to Volume XI (ELM, 2012) for details.

16.6.5 Access roads

Access to the project area is by means of a dual-lane bitumen road called RN5, which runs for 50 km from the city of Pointe Noire to the north of the town of Madingo-Kayes. From the town, the final 40 km to the mine site is via a sandy track road. Three other sections of road are needed to be established or upgraded, which are as follows:

- Existing Track 1 (ET1): The ET1 extends from North of Madingo-Kayes (end of the RN5) to the future mine site. This road section is approximately 35.5 km long. This road is presently an existing sand track of various widths, and it crosses several villages between Madingo-Kayes and Koutou-Kobambi. Due to the current state of existing tracks and conditions during the rainy season, access to the mine, process plant and employee facilities will have to be improved during construction, as well as for operations. Provisions have been made for initial upgrading of the road to serve construction requirements and then permanent improvement after the construction phase;
- Existing Track 2 (ET2): A section of ET2, called ET2b, which extends from the employee facilities site to the Haul road (CH31), will be upgraded, and is approximately 2 km long. The length of this road section could be shortened in the FS stage depending on the final location of the base camp. This road is presently an existing sand track of various widths and must be upgraded to allow the traffic of trucks supplying the employee facilities area; and
- Service road (NT): Since 10 kilometers of the haul road will be used for access to the process plant site, a part of NT, called NTa, which extends from ET1 (CH24.5) to the Haul road (CH24), will be upgraded. Approximately 1.5 km of road will then need to be upgraded. The remaining section of the NT will be maintained to ensure access to the coastal sites during the construction period.

Prior to the construction phase, the roads will be repaired seasonally (mainly drainage and some earthworks) to allow for year round traffic. Once production starts, these roads will be upgraded along their entire length to allow for improved traffic conditions for the operational phase.

16.7 Waste Disposal

16.7.1 Disposal Philosophy

Ocean disposal of highly saline brines is an accepted practice for desalination plants and has also been approved for the proposed MagMinerals operation in the ROC based on compliance with International Finance Corporation (IFC) effluent guidelines. Ocean disposal relies on post-discharge dilution; this approach is considered generally acceptable because the brine is a concentrated version of the elements that are already present in seawater and outside of the natural mixing zone the brine is undetectable (typically a short distance from the point of discharge).

Initial data indicate that the waste brine from processing of the Sintoukola ore will comply with IFC Environmental, Health and Safety effluent discharge guidelines and can therefore be safely discharged to the ocean with limited impacts on sensitive marine life and seabed habitats. To further minimize the impact of the waste brine, pre-discharge dilution will also be applied (at a ratio of 1 volume of waste brine to 5 volumes of seawater); pre-discharge dilution will reduce salinity of the waste brine to approximately 125 gpl (compared to seawater salinity of approximately 36 gpl).

Low volumes of solid waste from processing will be stored on land in the RSF. Initial data indicate that excess supernatant from the RSF can be co-disposed with the waste brine into the ocean.

Further modelling will be undertaken during the FS to model that natural mixing under the full range of offshore conditions can effectively dilute the ocean discharges (waste brine and excess RSF supernatant), that there is no significant impact beyond the mixing zone and that the impacts within the mixing zone will be acceptable.

16.7.2 Brine Management

The salt brine, generated as a waste product in the process plant, will be dissolved and diluted with seawater before being discharged into the ocean. The salt brine will be diluted with sea water, resulting in a maximum salt content of 125 grams per litre (gpl) following dilution. The diluted sea brine will be pumped via a pipe attached to the jetty, and discharged into the ocean via diffusers installed in the seabed; this is a standard approach for desalination plants. Numerical modelling calculated that the salt concentration will be reduced to within ±2.4 gpl of the background ocean salt concentration within 250 m of the diffusers.

16.7.3 Solid Residue Storage

The insolubles will be stored in a valley impoundment, referred to as the RSF and shown in Figure 16-2. The RSF will be constructed in phases, initially constructed with a single embankment and followed by five raises, to ultimately contain 1.66 Mm^3 of insoluble material. Raises will be performed in three to four year increments to accommodate the current LoM plan and minimize initial capital costs. A diversion channel was designed to convey water around the perimeter of the RSF, and minimize the amount of upgradient surface water mixing with supernatant solution.

Geotechnical testing on the insoluble material indicated that separation of the solids and liquids does occur and therefore decanting of the water should be possible. Geochemical testing of the insoluble material suggested that a liner would be required, and the RSF was designed with a geosynthetic liner to prevent any seepage into the environment.

The starter embankment was designed to accommodate a minimum of 2 years' deposition, based on a production of 7.74 tph of insolubles. Starter embankment fill will consist of silty sand material obtained from within the RSF footprint, or from within 2 km of the RSF site. Raises above the compacted starter embankments are to be constructed with a similar silty sand material.

The design assumes spigot discharge from the main embankment, a beach angle of 1% and an in-place dry density of 0.9 tonnes per cubic metre (t/m^3).

A monthly water balance has been prepared for the site, for the LoM to estimate the volume of makeup water that can be supplied back to the process plant. This estimate was based on the available information and interpreted design parameters, or assumed where necessary. The inflows to the RSF area include residue slurry water, direct precipitation and runoff. Water losses are through evaporation and interstitial losses within the residue body. The average excess water that can be pumped to the brine distribution tank is approximately 54 m^3/hr. However, it should be noted that during the wet months, the required daily pumping rates will be higher than during the dry months. Towards the end of the LoM (final 5 years), the residue beach area is sufficiently large that, during the dry months, the losses will be greater than the inflows, and therefore no pumping to the brine distribution tank will be required.

The results of geotechnical testwork, along with the RSF design and costing details and assumptions, are presented in Volume VII (SRK, 2012d).

16.8 Communications

The ROC has an extensive cellular network comprising four operators using the GSM network standards.

Shevon Holdings was commissioned by ELM to design and cost a communication solution for the Sintoukola Project. The main areas considered include the following:

- Process plant and port site;
- Employee camp;
- Mining operations:
    - Surface; and
    - Underground.
- Camp;
- Roads and haulage vehicles;
- Haulage vehicle tracking;
- Ponte Noire office; and
- Johannesburg office.

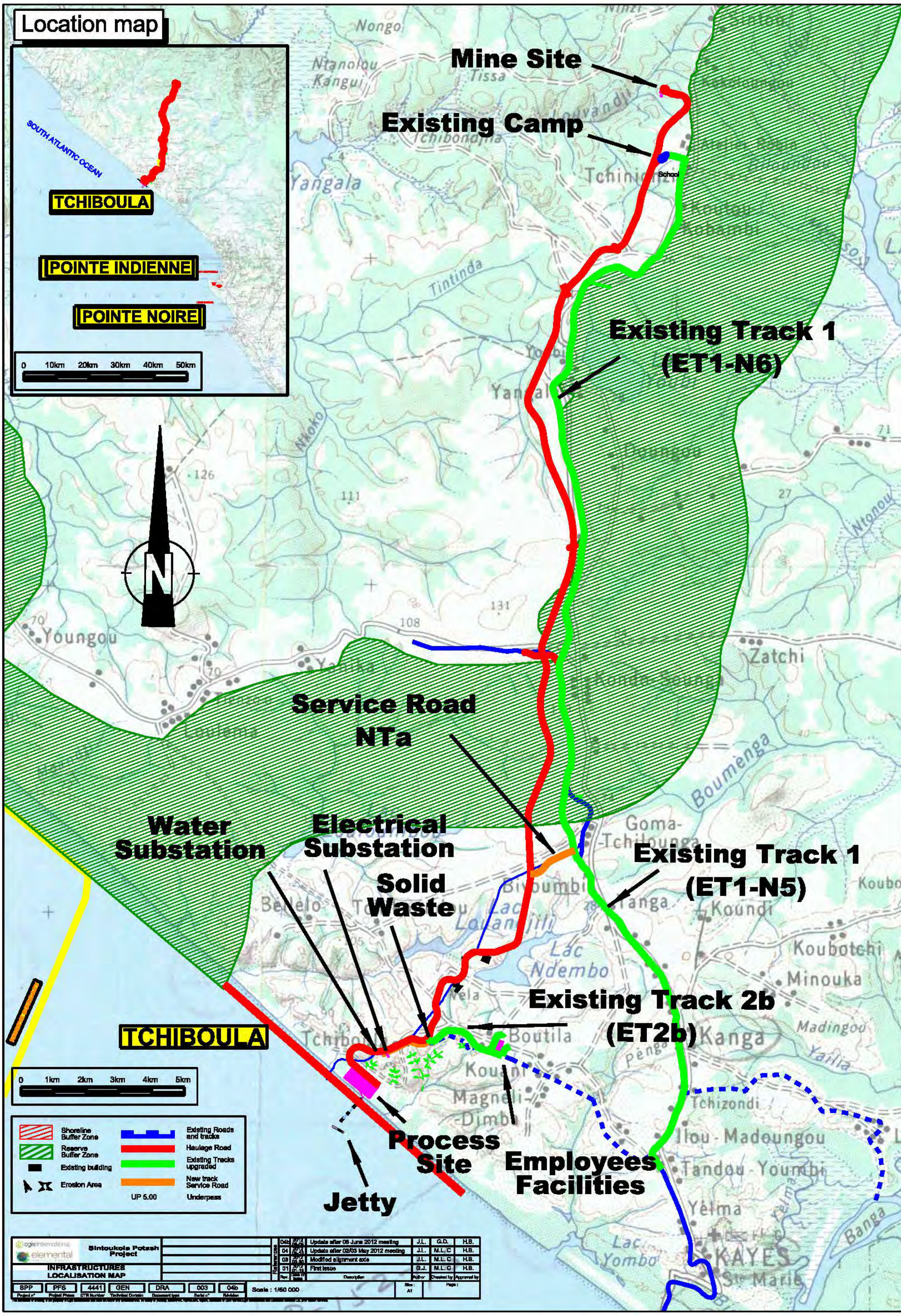
Communications for the Sinotukola Project will combine the following major technologies:

- Satellite Link, VoIP and IP Linking. Connection to the process plant and port site area, employee facilities, mining area (surface and underground), exploration camp and Ponte Noire office will be via satellite links to various sites. For reasons of redundancy, efficiency and control, there will be a Satellite dish at the main points each receiving its own bandwidth. In principle, the VSAT's would supply the following:
    - Internet, E-Mail and VoIP to the areas required (mine, process plant, exploration camp and employee facilities; and
    - IP link facilitating communications between the different Mine areas, Ponte Noire and Johannesburg offices via VoIP and if necessary direct remote connection.
- VHF Radio Trunking solution supplying radio communications, to all areas noted above. Further the radio network will form the basis of the haulage vehicle tracking system.
    - Network Design. To provide the coverage expectancy, three radio hi-sites were identified:
        - Hi-Site 1: This will cover the process plant and part of the residential area, as well as contribute coverage along the road;
        - Hi-Site 2: Provides coverage along the road and into the residential area; and
        - Hi-Site 3: Provides surface coverage at the mine and the exploration camp area.
    - Mobile Radios. The TM8255 MPT1327 trunked radio is ideal for a wide range of voice and data applications where comprehensive trunked services are required. Each mine vehicle will be installed with a mobile radio and a URM for AVL purposes.
    - Handheld Radios. Hand held radios will be provided to be used on the surface as well as underground.
    - Licensed Microwave Radio. Licensed microwave radio will be used to connect the three hi-sites as well as the mine and process plant. This solution is based on a 1,4

GHz with a channel size of 1,75 MHz, using 64QAM modulation providing 8,362 Mbps of capacity.

  - Unlicensed Microwave Radio. Additional microwave links that do not form part of the critical network between the mine and the process plant will be unlicensed microwave links.

- A microwave backbone linking the mine, process plant and employee facilities. The Microwave link will be IP based used to carry the VHF converted radio signals as well as the VoIP data traffic and any IP linking traffic that may further be required in the future. Future requirements may result in live IT linking between site networks, in which case the microwave network would facilitate this as well.



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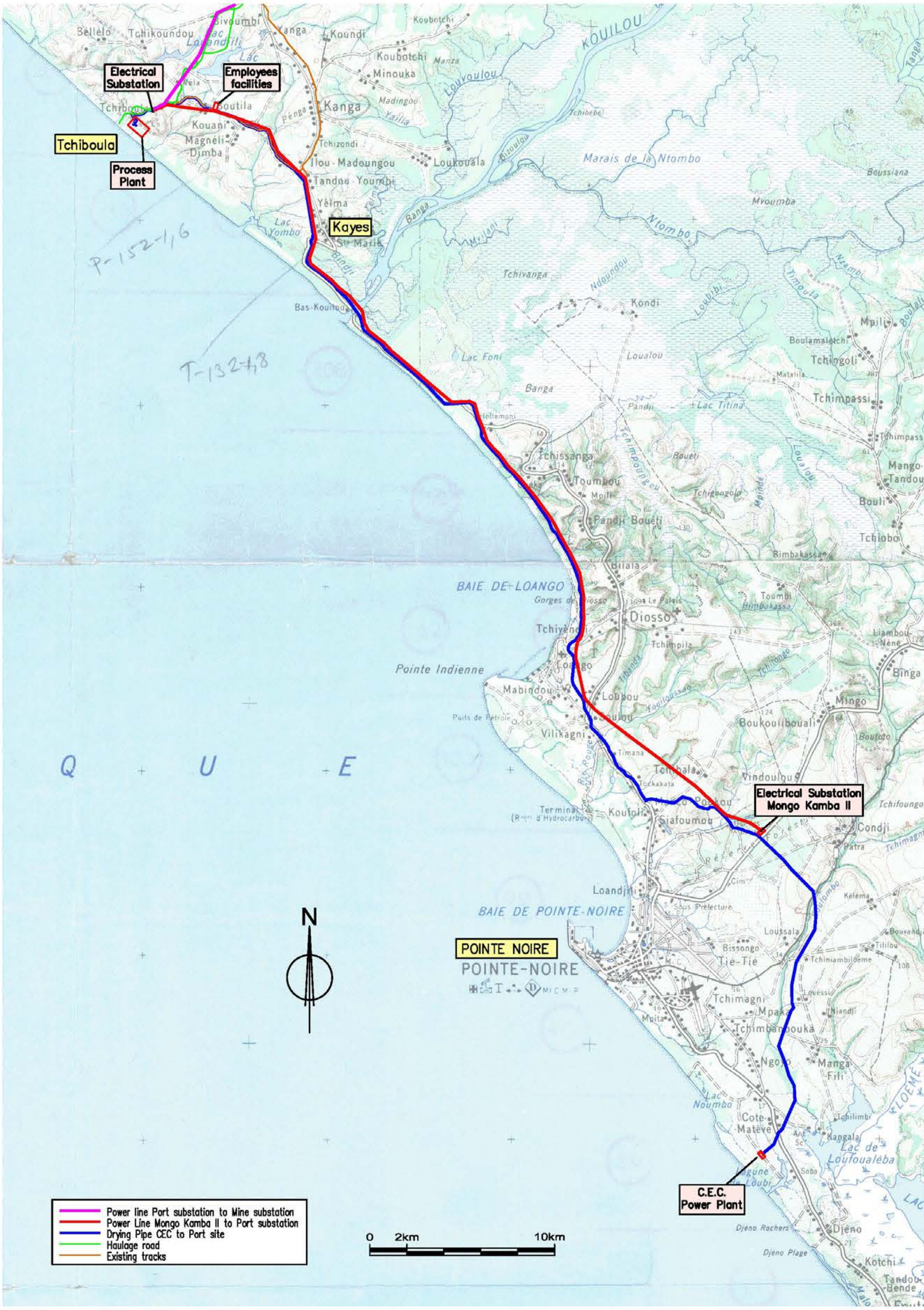
Sintoukola Potash Project, Republic of Congo

Source: EGIS, 2012

Figure 16-1

Project Infrastructure



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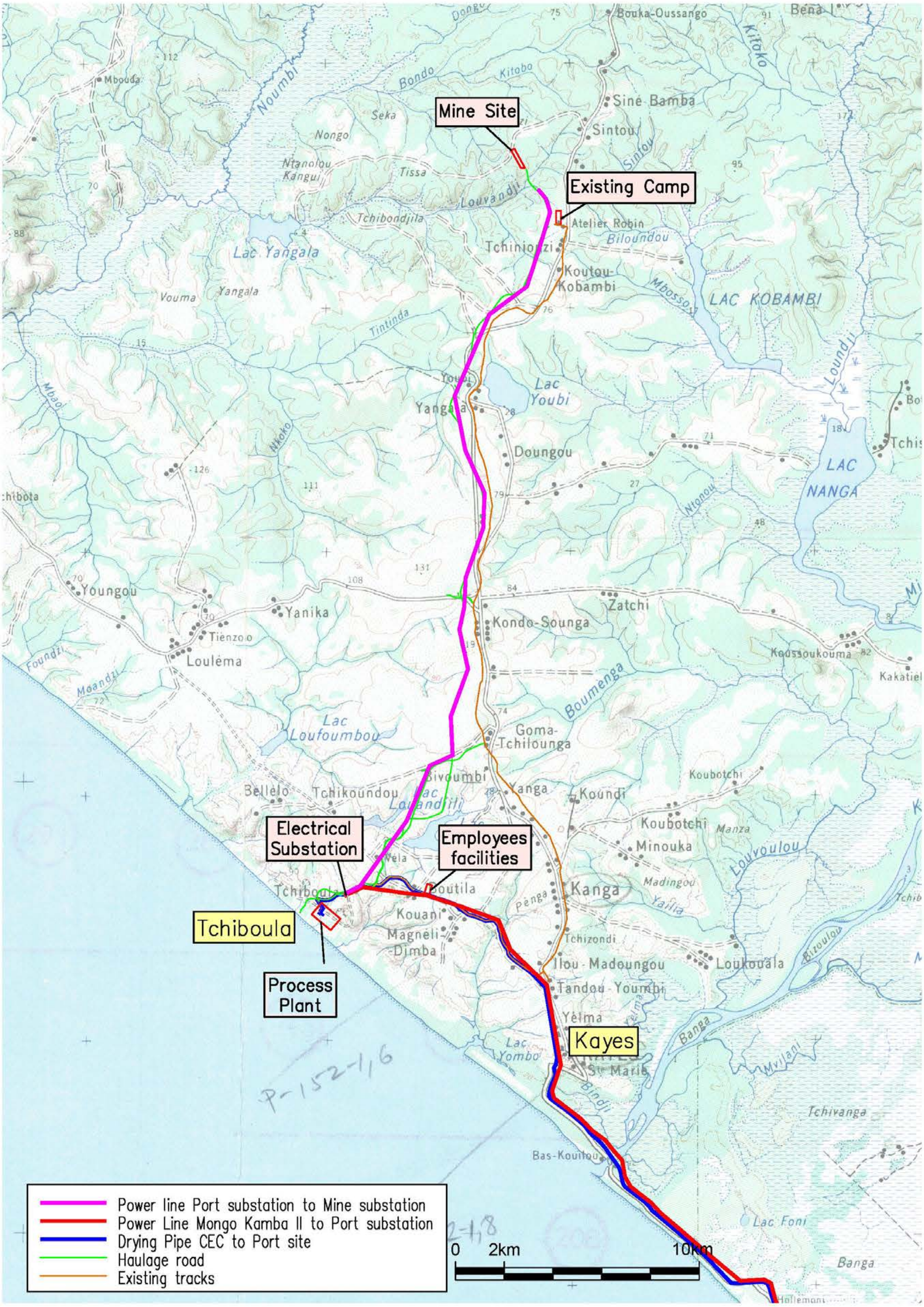
Sintoukola Potash Project, Republic of Congo

Source: EGIS, 2012

Figure 16-2

Power and Gas Lines Between Process Site and Pointe Noire



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Sintoukola Potash Project, Republic of Congo

Source: EGIS, 2012

Figure 16-3

Power and Gas Lines Between Process and Mine Site

srk consulting

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Sintoukola Potash Project, Republic of Congo

Source: SRK, 2012

Figure 16-4

RSF Location

17 Market Studies and Contracts (Item 19)

17.1 Summary of Information

Potassium (K) is one of the three fertilizer nutrients essential for agricultural output and demand is therefore driven by projected global food consumption.

ELM utilized market research from CRU and Fertecon Limited (Fertecon) to develop its potash marketing strategy. Both companies are highly respected independent potash commodity research analysts, utilized by potash industry participants. The reports covered all aspects of potash supply, demand, marketing, potash logistics and pricing.

Both companies foresee a strong growth of both demand and supply in potash in the next decade.

For the purpose of the PFS, Brazil is considered to be the target market for product from the Sintoukola Project. Projected growth in the Brazilian potash demand is sufficient to absorb all the Sintoukola production.

The preferred product in the Brazilian market is granular material, which will form the bulk of the production from the Sintoukola Project, at a targeted MoP grade of 60.5% K_2O. Fertecon provided Cost and Freight (CFR) price projections for the Brazil market through 2020. Thereafter, the 2020 price was kept constant.

17.2 Nature of Material Terms

No material contracts are in place at this time.

17.3 Relevant Market Studies

ELM commissioned CRU International Limited (CRU) to undertake a client specific study of the potash market and the various marketing requirements for potash sales into key markets.

ELM purchased Fertecon's Potash Outlook report that formed the basis of pricing assumptions used.

Both reports have copyright protection, but the following extracts have been approved for inclusion in this PFS report.

17.4 Commodity Price Projections

17.4.1 Demand

As can be seen from Figure 17-1, CRU predicts global potash demand to grow to over 73 Mtpa by 2020. This demand growth is driven primarily by emerging markets with large populations and strong agricultural sectors. China, Brazil, India and SE Asia all show significant growth, while the USA and the rest of the world will remain large and steady consumers, but without much growth. This view is confirmed by Fertecon, who predict a market demand of 75 Mtpa by 2020.

17.4.2 Supply

By the end of 2010, the world potash industry had capacity for 65.5 Mtpa, based on more than forty-five operations in thirteen countries. In the five years since 2005, this installed production capacity has increased by 8.4 Mtpa, mostly through the expansion of existing production sites in Canada, China and Russia. Two thirds of the current capacity total is located in just three countries – Canada, Russia and Belarus. Worldwide, ten companies control 94% of the total capacity, with the three largest accounting for one half of the total.

Figure 17-2 shows the potash production and origin over the last two decades, and projected production for the next decade. It can be seen that CRU expect an addition of approximately 35 Mtpa of potash production capacity to the market between 2011 and 2020. Similarly, Fertecon predict a growth of 32 Mtpa in capacity over the same period.

Figure 17-3 shows the key role that a few major producers play in the supply of potash. Approximately one third of global capacity is controlled by Uralkali and Belaruskali in the Former Soviet Union and another third of global production comes from the Saskatchewan deposits through Agrium, Mosaic and PotashCorp. All three companies sell their product as members of Canpotex. ELM has the opportunity to supply potash from a new production district, unencumbered by existing marketing and export associations.

17.4.3 Target Market

Five target markets were considered in detail by CRU (Brazil, China, India, South-East Asia and South Africa), based on following observations:

- Brazil’s import of potash from Russia, Belarus, and Canada exceeded 5.5 Mt in 2011, and is forecast to expand consumption by an additional 2.4 Mtpa by 2017. The Sintoukola Project’s location immediately across the Atlantic ocean provides a huge potential shipping advantage, and the corresponding market demand suggests that SPSA should have a strong market potential, provided that similar, or lower, operating costs can be achieved;
- Even though South Africa is a relatively small potash market, it is the largest African market for potash and could certainly absorb some of the production, particularly considering that the major producers do not have a strong presence. The Sintoukola Project would have a significant shipping advantage, having a US$10/t freight advantage over Jordan and Israel, which are otherwise the most competitive producers for this market;
- China offers a huge potential market, but Canada has the most competitive seaborne freight and Russia transports much of its potash by rail to offset their ocean freight disadvantage;
- India is a huge market for the major producers, but unlike China, India has few barriers to entry and the distribution network is capable of handling new entries into the market. Middle Eastern producers are the most competitive supplier to India in terms of shipping rates, followed by the Sintoukola Project. Russia and Canada have a considerable disadvantage in this market compared to SPSA, with the shipping differential estimated to be in the order of US$12 to US$17/t and US$15 to US$19/t, respectively; and
- South-East Asia is a booming market for potash and consumption is forecast to grow by 1.2 Mtpa over the next five years and a further 0.8 Mtpa by 2020. The Middle East is again the most competitive supplier, followed by the Sintoukoula Project. This market is widely dispersed geographically in terms of field application and logistics, and suppliers will be able to supply this market in several ports, which is an opportunity to SPSA.

The research indicates that ELM should focus exports on Brazil and South Africa, mainly due to highly competitive freight rates and expected ease of entry into these markets. India, China and

South-East Asia also offer promising options, with SPSA being more competitive than most of the other major producers.

17.4.4 Recommended Mode of Sale

SPSA has the options of selling its product on an FOB Pointe Noire basis, establishing its own in-market presence with offices and employees and/or agents, or a combination of both. SPSA is currently assessing each of these options. An appropriate allowance has been made in the TEM as recommended by CRU.

17.4.5 Product Quality

In Brazil, some 85 to 95% of the total KCl demand is in the form of granular grade material, mostly for bulk blending. In 2009 over 3.1 Mt of KCl imported into Brazil was in the granular form, out of 3.45 Mt in total. The remainder was in the standard form.

Consequently, SPSA will produce a predominantly granular product once operations are in steady state. The smaller amount of standard product that will be produced by SPSA can be sold into any of the potash markets.

The Brazilian market accepts Muriate of Potash (MoP) product with a grade of at least 60% K_2O, while lower grades will lead to price penalties and a poor reputation. Some allowance for product grade variations is therefore made in the process plant design, with a targeted K_2O grade of 60.5% in MoP.

17.5 Commodity Price Projections

Fertecon’s potash price forecast to 2020 is shown in Table 17.5.1. The CFR price reflects the price that buyers will pay for standard product at the port in Brazil.

The economic model uses a netback Tchiboula price, which is determined from the Brazilian CFR price, the granular product premium and shipping rates from the ROC to Brazil. Prices beyond 2020 are held fixed at the 2020 level. These netback prices are shown in Table 17.5.2.

Table 17.5.1: Average Annual Potash Price Projections – Standard KCl (Real US$ per t)

Year	Brazil CFR (US$/t KCl)
Actual	
2009	647
2010	390
2011	506
Forecast	
2012	525
2013	550
2014	570
2015	540
2016	510
2017	510
2018	530
2019	550
2020	590

Source (Fertecon, 2012) Potash Outlook May 2012

Table 17.5.2: Sintoukola Netback Potash Price Calculation (US$)

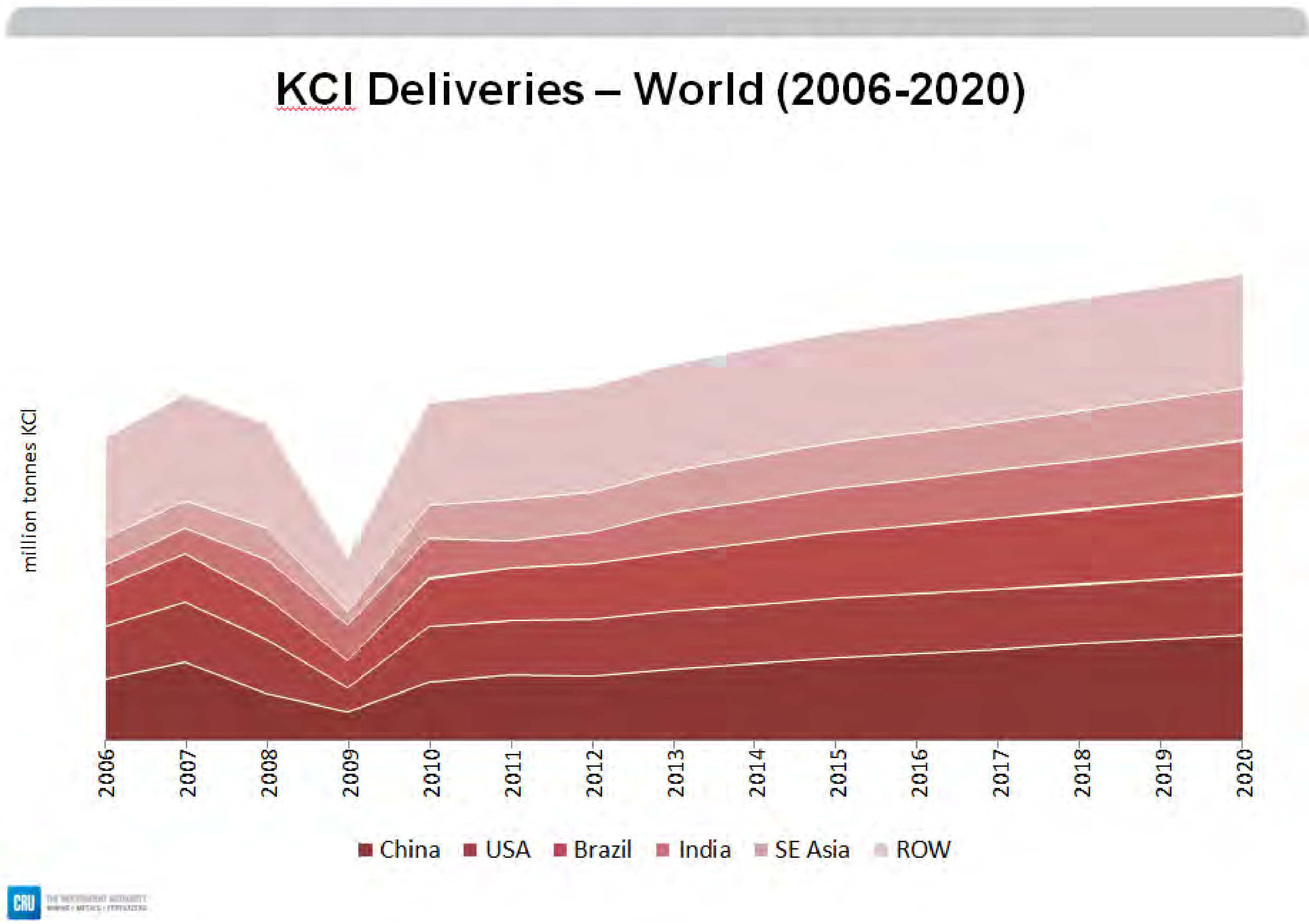
Item	Source	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021+
Brazil Standard CFR	Fertecon	525	550	570	540	510	510	530	550	590	590
Premium for Granular Product	CRU/ Fertecon	15	15	15	15	15	15	15	15	15	15
Freight Pointe Noire to Santos	CRU	26	23	22	22	23	24	24	24	25	25
Netback Price at Sintoukola Jetty		514	542	563	533	502	501	521	541	580	580

The Brazilian CFR price in Table 17.5.1 is based on standard product. CRU state that “as a general rule in recent history, the price of granular potash is generally quoted at a US$15 to 20 per t premium to standard grade in international trade. While this premium is very much a function of market preference versus supply availability, it generally is a reflection of the overall global supply and demand balance. In other words, the weaker the general market conditions are, the lower the premium that can be extracted from the market, and of course the opposite is true as well.” As a result, ELM added US$15/t to the netback price in Table 17.5.2 for all granular product sold.

17.6 Contracts and Status

No material contracts are in place at this time.

Two Memoranda of Understanding (MOU’s) are under discussion with possible infrastructure suppliers.



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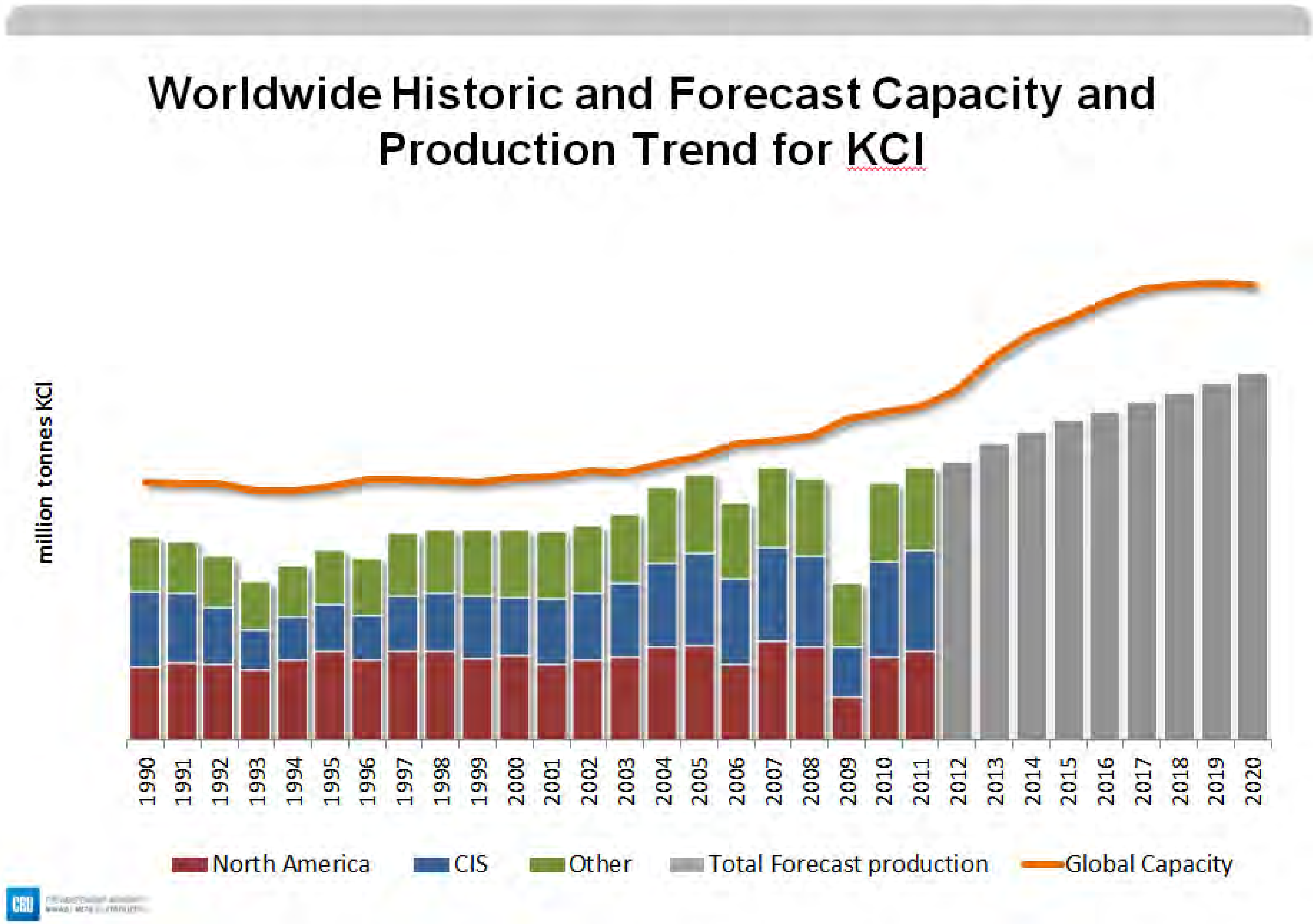
Republic of Congo

Source: ELM, 2012

Figure 17-1

Predicted World KCl Demand



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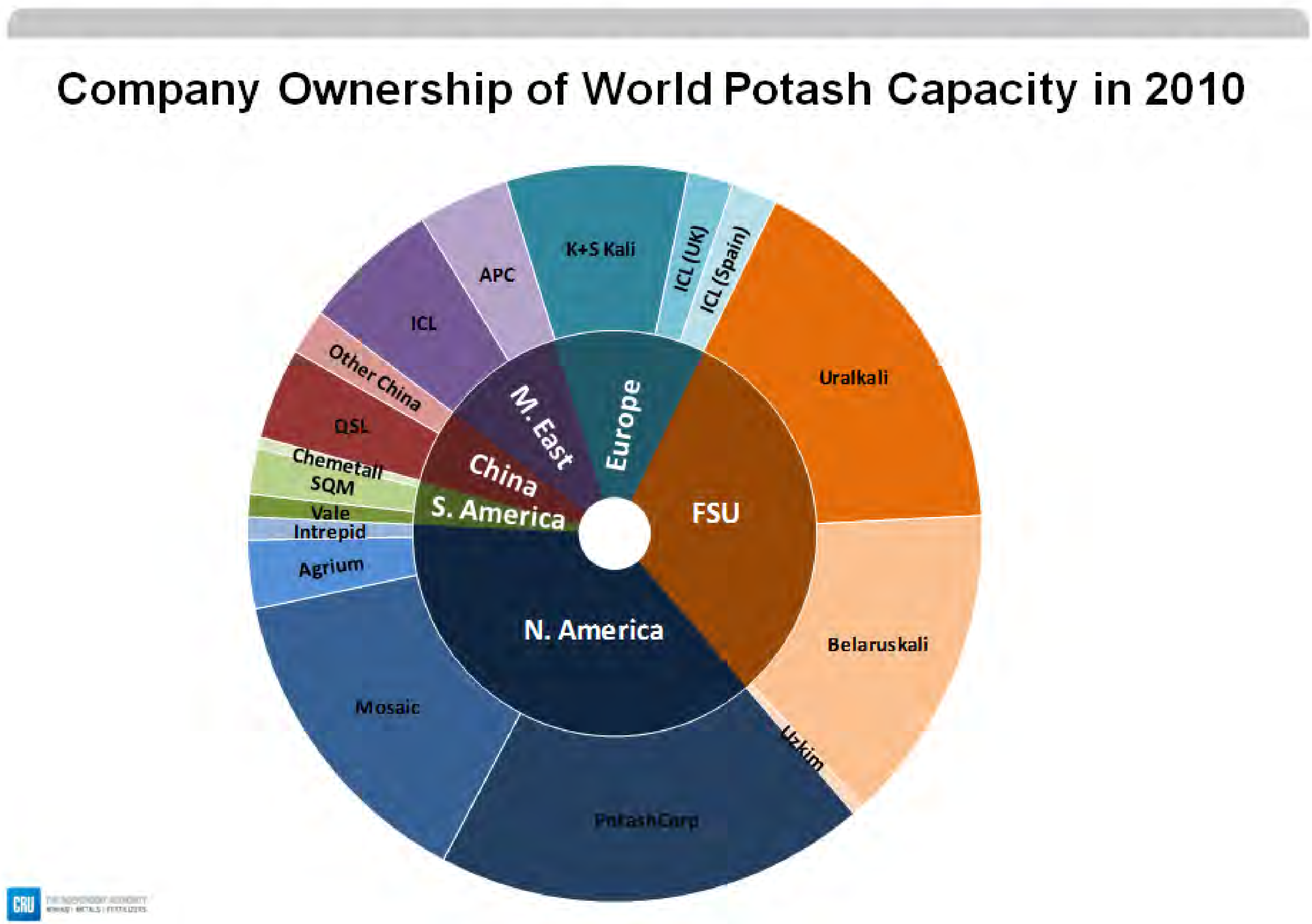
Sintoukola Potash Project
Republic of Congo

Source: ELM, 2012

Figure 17-2

Historic and Forecast Potash Supply


Company Ownership of World Potash Capacity in 2010
K+S Kali
APC
ICL
ICL (UK)
ICL (Spain)
Uralkali
Other China
QSL
Chemetall
SQM
Vale
Intrepid
Agrium
Mosaic
PotashCorp
Uzkim
Belaruskali
M. East
Europe
China
S. America
FSU
N. America

18 Environmental Studies, Permitting and Social or Community Impact (Item 20)

18.1 Introduction

This Section summarises the social and environmental studies and assessment of project risks undertaken as part of the SEIA in the period March 2011 to June 2012. As the Sintoukola Project is a major industrial development, the SEIA is required by Congolese law and is being undertaken in accordance with Congolese Decree 2009-415 (which defines the scope, content and procedures for an SEIA) and international standards.

Volume IX (SRK, 2012f) describes the SEIA studies and related analysis in more detail.

18.2 Environmental and Social Setting

18.2.1 Administrative Setting

The ROC is located in west-central Africa spanning the equator. It borders Gabon, Cameroon, the Central African Republic, the Democratic Republic of the Congo, and the Angolan exclave of Cabinda, with a short stretch of coast on the South Atlantic Ocean.

The exploration license covers an area of 1,436.5 km^2 along the northern part of the coastline in the west of the country (Figure 2-1). The main focus of exploration, the Kola deposit, is located in the eastern part of the exploration license and lies, approximately 90 km north of Pointe Noire.

The ROC is divided into twelve departments. Each department is subdivided into districts (sous-préfectures). The Sintoukola Project infrastructure will be located in two sous-préfectures (Madingo-Kayes and Hinda) in Kouilou Department.

Land tenure in the ROC is complex. In principle, land is sacred and cannot be sold. Landowning families called *terriens* (descendants of families who founded the village) maintain guardianship of the land. Access to most land in the Sintoukola Project area is through the acquisition of usufruct (the right to use the land without obtaining the legal title to the land itself), obtained by payment to the relevant *terrien*. Indigenous people are not required to pay for land use, and can settle in any village, on any land they wish, based on their status as the original inhabitants of the ROC. However, in practice indigenous families also make arrangements with the local *terrien* in terms of usufruct of the land.

Land tenure is further complicated by two historic upheavals in the approach to land ownership. In 1968 a socialist approach to ‘access to land’ was adopted and all land was appropriated by the government. This was reversed in 1991 when an act was passed to hand the land back to the *terriens*. Although subsequent plans called for a national commission to drive the process of developing cadastral maps for the *terriens’* land, the commission has not been assembled and cadastral mapping only started in mid-2012.

18.2.2 Physical Setting

The ROC is located in the tropics and the climate is characterised by high temperatures and humidity that remain relatively constant throughout the year. Seasonal variations in rainfall occur due to the movement of the tropical rain belt between the northern and southern tropics over the year. There is a distinct wet season (October to May) and dry season (June to September). Rainfall declines slightly in December and January and this period is referred to locally as the ‘small dry season’. The mean annual rainfall at Pointe Noire (the nearest location to the Sintoukola Project with long-term data) is 1,242 mm (based on records from 1931 to 2009).

The Sintoukola Project is located entirely within the Coastal Belt, a distinct geomorphological zone comprising an approximately 50 km wide strip of land between the Mayombé Mountains and the coast. The terrain of the Coastal Belt is low lying (less than 200 m amsl), with rolling hills, low plateaus and wide, swampy valley plains. There are three distinct terrain categories, all of which are relevant to the Sintoukola Project: coastal plains and dunes, coastal plateau, and alluvial plains and swamps.

Air quality in the area is generally good with low measured inhalable particulate concentrations attributed to high rainfall rates in the area (rainfall provides a natural form of dust mitigation by wetting surfaces and decreasing dust concentrations due to regular washout).

The Sintoukola Project area is bound by two major river systems flowing in wide alluvial plains close to the South Atlantic Ocean. These are the Noumbi (to the north-west) and the Niari-Kouilou (to the south-east). The Noumbi River drains in a south-westerly direction, through the CDNP (refer to Section 18.2.3) into the ocean some 100 km north of Pointe Noire. The Niari-Kouilou is the second largest drainage system in the ROC and drains through the northern extent of the Tchimpounga Nature Reserve (see Section 18.2.3) into the ocean some 40 km north of Pointe Noire.

The surface hydrology of the Sintoukola Project area is dominated by drainage belonging to the Noumbi River system; this includes the northern and central portion of the mine area, and the segment of the proposed transport corridor from the mine area to near the intersection with the N5. The south-eastern portion of the mine area hosts easterly draining rivers which form part of the Niari-Kouilou river system. The remainder of the Sintoukola Project area lies within smaller catchments that drain into the ocean. River flow rates within the Sintoukola Project area are fast, leading to rapid peaking and recession of river flows in response to rainfall.

Groundwater flow is interpreted to be away from the north-south topographic divide, to the north-west/west and to the south-east/east.

Surface waters across the Sintoukola Project area are moderately acidic with generally low metal concentrations, although copper and zinc are somewhat elevated along with additional elements in the coastal area. Given the current and past absence of industry, it appears that these reflect natural background concentrations for this area. Groundwater across the Sintoukola Project area is generally good quality, moderately acidic to neutral with low dissolved solids and low metal concentrations.

Soils in the Sintoukola Project area have a variety of textures, structures, depths and chemical composition, varying from moderately shallow and wet soils that are associated with a saturated zone at or close to surface, to deep and very deep sandy soils. Water holding properties, organic content and resistance to erosion are low for sandy soils, which may limit their ability to support some land uses.

18.2.3 Biological Setting

CDNP overlaps with the northern and central part of the Sintoukola permit area (Figure 2-1) and defines the biological setting for a significant proportion of the Sintoukola Project area. The CDNP was created by Presidential decree in 1999 and extends from the ocean to the Mayombé Mountains, covering 504,950 ha. It is the only protected area in the ROC that includes marine and coastal areas and is recognised by the United Nations Educational, Scientific and Cultural Organization (UNESCO) as a coastal national park. It is also included on the Congo's Tentative List for nomination as a UNESCO World Heritage Site and was designated a Ramsar wetland of international importance in 2007.

CDNP comprises a diverse complex of marine and continental wetland types (including littoral coast and beaches, mangroves, lagoons, lakes, permanent rivers and streams, papyrus marshes and submerged swamp forests). Fauna diversity is high (more than 1,200 species) with species of global conservation concern including the forest elephant, gorillas and marine turtles

The CDNP is managed by the Wildlife Conservation Society (WCS) and a conservator appointed by the Congolese Direction Générale de la Faune et des Aires Protégées. Three distinct zones are used to control activities within and around the CDNP:

- Integral protected zone – no new resource extraction, agriculture or logging are permitted in this zone ; and
- Eco-development zone – sustainable use of natural resources by local communities is allowed, along with logging, agriculture, mining and oil exploration or exploitation (subject to appropriate environmental and social studies and associated permits).

Additionally, a 5 km buffer zone is defined around the borders of the park. Mining and exploration are not permitted in this zone without the prior approval of the relevant authorities.

The Sintoukola Project mine and majority of the haul road and power line will be located in the eco-development zone, while short lengths of the haul road and power line will cross the buffer zone; the rest of the project facilities are located entirely outside the CDNP (Figure 2-1).

The habitat types in the vicinity of the mine site are predominately forests (of varied quality) with small areas of savanna and grassland. Generally, forests are in medium to good condition, but with degraded quality around villages and in the vicinity of accessible roads. Along the proposed haul road route the predominant habitat types are grassland/savannah mosaic, with patches of degraded forest. Large areas of eucalyptus plantations are located approximately 25 km from the mine site. Towards the coast, the habitat is dominated by gallery and littoral forests of high diversity due to the occurrence of varied habitat types in close proximity. There is an abundance of freshwater and brackish aquatic environments across the Sintoukola Project area.

Although some areas have been obviously transformed (e.g. eucalyptus plantations) and have low biodiversity value, natural habitat still occurs close to human populations (although some of these natural areas are currently being disturbed). Despite the presence of a number of human communities, fauna and flora diversity and species richness in the Sintoukola Project area are high. It appears that fauna have adapted their activities to avoid humans and that human density and activity patterns in the Sintoukola Project area have to date not reached a level that significantly limits wildlife movements, as has been observed in other parts of central Africa.

Marine habitats in the vicinity of the jetty site are relatively uniform. The dominant habitat type is consolidated sediment or fine to medium grained sand, with some areas of sub-tidal rocky reefs (that support attached invertebrate communities). Preliminary data indicate low dissolved oxygen levels, low water temperatures and low light levels, which may affect the biological productivity of the area.

The beaches within the vicinity of the proposed jetty host nesting sites for olive ridley sea turtles (*Lepidochelys olivacea,* listed as vulnerable by the IUCN) and leatherback turtles (*Dermochelys coriacea,* listed by the IUCN as critically endangered). Long-term data show a decreasing trend in nesting and numbers along the entire coastline of the ROC, which is attributed to impacts from legal and illegal fishing, natural and anthropogenic beach erosion and human activities along the coastal margins. Endangered and sensitive megafauna are also present offshore, including the rare Atlantic humpback dolphin (listed by IUCN as vulnerable) and other sensitive species (such as bottlenose dolphins and humpback whales). It is likely that the range of these megafauna includes the proposed offshore Sintoukola Project area.

The power line and gas pipeline from Pointe Noire to the Sintoukola Project area will run along the existing Route National 5 (RN 5 which is a principal national road) and roughly parallel to the southern boundary of the Tchimpounga Natural Reserve (TNR). The TNR is an area of approximately 52,000 ha of coastal plain savannah and galleried mosaic forest patches established by Presidential decree in 1999. The reserve is managed by the Jane Goodall Institute (JGI) and a Congolese conservator and is host to the largest chimpanzee sanctuary and rehabilitation centre in Africa.

18.2.4 Social Setting

Detailed social baseline studies have been undertaken across the Sintoukola Project area and have focused on fourteen villages and three blocks (sub-settlements within a village) in the immediate vicinity of the project-related infrastructure. The total population of these is approximately 6,000 people and the majority of villages have less than 500 inhabitants. Local communities are comprised predominately of people of Vili ethnicity, with smaller populations of Teke, Lari, Loumbou, Zairois, Yombe Bembebe and Babongo.

With respect to international law and standards, the Babongo population in the ROC is classified as indigenous as it comprises a non-dominant group with distinct language, culture and beliefs that may experience exclusion and discrimination by mainstream society. Congolese Law No. 5 – 2011 has recently been enacted to promote and protect the rights of indigenous populations. In the Sintoukola Project area the level of integration of the Babongo with other ethnic groups is relatively high, however, exclusion and discrimination issues do exist and they remain a vulnerable group.

The Sintoukola Project area has a rich cultural heritage dating back at least 500 to 600 years and comprising ancestral village sites, archaeological artifacts, cemeteries, sacred sites and socio-religious institutions (the last being an intangible form of cultural heritage). Many of these are significant in terms of day to day activities and specific rituals and dominant beliefs in the Sintoukola Project area.

Each community is dependent on its surrounding natural resources, with hunting, fishing and gathering in nearby areas supplementing subsistence agriculture (despite the abundance of grass and other fodder there is no significant animal husbandry and no extensive grazing within the

Sintoukola Project area). The growing season starts around October and continues until August with only a short period where plant growth ceases in the dry season.

The contribution of food produce sales to local livelihoods is restricted by poor road infrastructure and the high relative cost of travel between communities and potential markets in Madingo-Kayes and Pointe Noire. Due to low economic activity in the Sintoukola Project area, poverty is widespread in the local communities and accommodation and living conditions are basic at best.

The provision of public services such as education, sanitation, water supply and healthcare is limited; facilities are either absent, of poor quality, under-resourced or remote from the village. Where facilities do exist, uptake by local people may be limited due to service costs, travel-related access issues and, in the case of schooling, competing demands from domestic and subsistence activities. As a consequence, few people in the local population have even basic educational qualifications or access to acceptable sanitation facilities, clean water or formal healthcare.

The lack of proper sanitation facilities and clean water coupled with poverty, rudimentary local healthcare and basic living conditions aggravate exposure to, and consequences of, disease. Malaria is a major public health challenge in the Sintoukola Project area and is regarded as one of the leading contributors to the local burden of disease. Communicable and waterborne diseases add significantly to the burden. The prevalence of HIV is unknown in the Sintoukola Project area, but based on recent national surveys this may also be a determinant of public health.

Most communities in the Sintoukola Project area can be classified broadly as 'vulnerable' as poverty is widespread and there is only limited resilience to any adverse impacts on their limited livelihoods. The vulnerability of certain groups may be heightened where they face specific and additional constraints; examples include women and girls (gender disparity in education and limited access to income-generating opportunities), female headed households, the Babongo (exclusion and discrimination), the elderly, disabled people and those without access to land.

18.3 Environmental Legislation and Permitting and Lender Requirements

18.3.1 Environmental Legislation and Permitting

Congolese environmental legislation establishes requirements for the effective environmental and social management of the Sintoukola Project throughout its lifecycle, Key legislation includes[1]:

- Law on the Protection of the Environment (Law 03-1991);
- Mining Code (Law 04-2005);
- Water Code (Law 13-2003);
- Forestry Code (Law 16-2000, as amended by Law 14-2009);
- Decree for Creation of the Conkouati-Douli National Park (CDNP) (Decree 99-136); and
- Law on the expropriation of land (Law 11-2004).

This legislation also defines the environmental permits (including authorisations and approvals) that the Sintoukola Project must obtain; the principal permits are outlined below. Environmental permits

1 Supplementary information on environmental legislation and permits is provided in Volume IX.

sit within a broader framework of project permits. Procedural aspects of obtaining environmental and other permits will be further defined and refined in consultation with Congolese authorities during the FS.

Environmental authorisation

Environmental authorisation will be obtained for the Sintoukola Project by submission and approval of the SEIA according to requirements set out in the Law on the Protection of the Environment, Decree 2009-415 and Articles 50, 57 to 59 and 101 of the Mining Code. Environmental authorisation is required as part of the application for a mining permit

Land acquisition agreements

In the ROC the land acquisition, resettlement and compensation process is predominantly executed by the government on behalf of the Sintoukola Project proponent. Once the required land has been identified, the Ministry of Land submits a formal request to the Cadastral Office in Brazzaville to demarcate the Sintoukola Project land take and for its surveyors to determine the nature and the occupancy of such land; this includes the definition of customary rights. This work is currently underway. Once this has been completed, the Ministère des Affaires Foncières et du Domaine Public discusses and agrees resettlement and compensation measures with landowners and local authorities and issues land permits. Staff from the Ministère des Affaires Foncières and SPSA together form the land commission, which carries out the necessary land acquisition and compensation process and which is responsible for overall budgets, performance and schedule of the resettlement work.

Authorisations for Emissions, Discharges and Waste Disposal

It is currently understood that authorisations for emissions, discharges and waste disposal should be sought as part of the SEIA approval process. Emissions to air, the oceanic discharge of waste brine, land-based disposal of solid waste from processing and other emissions, effluents and wastes will be managed through a comprehensive Social and Environmental Management Plan (SEMP). The SEMP will be submitted as part of the SEIA report.

Authorisation for Forest Clearance

Deforestation of forest or forested land is subject to prior authorisation under the Forestry Code. Such activity is permissible in the eco-development zone of the CDNP, subject to an independent SEIA (as is currently being undertaken).

Agreement with the Authority Managing CDNP

The Decree that created CDNP requires the Company, as holder of a valid exploration permit that overlaps with the CDNP, to sign an agreement on cooperation with the CDNP's managing authority (WCS in partnership with a conservator representing the Congolese government). The agreement must be discussed with local communities before signing and will include details of applicable conditions for operating in the eco-development zone.

18.3.2 Lender Requirements

In addition to meeting Congolese legal requirements, the SEIA process has been planned and implemented to ensure that it meets the high standards expected by many of the largest potential lenders and in particular the Equator Principles Financial Institutions (EPFI's). EPFI's require that the International Finance Corporation Performance Standard (IFC PS) and World Bank Group's Environmental, Health, and Safety (EHS) Guidelines are followed.

The SEIA will enable ELM to identify how it can meet the environmental and social requirements of EPFIs and undertake the Sintoukola Project according to international standards.

18.4 Environmental and Social Studies and Management

ELM is complying with Congolese legislation and the Equator Principles, in the planning of the Sintoukola Project. Social and environmental management elements that are in place are outlined in Sections 18.4.1 to 18.4.3.

18.4.1 Corporate Policy

Social and environmental management has been undertaken in accordance with international good practice in order to meet ELM's corporate objective and environmental management policy to "*operate at all times with a high social, environmental and ethical awareness*". ELM has committed to developing a project-specific social and environmental policy during the FS.

18.4.2 Status of the SEIA

As noted in Section 18.3, the SEIA is being undertaken in accordance with Congolese legal requirements, the IFC and EHS (which represent *de facto* international standards). Volume IX (SRK, 2012f) contains detailed information on the SEIA process and the principal phases followed from initiation to submission of the SEIA report to the Congolese authorities. A summary of key aspects is presented below.

The SEIA scoping phase was undertaken between June and December 2011. This consisted of a social and environmental scan of the Sintoukola Project area, reconnaissance surveys for key specialist disciplines and scoping consultations with stakeholders. Issues raised during these consultations (summarised in Volume IX) (SRK, 2012f) were central to the development of appropriate Terms of Reference (ToR) for subsequent stages of the SEIA.

The key outcomes of the scoping phase were a summary of available environmental and social information and preliminary impact identification for input to early Sintoukola Project design by ELM (for example environmental and social inputs to trade-off studies considering alternative infrastructure options). The scoping phase culminated with the development of draft ToR for the SEIA. Submission of the draft ToR to the Ministry of Sustainable Development, Forestry and the Environment (MSDFE) in February 2012 signified the formal initiation of the environmental authorisation process for the Sintoukola Project. The ToR for the SEIA have been approved by the MSDFE.

Specialist baseline studies have been initiated covering the following disciplines:

- Social;
- Indigenous peoples;
- Terrestrial biodiversity;
- Marine biodiversity and physical environment;
- Natural resource use;
- Cultural heritage;

- Health;
- Soils and land capability;
- Hydrology and hydrogeology;
- Geochemistry;
- Geology;
- Noise;
- Air quality;
- Traffic;
- Visual; and
- Macroeconomics.

Baseline studies are typically the most time consuming phase within the SEIA process, and therefore can have implications for the project timeline. An aggressive approach has been taken to scheduling the baseline studies in order to optimise data collection particularly for studies requiring long-lead times or seasonal data. With the exception of traffic and marine investigations, baseline studies were completed by mid-September 2012, with baseline reporting scheduled to be completed by the end of September 2012. Traffic studies and reporting will take place in September and October 2012.

Following baseline reporting, a detailed assessment of the identified impacts in the Sintoukola Project's area of influence will be undertaken. Impact evaluation can only start once the Sintoukola Project is sufficiently defined for impacts to be accurately evaluated. For some impacts, the evaluation process will involve the use of predictive modelling to determine impact significance (such as air quality modelling, noise modelling, visual modelling, contaminant fate and transport modelling).

Based on the impact assessment, specific management measures will be defined in the SEIA for each impact. The significance of impacts will be evaluated, without and with the management measures, to help distinguish impacts that should influence the various decisions pertaining to Sintoukola Project approval – including ELM's decision on the feasibility of the Sintoukola Project and regulatory authorities' decisions on the granting of environmental approvals – and to prioritise the implementation of management actions by ELM. At the current stage in the SEIA process (pre-impact assessment) it appears that the majority of potential environmental impacts identified during the scoping phase can be readily managed through the implementation of standard environmental management plans. However, there are several potential impacts, including some related to social and community issues, which will warrant specific management measures, which may have significant implications for the Sintoukola Project. These are discussed further in Section 18.5.

Two SEIA reports will be prepared to satisfy Congolese regulatory and international standards. One will be prepared and submitted for in-country permitting purposes in December 2012, with content and structure in accordance with Congolese guidance; regulatory authorities note that this report should not refer to international standards. A separate SEIA report that will be explicit about observance of and compliance with international standards will be issued in February 2013.

18.4.3 Status of Existing Environmental and Social Management

As required under Congolese legislation (Section 18.3.1), an SEIA was prepared by Environment Plus (a Congolese environmental consultancy) for Phase 1 exploration activities and associated works (such as clearing paths, cutting forest lines, seismic survey, preparing drilling platforms etc.). An addendum to the Phase I SEIA has been prepared to cover the Phase 2 exploration activities.

The Phase 1 exploration SEIA includes a framework environmental management and health and safety management system. The principles of these systems are in accordance with good practice; however this has not been fully transposed into a formal management system. ELM is aware of the need to establish a formal social and environmental management system (SEMS); development is being undertaken as allowed by the limited resources available during the exploration phase. The framework for social and environmental management is being updated as part of the SEIA process (Section 18.6).

Many of the mitigation and management measures outlined in the SEIA for Phase 1 exploration have been implemented at the exploration camp and drill sites. However, a formal audit has not yet been undertaken. The addendum to this SEIA requires that the Phase I mitigation and management measures are applied to Phase 2 exploration activities.

Several environmental guidance notes have been issued by ELM to cover engineering studies as part of the PFS. Activity-specific environmental management measures are integrated with the scope of work for these studies and an environmental specialist is present during work in potentially sensitive locations.

ELM and SPSA has prepared a number of codes of conduct and policies relevant to activities undertaken during the PFS (and beyond):

- <u>Code of conduct for contractors and expatriate employees</u>: addressing social and community interactions and establishing social and environmental standards of behaviour/conduct;
- <u>Community development policy:</u> establishing the principle of supporting the social and economic advancement and capacity building of communities whose lives are affected by Sintoukola Project activities while respecting the communities' own vision of development;
- <u>Community health and safety policy:</u> establishing the principle of avoiding or minimising project-related risks and impacts on community health, safety and security;
- <u>Human resources policy:</u> recognising that the maintenance of ELM's reputation and economic growth through employment creation and income generation requires a stable workforce, a humane and safe workplace and the protection of the fundamental rights of workers;
- <u>Recruitment policy:</u> establishing the principle that recruitment should attract the best qualified candidate for the job, and to do so in a manner that is fair, transparent and equitable, providing for the inclusion of disadvantaged and marginalised groups, as well as those with low-level or no skills;
- <u>Security personnel policy:</u> recognising the need to ensure that the safeguarding of personnel and property is carried out in a legitimate manner that avoids or minimises risks to the community's safety and security, avoids human rights abuses and avoids risks to company/community relations posed by the use of security personnel;
- <u>Stakeholder engagement policy:</u> establishing the principle that stakeholder engagement is the basis for building strong, constructive, and responsive relationships that are essential for the successful management of the company's environmental and social impacts; and

- Draft environmental management policy: establishing the principle of operating at all times with a high social, environmental and ethical awareness.

18.4.4 Stakeholder Engagement

Stakeholder engagement is a broad, inclusive, and continuous process of relationship building and maintenance between a company and its stakeholders, which includes a range of activities and spans the entire project lifecycle. The SEIA stakeholder engagement is one form of stakeholder engagement required for the Sintoukola Project and is being undertaken according to Congolese legal requirements and international standards. Besides the SEIA stakeholder engagement activities, ELM has additional stakeholder engagement responsibilities throughout the life of the Sintoukola Project (these are described in the Stakeholder Engagement Plan (Appendix B of Volume IX, SRK, 2012f)).

The main stakeholder groups are the same for both SEIA and broader Sintoukola Project engagement:

- Central government, in particular the various ministries responsible for environmental protection and the regulation of industrial development;
- Local government authorities;
- Local communities and indigenous people;
- Conservation NGOs active in the Sintoukola Project area (WCS, Renatura and JGI); and
- Local churches and religious/cultural groups.

ELM has developed positive relationships with local and central government authorities and the three conservation NGOs (although this does not entirely eliminate potential material risks to the project – see Section 18.5). However, ELM has faced a number of challenges in dealing with local communities and indigenous people in the Sintoukola Project area, particularly in the early stages of Sintoukola Project exploration:

- Grievance mechanism: implementation of a formal system (grievance mechanism) to record, address and respond to issues raised by stakeholders (as required by IFC PS 1) has been delayed;
- Managing the flow of project information: inconsistent or incomplete information from the company and inaccurate information from other sources has eroded community trust;
- Managing community expectations: local communities have developed high expectations of benefits from the Sintoukola Project, particularly with respect to employment and community development, and do not understand that the Sintoukola Project may not proceed. The company needs to address this;
- Addressing the marginalisation of indigenous people: ELM needs to ensure that the Sintoukola Project does not result in increased discrimination or marginalisation of the Babongo (see Section 18.2.4) and that they have equal access to benefits arising from the Sintoukola Project; and
- Addressing culturally sensitive sites: while ELM's preferred approach is to avoid the disruption of sacred and other culturally sensitive sites, some sites will only be identified once construction is underway. In these cases, where avoidance is not possible, the relocation of physical artifacts or the 'transfer' of the spiritual / sacred value will need to be

completed in consultation with affected communities. At present there is no precedent for such relocation and transfer; an acceptable approach will need to be developed.

In response to these challenges, ELM has:

- Employed an experienced Social Manager and Head of Community Affairs and a supporting Community Liaison Officer. Both are permanent contracts based at the current exploration site, in close proximity to the potentially affected communities;
- Implemented a grievance mechanism to ensure that the company can identify and quickly respond to community concerns and issues;
- Had meetings with the affected communities explaining the different stages of the Sintoukola Project; the uncertainty of employment during the exploration stage;
- Established groups of village representatives as part of the SPSA stakeholder engagement and communication plan;
- Drafted a protocol for visiting consultants' interaction with the communities;
- Implemented a communication plan to ensure that consistent and accurate information is regularly discussed with local communities (and other stakeholders). This includes a gradual realignment of community expectations as the company's socioeconomic strategy is developed;
- Ensured that specific engagement with the Babongo has been conducted as part of the SEIA and Sintoukola Project stakeholder engagement activities; specific social baseline data have also been collected in collaboration with local Babongo families; and
- Commenced the development of a process for dealing with culturally sensitive sites with a cultural heritage expert.

18.5 Potential Material Impacts / Risks

Through the SEIA process, a number of risks have been identified that will need to be considered and managed as the Sintoukola Project is further developed. A risk is a potential event that may affect the planning and management of the Sintoukola Project. A risk that may detrimentally affect Sintoukola Project planning and management is a *negative risk*; a risk that may enhance Sintoukola Project planning and management is an *opportunity*.

The significance of each risk is defined by considering the probability that the event will happen and the severity of the consequence if it happens, taking into account planned preventative and mitigation measures. Significant negative risks may have *material* consequences for the company and/or project.

For the purposes of this document, the focus is on material negative risks. These are risks that:

- Are significant and may be impossible or difficult for the company to effectively prevent or manage (including cases where mitigation and management measures have a high probability of failing or the company has no control over the event(s) causing the risk);
- May cause significant delays to the Sintoukola Project schedule; and
- May increase costs or financial liabilities.

What follows is a qualitative assessment of the cause and implications of potential material risks. While the material risks noted below are considered likely to be relevant to the Sintoukola Project, their significance has not been quantified. The implementation of effective preventative and mitigation measures by SRK or ELM (also summarised below) may reduce or eliminate some of these material risks.

18.5.1 Risk of Project Delay Due to Delays in Obtaining Environmental Permits

Delays in the issue of environmental permits (Section 18.3), which are an integral part of the mining permit application, may impact the Sintoukola Project schedule. The Sintoukola Project cannot proceed to the construction phase without the mining permit. Delays to key construction activities may have significant commercial implications if the Sintoukola Project's proposed ramp up schedule to full production is not achieved.

Delays in the issue of environmental permits may result from:

- Delayed submission of the SEIA report: late changes to the Sintoukola Project design may necessitate significant revisions to the SEIA prior to submission to the Congolese authorities;
- Delayed review of the SEIA report: the expected period for review (as per Congolese legislation) may be extended because Congolese regulatory authorities require additional time due to insufficient capacity;
- Delayed approval of the SEIA report: the Congolese authorities may require substantial revisions to the SEIA report and further field work; and
- Irregularities in the review and approval process: individuals may seek to operate as 'gatekeepers', linking progress of the SEIA through the review and approval process to political or personal gain.

The risk associated with delays in SEIA submission has been managed by the integration of environmental and social considerations with Sintoukola Project design activities from the outset. In this context, the risk of late changes to the Sintoukola Project design causing delays to submission of the SEIA report is considered to be small.

The challenge of managing risks associated with delays in the review and approval process once the SEIA report has been submitted is being met through:

- Compliance with reporting requirements: the structure, scope and content of documents submitted to the government will be as agreed with the regulators in the approved SEIA ToR (see Section 18.4.2);
- Recognition of capacity issues: reports will be accompanied by concise executive and non-technical summaries to assist understanding by non-specialist audiences; presentations and discussions will be used to 'walk through' the reports with the regulators;
- Elimination of study gaps: the SEIA study is geographically and scientifically comprehensive, gathering detailed seasonal data from all the areas potentially affected by the Sintoukola Project. This reduces the risk that significant gaps requiring further work will be identified by regulators during the review process; and
- Zero tolerance for bribes: both ELM and SRK have a zero tolerance policy with respect to bribes.

18.5.2 Risk of Project Delay Due to Inadequate Management of Impacts on High Value Biodiversity

A significant proportion of the Sintoukola Project is located within CDNP, an internationally recognised protected area that is co-managed by the Wildlife Conservation Society (WCS), a high profile international conservation non governmental organization (NGO). Initial baseline study data indicate that significant biodiversity remains in the eco-development zone despite the widespread use of natural resources by human communities and larger-scale logging activity (see Section 18.2.3). Sintoukola Project facilities at the coast, while outside the CDNP, are near or cut across a beach that is a known nesting site for critically endangered and vulnerable sea turtles of international conservation importance. A local conservation NGO (Renatura) has been running a monitoring and protection programme at this and other beaches in the area for more than 10 years. WCS also has an offshore monitoring programme that tracks the migration of rare dolphins and other marine mammals of conservation concern along the Gabon and Congo coast, including the area that will host the Sintoukola Project's transhipment activities.

In this sensitive environment, construction and operation of the Sintoukola Project may directly and indirectly impact terrestrial or marine habitats and species of conservation importance and Sintoukola Project impacts will be closely scrutinised by Congolese regulators and conservation NGOs. Failure to adequately manage biodiversity impacts may result in Sintoukola Project delays from:

- Delayed approval of the SEIA report: Congolese regulators may consider the assessment of biodiversity impacts and development of related management measures to be insufficient in the context of CDNP and the sensitive species present at the coast and offshore. Further baseline studies and impact assessment may be required, delaying approval of the SEIA report, which in turn will delay the environmental permit and mining permit (see Section 18.5.1); and
- Resistance from conservation NGOs and other stakeholders: WCS and Renatura may withdraw support for, and actively resist, the Sintoukola Project if they consider that the residual biodiversity impacts following implementation of the management measures will be unacceptable. Active NGO resistance may occur even if the Sintoukola Project has been approved by the Congolese regulator and the environmental permit issued. Resistance may take the form of legal challenges and the rapid mobilisation of an international campaign against the Sintoukola Project involving (for example) other conservation NGOs and anti-mining activists. Such a campaign may have implications for Sintoukola Project financing and apply sufficient pressure that the regulator reviews and reconsiders its initial approval.

Measures are being implemented to manage these risks through:

- Implementation of the mitigation hierarchy: the mitigation hierarchy has been embedded in the Sintoukola Project design process from the outset; with impact avoidance being preferred over impact reduction or minimisation of the risk of impacts being considered unacceptable by the regulator or NGOs is reduced. The concept of biodiversity offsets will be considered as necessary (as required by IFC PS6);
- Comprehensive biodiversity studies: terrestrial and marine biodiversity studies are being undertaken to international standards, collecting detailed wet and dry season data. The

comprehensive data gathered will underpin the impact assessment process and support the identification of appropriate mitigation measures; and

- Engagement with NGOs: planning for, and implementation of biodiversity studies involves both WCS and Renatura, who will undertake marine mammal and turtle baseline studies respectively. This approach has the benefit of strengthening the perceived legitimacy of the work, demonstrates transparency, provides local knowledge and encourages long-term cooperative relationships built on mutual trust.

18.5.3 Risk of Project Delay from Government-led Land Acquisition, Resettlement and Compensation Process

The land acquisition process must be completed before construction of the Sintoukola Project can begin. Some resettlement will be required, although the location and extent has not yet been determined and economic displacement may also occur. Land acquisition, resettlement and the associated compensation process are led by the government. A Land Commission has been set up by the Ministry of Land to oversee the process of land acquisition for the Sintoukola Project. ELM, through its subsidiary SPSA, is a member of this Land Commission. Its role on the Commission is only to provide the necessary information, funding, logistics and technical support to allow the process to proceed (for example, identification of the footprint of specific project activities). Its role in setting standards to be achieved or the pace of the process is relatively limited, which may introduce a number of significant risks resulting in project delay:

- Constraints on meeting international standards: both Congolese and international standards (IFC PS 5) require the compensation of affected landowners and people. However, international standards also require the restoration of livelihoods, which is not a requirement under Congolese law. While there is strong government support for completion of the process to national standards, there appears to be some resistance to meeting international standards by extending the process to accommodate livelihoods restoration. Extended negotiations with the government may be necessary in order to understand and address the government's concern and ensure that livelihoods replacement is adequately addressed and that the overall process meets international standards. Failure to meet international standards will prevent access to Sintoukola Project funding from EPFI's; and
- Failure to allow adequate time for land acquisition and resettlement: land acquisition and the planning and execution of compensation and resettlement are processes that require a detailed census and extended stakeholder consultation and engagement. Land tenure is complex (see Section 18.2.1) and there is currently no reliable information on land ownership within the areas where Sintoukola Project infrastructure will be located. To address this, a government-led cadastral survey commenced in June 2012, in parallel with initiation of the land acquisition process. There is a significant risk that delays to the cadastral survey or the land acquisition, compensation and resettlement process will delay initiation of the construction phase (in May 2013).

ELM is implementing various approaches to mitigate the risks associated with the land acquisition, resettlement and compensation process such as:

- Promotion of international standards: as ELM is a member of the Land Commission it has the opportunity to discuss and promote compliance with international standards during the regular commission meetings; and

- Supporting the cadastral survey: ELM is supporting the cadastral survey efforts by government staff through the supply of cars, GPS and computers; this logistical support is expected to facilitate more efficient acquisition and logging of cadastral data.

18.5.4 Risks Arising from Inadequate Engagement with Local Communities

ELM is now implementing a comprehensive engagement programme with local communities and indigenous people that meets the requirements of both Congolese law and international standards. As well as managing engagement going forward, this programme must also retrospectively address issues that have accumulated during the early stages of the Sintoukola Project (see Section 18.4.4), including initial inadequate responses to concerns raised by local communities and management of community expectations. Considerable time and effort is required to rebuild trust and constructive relationships with stakeholders. Failure to achieve this could result in protests and violent acts that can interrupt work and tarnish the company name on international markets.

18.5.5 Risk of Project Delay and/or Loss of Social License to Operate as a Result of Inadequate Influx Management

A common impact of major mining and infrastructural projects in developing countries is the influx of opportunity seekers. Influx is not negative by definition, but its impacts can be damaging where the migrants are not readily assimilated, placing stress on services, disrupting existing communities, and in some cases living in unhealthy and crowded conditions (already an issue for local communities).

Effective management of population influx to the Sintoukola Project area will require a partnership between the company and local and central government authorities; unilateral activities by the company are unlikely to be sustainable or effective. This reliance increases the risk that influx management will be delayed or inadequate in terms of scope and available resources. Failure to adequately manage influx may have the following consequences:

- Delays to the cadastral survey and land acquisition process: the influx of significant numbers of migrants during the cadastral survey and land acquisition, resettlement and compensation may slow progress and extend the period required to differentiate long-term residents and new migrants;
- Employment related tensions in local communities: the Sintoukola Project will require a significant number of skilled workers that cannot be sourced from local communities during both the construction and operational phases. Local communities have high expectations regarding employment that are unlikely to be met by the limited number of unskilled and semi-skilled positions that can be filled by local people; and
- Impacts on community infrastructure and natural resources: measures being put in place to mitigate significant potential impacts of the Sintoukola Project on community services, infrastructure and natural resources can be undermined by poorly managed influx, which can place additional strain on otherwise well-managed community services, infrastructure and natural resources. As the influx is driven by the presence of the Sintoukola Project, local communities and other stakeholders are likely to consider the company responsible, despite the role played by local and central government.

A number of approaches may be required to mitigate the risks associated with influx management:

- Preventing speculative land rights claims: the government will issue a 'décret en défense', which prevents migrants from claiming new land rights from the date of the decree, effectively 'freezing' the number of people that will be considered in the cadastral survey and land acquisition, resettlement and compensation process and preventing major disruption and delays from ongoing influx of migrants (although migrants may still come and squat in search of work); and
- Promotion of local training: ELM is drafting a training strategy to assess ways to increase the number of skilled workers that can be drawn from local communities, thereby reducing the number of workers drawn from outside areas.

At present the company has yet to explore a partnership approach on influx management with local and central government authorities; until an effective partnership has been established with a credible work programme and supporting resources, influx will remain a significant potential source of Sintoukola Project delays and risk to the social license to operate.

18.5.6 Risk to Company Reputation and Social License to Operate Associated with Road Safety

The Sintoukola Project will require a newly constructed private haul road and a service road (an upgrade of the existing RN 5 between the mine site and Madingo-Kayes) that will be shared with public users.

The section of the RN 5 that will be upgraded passes through many of the communities in the Sintoukola Project area. Due to very poor road conditions, the current volume of traffic using the RN 5 between the mine site and Madingo-Kayes is low and vehicles typically travel at relatively low speeds. Upgrading of this road will result in increased volumes of both company and public vehicles travelling at significantly higher speeds. This will increase the risk of vehicle collisions and pedestrian injuries and fatalities. Incidents involving company vehicles, particularly those resulting in community fatalities will damage the company's relationship with the affected community.

The haul road will use a blend of fencing and embankments to restrict vehicular and pedestrian access to the haul road, which, with the exception of a 4 km stretch at the coast, is not designed for public use or light vehicles. However, there remains a significant risk that local people will seek to cross or walk along the haul road, placing them in close proximity to 145 tonne road trains, with a significant risk of injury or fatalities occurring. Responsibility for incidents may still be assigned to the company by local communities even though the haul road is clearly not for public use.

The haul and service roads are essential components of the Sintoukola Project. In this context the company is designing preventative and mitigation measures to address the risks to social license and reputation due to road safety issues:

- Use of road signage and speed control measures within community boundaries;
- Road safety programmes with local communities to improve vehicular and pedestrian road safety and deter entry to the haul road;
- Training of ELM drivers; and
- Construction of vehicle and pedestrian underpasses for the haul road in consultation with affected communities to reduce the risk of entry to the haul road.

18.6 Management Plans

ELM is observing Congolese legislation and the Equator Principles in its planning of the Sintoukola Project. A framework for the social and environmental management system (SEMS) has been proposed with consideration of the requirements of IFC PS 1 and Congolese legislation. It also takes account of the main principles in the International Standards Organisation (ISO) 14001 Standard (ISO 14001:2004, 2004). At this stage no formal SEMSs have been implemented for the Sintoukola Project.

The SEMS will be organised according to the “plan-do-check-act” business performance improvement cycle. More detail on each element of the SEMS will be given as part of the SEIA; this will include the first version of the social and environmental management programme (SEMP) for the life of the Sintoukola Project, including project execution.

The SEMP will include an action plan that addresses the identified impacts of the Sintoukola Project. The SEMP will also contain supporting documents that guide the execution of actions in the action plan.

The supporting documents in the SEMP will include a series of detailed management plans dealing with the main impacts of the Sintoukola Project and could also include method statements or schedules providing details for the execution of specific actions. The series of management plans can be expected to cover at least the following subjects:

- Construction management;
- Closure and rehabilitation;
- Emergency preparedness and response and community emergency response;
- Waste management;
- Water management, including erosion control;
- Brine and insoluble disposal;
- Emissions (including noise) and dust control;
- Wildlife management;
- Community health and safety;
- Shipping and jetty management;
- Operational stakeholder engagement plan;
- Influx management plan;
- Resettlement action plan;
- Recruitment;
- Community development; and
- Training and skills development

The SEMS and SEMP will be dynamic documents, which will be continually revised and updated through the life of the Sintoukola Project.

18.7 Mine Closure

The following section briefly describes the requirements, objectives, considerations and proposed actions for the closure phase of the Sintoukola Project. Further information on these aspects is contained within the conceptual closure plan which is provided in Volume IX. This plan will be updated upon completion of the SEIA during the FS phase, and regularly throughout the life of the Sintoukola Project.

18.7.1 Legal Requirements

According to Article 50 of the Congolese Mining Code (Law 4-2005), a site rehabilitation plan must be submitted as part of a wider package of documents to the Ministry of Mining to obtain a mining permit. A soil rehabilitation or management plan is also required upon expiration of the mining permit (Article 63). The Mining Code does not specify further requirements on the detail of this plan and there is no regulatory requirement contained in the Mining Code (Law 4-2005) to post a financial guarantee for closure during the life of the operations. In addition to the standard legislation, SPSA has also signed a convention with the ROC Government which establishes specific requirements for the Sintoukola Project. Article 9 of this convention stipulates that at the end of mining activities, any material or equipment that can no longer be the subject to re-export, will be yielded to the Geology Administration.

As the Sintoukola Project is still in the planning phase and the SEIA process is not yet complete, no closure-specific commitments have been made to regulatory authorities or other stakeholders to date.

18.7.2 Other Requirements

During the life of the Sintoukola Project, other obligations or commitments relating to closure may arise from:

- Corporate requirements;
- International standards;
- Commitments to stakeholders; and
- Recommended management measures identified during the SEIA process and approved by regulatory authorities.

This plan has been prepared in accordance with international good practice in order to meet ELM's objective to "operate at all times with a high social, environmental and ethical awareness".

18.7.3 Closure Objectives

At the time of final closure of the Sintoukola Project, the mine areas should be reclaimed to a safe and environmentally sound condition consistent with closure commitments developed during the life of the Sintoukola Project. Specific closure commitments may be tied to the final land use for the Sintoukola Project area which should be determined in collaboration with local communities and other project stakeholders, including conservation NGOs active in the area. Given the location of the mine site and part of the haul road within the eco-development zone of the CNDP specific closure objectives related to biodiversity conservation and management and ecosystem services and functions may also be developed.

Until such time as specific closure objectives are defined, general objectives will be to:

- Maintain worker health and safety throughout closure activities, including concurrent closure;
- Protect public health and safety;
- Demonstrate chemical stability compatible with site conditions;

- Demonstrate physical stability compatible with site conditions;
- Create self-sustaining ecosystem compatible with site conditions;
- Minimise need for reclamation maintenance;
- Maintain community relations; and
- Reduce closure liability during operations through an aggressive concurrent closure program.

18.7.4 Factors Affecting Closure

In addition to the legal requirements and other obligations, the following environmental and social factors were considered when preparing the closure plan:

- Climate: The periods of heavy rainfall that occur throughout the wet season could cause soil erosion, particularly in exposed areas where vegetation has been removed and high velocity sheet flow or gully storm water run-off occurs;
- Protected areas: The presence of CDNP and TNR is likely to result in substantial stakeholder input with respect to post-closure land use and ecological rehabilitation for the Sintoukola Project;
- Sensitive marine and coastal ecosystems: The shoreline in the vicinity of the proposed jetty and other coastal facilities includes a number of ecosystems which may be sensitive to changing hydrological conditions;
- Migration: Out-migration is common and driven by the search for employment / income generation. Migrants that succeed in finding jobs tend not to return to their home village until retirement or until a change in circumstances forces them to return; and
- Livelihood strategies: Farming is the most widely practiced livelihood strategy within the Sintoukola Project area and the final post-closure land use should be complementary to this.

18.7.5 Closure Assumptions

As the Sintoukola Project is currently at the PFS stage, baseline and engineering design studies are still ongoing. Consequently, a number of assumptions about the closure and the reclamation approach for the site have been made. During the feasibility Sintoukola Project phase, this conceptual closure plan will be updated to ensure that the approach, design basis and assumptions reflect the feasibility study project description. A full list of assumptions is provided in the conceptual closure plan presented in Volume IX but key items are listed below:

- The service road will remain the property of the government throughout the Sintoukola Project and following its closure;
- Some facilities will be handed over to a third party and will remain in use post closure:
  - accommodation camp,
  - administrative buildings (at the coastal facility),
  - power transmission infrastructure,
  - gas pipeline, and
  - jetty facilities and breakwater.
- Soil and cover materials will be salvaged or re-used within the Sintoukola Project area; an assumed 5% deficit will be addressed by the import of material from Pointe Noire;

- Quarries for concrete and road aggregate are external to the Sintoukola Project and have not been included within the closure plan;
- The haul road will be reclaimed at closure to reduce accessibility in CDNP;
- No waste rock storage facilities will be present at closure as waste rock from the shaft sinking will be incorporated into the landfill and earthworks required for construction of road embankments and site buildings; and
- No long-term water treatment is required.

18.7.6 Closure Actions

In the absence of stakeholder input regarding the preferred final post-closure land use of the site, SRK have assumed the Sintoukola Project site will be returned to pre-mining conditions with the exception of infrastructure which may be transferred to third parties. The following initial closure concepts have been developed for the key project components to meet this land use:

- The underground shaft access points will be plugged using a concrete cap;
- The haul road, will be regraded, ripped and revegetated with native species in accordance with revegetation management measures developed during the environmental management planning phase prior to construction and refined during the life of mine;
- Except as noted in Section 18.7.5, all buildings, concrete containment areas, and facilities will be removed. This would involve removing equipment, decontamination of buildings and surrounding soils, salvaging material for reuse, recycling or disposal, breaking walls and foundations to grade, and ripping and revegetating the cleared areas;
- The RSF will be capped and revegetated to control the entry of water and erosion;
- Buried sections of pipes will be capped and left in place; surface pipelines will be removed;
- Inert, non-hazardous demolition waste will be disposed of in the underground workings (if generated near the mine site). Hazardous waste generated during decommissioning will be temporarily stored on site and disposed of in a licensed facility;
- All fuel storage and dispensing facilities will be removed; any contaminated soils will be removed for treatment or disposal at a licensed facility; and
- Surface water monitoring, groundwater monitoring and stability monitoring of some site components (i.e. RSF) will be required after closure to ensure the effectiveness of the closure approach. Roads required for post-closure monitoring will be maintained during the post-closure monitoring periods.

18.7.7 Closure Costs

Preliminary closure costing for the Sintoukola Project was carried out using the Standardized Reclamation Cost Estimator (SRCE) model which provides a systematic methodology for mine closure cost estimates.

Considering the assumptions above, the total direct closure costs for the Sintoukola Project are estimated at approximately US$18.0M. With add-ons such as engineering, design, and construction plans, contingency, insurance, and contractor profit, the grand total is equal to approximately US$22.5M.

19 Capital and Operating Costs (Item 21)

19.1 Capital Cost Estimates

Each consultant developed capital costs for their respective scope areas, and are all based in US dollars (US$). These estimates allowed for direct costs only, and SRK included an allowance for indirect costs in the economic modelling (Section 20).

The capital cost estimates provided by each consultant are based on their work performed as part of the PFS design, project scope of facilities, civil, electrical and mechanical equipment lists and quantities, PFD's and/or drawings, and budget quotations from suppliers, as purchased quotations from other projects and/or in-house data.

For the purpose of the Sintoukola PFS, initial capital is defined as any capital spent during construction and the ramp up period, while any capital spent after reaching nameplate capacity (2.0 Mtpa MoP), referred to as production start, is considered sustaining capital, unless otherwise specified.

19.1.1 Mining

Underground Mining

A capital estimate based on the selected mining method and mine production schedule was prepared by SRK. Total mine and mine site capital costs include shaft sinking, vertical conveyance, mine equipment, conveyors, ventilation, mine services, surface equipment and buildings, and capitalized pre-production operating costs. Sustaining capital is recognized after six months following production start and is determined by accumulated equipment hours and mine ramp up schedule. Table 19.1.1.1 shows the mine capital summary.

Table 19.1.1.1: Mine Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
Shaft Sinking		146,218	-	146,218
Vertical Conveying		18,308	7,373	25,681
Mine Equipment		47,208	132,748	179,955
Conveying		35,311	29,386	64,697
Mine Ventilation		2,307	200	2,507
Refrigeration		3,720	-	3,720
Mine Services		4,675	150	4,825
Capitalized Pre Production Operating Costs		18,411	-	18,411
Subtotal Underground Mining		276,157	169,856	446,013
Contingency	20.0%	55,231	33,971	89,203
Total		331,388	203,827	535,215

Material Handling

The capital cost estimated for the Mine Site Material Handling for the Sintoukola Project is summarized in Table 19.1.1.2.

Table 19.1.1.2: Mine Site Material Handling Capital Summary

Description	Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Surface Material Handling (Stockpile/Load out) (by AMEC)	36,789	4,395	41,185
Contingency	8,873	1,214	10,086
Total	**45,662**	**5,609**	**51,271**

Mine Buildings and Surface Facilities

The capital cost estimated for the ancillary surface facilities at mine site is summarized in Table 19.1.1.3.

Table 19.1.1.3: Mine Site Surface Facilities Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
Mine Buildings and Utilities		39,623	4,600	44,223
Contingency	17.2%	6,815	791	7,606
Total		46,438	5,391	51,829

19.1.2 Hauling and Road Train

The installation of the wearing course of the haul road is postponed until the first shutdown period for process plant maintenance, and therefore the corresponding costs appear in the sustaining capital column. The capital cost estimated for the hauling is summarized in Table 19.1.2.1.

Table 19.1.2.1: Hauling and Road Trains Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
Haul Road		101,216	12,137	113,353
Road Train and Other Vehicles		14,448	23,558	38,006
Subtotal Hauling		**115,664**	**35,694**	**151,358**
Contingency	15.0%	17,350	5,354	22,704
Total Hauling		**133,013**	**41,048**	**174,062**

19.1.3 Processing

Process Site

The process plant presented in this report has allocated space within the dry process area for five compactors, with three being installed initially, with the installation and costs of the final two compactors being deferred two years after initial commissioning. The cost for these two additional compactors has been estimated to be US$19.9M, and is considered sustaining capital. The cost includes a multiplier of 1.5 on the labour to adjust for the brownfield work, but does not include any indirect costs.

Also, the plan is to initially construct a 100 kt product storage facility. In year 4 of production, the product storage facility will be expanded to achieve a total capacity of 200 kt. This allows initial capital to be deferred until the mine has reached production capacity and the added warehouse storage can be utilised. The capital cost to expand the product storage warehouse is US$23.5M, and is considered sustaining capital. This cost includes expanding the building by extending its length by 105 to 305 m, removing the end wall and extending the conveyor terminations.

An allowance of 3.6% of mechanic equipment costs has been included in the sustaining capital estimate.

The capital cost estimated for the Process Site for the Sintoukola Project is summarized in Table 19.1.3.1.

Table 19.1.3.1: Process Plant Capital Summary

Description		Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Ore Handling		14,323	2,790	17,114
Wet Process		83,036	16,175	99,211
Dry Process		68,154	33,056	101,210
Product Dispatch		43,424	32,018	75,442
Tailings Process		30,541	5,949	36,490
Reagents		3,912	762	4,674
Process Brine & Water		6,104	1,189	7,292
Process Building and Utilities		212,545	41,401	253,946
Subtotal Process Site Processing Capital Cost (by AMEC)		462,039	133,340	595,379
Contingency	24.0%	110,715	31,951	142,666
Total Process		**572,754**	**165,292**	**738,045**

Ancillary Surface Facilities

The capital cost estimated for the ancillary surface facilities at process site is summarized in Table 19.1.3.2.

Table 19.1.3.2: Process Site Capital Summary

Description	Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Non Process Specific Buildings found at the Process/Marine Site	16,829	18,501	35,330
Waste Treatment Plant	4,271	1,657	5,928
Power Distribution	5,151	297	5,448
Process Site Utility Network	4,888	826	5,714
Process Site Platform	42,647	-	42,647
Subtotal Ancillary Surface Facilities	73,787	21,280	95,067
Contingency	11,068	3,192	14,260
Total Ancillary Surface Facilities	84,855	24,473	109,327

19.1.4 Marine

The capital cost estimated for the marine facilities and transshipment equipment is summarized in Table 19.1.4.1.

Table 19.1.4.1: Marine and Transhipment Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
General		4,428	515	4,943
Trestle & Loading platform		26,192	5,708	31,900
Berthing & Mooring Facilities		7,675	2,719	10,395
Breakwater		33,357	6,661	40,018
Barges & Workboat		42,270	29,923	72,193
Material Handling Equipment		10,716	-	10,716
Subtotal Marine Facilities		**124,639**	**45,525**	**170,164**
Contingency	11.7%	14,614	5,891	20,505
Total Marine Facilities		**139,253**	**51,416**	**190,669**

19.1.5 Solid Residue and Brine Disposal

19.1.5.1 Solid Residue

The initial construction allows for a starter berm and lined storage facility, while the sustaining capital has allowed for five subsequent raises.

The capital cost estimated for the RSF is summarized in Table 19.1.5.1.1.

Table 19.1.5.1.1: RSF Capital Summary

Description		Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
RSF		10,545	9,249	19,794
Contingency	14.2%	1,582	1,387	2,969
Total RSF Facility		**12,127**	**10,637**	**22,763**

19.1.5.2 Brine Disposal

The capital cost estimated for the salt brine management is summarized in Table 19.1.5.2.1.

Table 19.1.5.2.1: Brine Disposal Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
Salt brine Management		31,891	6,528	38,418
Contingency	15.0%	4,784	979	5,763
Total Salt brine Management		**36,674**	**7,507**	**44,181**

19.1.6 Employee Facilities

The capital cost estimated for the employee facilities is summarized in Table 19.1.6.1.

Table 19.1.6.1: Employee Facilities Capital Summary

Description	Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Buildings	32,603	12,429	45,032
Utilities	1,838	209	2,047
Platform & Access	8,199	-	8,199
Capitalized Operating Costs	4,743	-	4,743
Subtotal Employee Facilities	47,383	12,637	60,021
Contingency	6,059	1,796	7,855
Total Employee Facilities	**53,443**	**14,433**	**67,876**

19.1.7 General Infrastructure

The initial capital allows for road upgrades sufficient for construction, while the final road construction is deferred and has been scheduled in the sustaining capital column.

The capital cost estimated for infrastructure is summarized in Table 19.1.7.1.

Table 19.1.7.1: General Infrastructure Capital Summary

Description		Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)
Communications Network		1,152	-	1,152
Power Supply		34,826	-	34,826
Gas Pipeline - drying only		38,905	-	38,905
Subtotal Power & Gas		**73,731**	**-**	**73,731**
ET1 (existing track to mine site)		5,125	58,899	64,024
ET2 (existing track to Tchiboula)		2,322	5,356	7,678
NT (new service track Tchiboula to ET1)		2,744	3,039	5,783
N5 (upgrade & repair)		-	-	-
Subtotal Roads		10,191	67,293	77,485
Water supply		11,208	3,166	14,374
Sewerage		1,396	-	1,396
Subtotal General Infrastructure Capital		**97,679**	**70,459**	**168,138**
Contingency	15.7%	15,374	11,090	26,464
Total General Infrastructure		**113,053**	**81,549**	**194,601**

19.1.8 Owner Costs

Owner's costs were calculated for the duration of the project and scheduled to coincide with the overall project implementation schedule. They are grouped into the following three main categories:

- Staffing. The owner's staff for project implementation are as follows:
    - A head office staff of nine people, composed mainly of senior management, including project director and manager, engineering and project controls managers, an expeditor, materials manager and safety and security officer;
    - A site staff of fourteen people, including managers for the functional areas, supervisors, IT staff, planner, safety officer, security officer and others;
    - An allowance has been made for a commissioning team of twenty people for a period of six months; and
    - A cost for the time of various consultants has also been included.

- Travel and accommodation. The travel costs accounts for transport around site, travel from Pointe Noire to site, local and international airfares, as well rest and recuperation for the owner's site staff. Accommodation accounts for hotel costs both internationally and in the ROC and includes the lodging of the owner's team on site during construction;
- Provisional and general costs. These costs are made up of the following items:
  - Head office administrative costs that reflect the cost of maintaining the ELM head office in Johannesburg;
  - Site surveys as required as part of the FS;
  - Third party inspections during construction;
  - Government relations, as an ongoing activity throughout project implementation;
  - Permits;
  - Office consumables and software;
  - Land purchase, compensation and resettlement, before start of construction;
  - Exploration, before and during the FS;
  - Technical studies, including the completion of the FS;
  - Temporary medical clinic in support of construction activities;
  - Community relations throughout the project implementation period;
  - Environmental permitting and mitigation during construction;
  - Mobile equipment and staff transport, including cranes, fire trucks, light vehicles, staff buses, fork lifts, etc. required for construction and operations;
  - Barge hire during construction, to account for material transport to site while the bridge crossing the Kouilou river is being repaired;
  - Recruitment and training facilities; and
  - Recruitment and training of operations staff.
- First fills for process equipment at 2% of equipment costs (AMEC); and
- Scaffolding was originally estimated at 4% (AMEC). ELM had contacted local contractors for estimates and revised the scaffolding costs to be 1% of total costs (ELM).

A notable exclusion from these costs is the contractor's camp. ELM plans to use part of the footprint and facilities from the final operations employee facilities for the construction camp. Therefore, these costs have been included in the infrastructure cost estimate. In addition, these costs have been reflected in some of the unit pricing, as various construction contractors will bring their own offices and facilities.

A summary of the Owner's Cost estimate is presented in Table 19.1.8.1.

Table 19.1.8.1: Owner's Cost Summary

Description	**Initial Capital (US$'000)**	**Sustaining Capital (US$'000)**	**LoM Capital (US$'000)**
Staffing Cost	11,558		11,558
Travel and Accommodation	5,238		5,238
Provisional & General Costs	60,954		60,954
Subtotal Owner Costs	77,750	-	77,750
Sunk Costs (Costs before Q3, 2013)	(27,601)		(27,601)
Scaffolding, Spares and First Fills	9,087		9,087
Closure Costs		24,100	24,100
Total	59,236	24,100	83,336

19.1.9 Project Indirect Costs

Project indirect costs include the following:

- Factors are applied to the direct cost estimate for legal, finance and insurance;
- Capital spares for process equipment at 5% of equipment costs (AMEC);
- Vendor representatives at 0.5% of equipment costs (AMEC);
- Engineering, Procurement, and Construction Management (EPCM) at 12.5% (ELM);
- Contingency was applied as a factor to the total direct costs provided by each consulting company; and
- Escalation was not allowed for.

19.1.10 Capital Cost Summary

The capital costs for all disciplines are summarized in Table 19.1.10.1.

Table 19.1.10.1: Capital Cost Summary

Description	Initial (US$ 000s)	Sustaining (US$ 000s)	LoM (US$ 000s)
Mining	352,569	178,852	531,420
Haul Road & Road Trains	115,664	35,694	151,358
Processing	535,825	154,621	690,446
Marine	124,639	45,525	170,164
Subtotal Waste & Brine	42,435	15,777	58,212
Employee Facilities	47,383	12,637	60,021
General Infrastructure	97,679	70,459	168,138
Owner's Costs	59,236	24,143	83,336
Subtotal Capital Costs	1,375,430	537,709	1,913,139
Contingency	252,464	97,615	350,079
Subtotal Capital + Contingency	1,627,894	635,324	2,263,218
EPCM	203,543	-	203,543
Insurance	19,540	-	19,540
Capital Expenditures - 2Q 2012	1,850,977	635,324	2,486,301

19.2 Operating Cost Estimates

All tables below show the operating costs as extracted from the economic model, which is based on the estimated LoM operating costs divided by the total MoP in the mine schedule. The operating costs presented in the discipline reports may vary slightly from those below, as they were based on a constant MoP production of 2.00 Mtpa.

Operating cost utilized diesel at $1.00 per litre, power costs of $0.07/kWh and natural gas costs of $0.11/m^3 for all discipline groups.

19.2.1 Mining

Underground Mining

Mine operating costs were calculated as a function of the production schedule and equipment operating and consumption factors. Main consumables costs were based on estimated local delivered prices, while other costs were estimated in US$ and Euros. Table 19.2.1.1 shows the mine operating cost summary, and Table 19.2.1.2 shows the mine operating cost by element.

Table 19.2.1.1: Underground Mining Operating Cost

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)	Cost - (US$ / t Mined)
Hoisting	51,756	1.15	0.34
Continuous Miner - Dev Ore	12,841	0.29	0.08
Continuous Miner - Prod. Ore	258,608	5.76	1.70
Shuttle Cars	221,925	4.94	1.46
Conveying	136,234	3.03	0.90
Bolters	11,833	0.26	0.08
UG Support Equip	114,529	2.55	0.75
Stockpile Rehandle	2,307	0.05	0.02
Ventilation	50,487	1.12	0.33
Dewatering	91	0.00	0.00
Ground Support	11,440	0.25	0.08
Utilities	99,612	2.22	0.66
Mine Operations Labor	224,969	5.01	1.48
Total	**1,196,632**	**26.63**	**7.89**

Table 19.2.1.2: Underground Mining Operating Cost by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)	Cost - (US$ / t Mined)
Labor	151,183	3.36	1.00
Fuel	28,589	0.64	0.19
Power	199,191	4.43	1.31
Consumables	605,525	13.48	3.99
Materials	212,144	4.72	1.40
Total	**1,196,632**	**26.63**	**7.89**

Material Handling

Operating costs by element have been estimated for the material handling facilities at mine site previously described, and are shown in Table 19.2.1.3.

Table 19.2.1.3 Material Handling Operating Cost by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)	Cost - (US$ / t Mined)
Labor	9,837	0.22	0.06
Fuel	993	0.02	0.01
Power	-	-	-
Consumables	-	-	-
Materials	-	-	-
Total	**10,829**	**0.24**	0.07

Mine Buildings and Surface Facilities

Operating costs have been estimated for the ancillary surface facilities at mine site previously described, and are shown in Table 19.2.1.4

Table 19.2.1.4: Mine Surface Facility Operating Cost by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)	Cost - (US$ / t Mined)
Labor	20,182	0.45	0.13
Fuel	-	-	-
Power	5,321	0.12	0.04
Consumables	438	0.01	0.00
Materials	10,009	0.22	0.07
Total	**35,949**	**0.80**	0.24

19.2.2 Hauling and Road Trains

Fixed operating costs include labour, maintenance material and other items. Personnel for the hauling, maintenance and operation consist of 209 staff. Maintenance material costs have been calculated on the basis of a maintenance program.

Variable operating costs are the costs that are dependent on production levels, and include fuel and consumables, mainly tires.

The operating costs are shown in Table 19.2.2.1.

Table 19.2.2.1 Hauling and Road Train Operating Cost by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)	Cost - (US$ / t Mined)
Labor	116,648	2.60	0.77
Fuel	93,179	2.07	0.61
Power	-	-	-
Consumables	76,716	1.71	0.51
Materials	303,686	6.76	2.00
Total	**590,230**	**13.14**	3.89

19.2.3 Processing

<u>Process Site</u>

Fixed operating costs are the costs associated with nominal capacity, including labour and maintenance material. Personnel for the process plant site consist of 91 staff and 239 hourly. The personnel makeup will be 28 Expatriates and 302 Nationals. Maintenance material costs have been calculated as 5% of the capital cost of the mechanical equipment. This rate is based on the historical relationship between these values.

Variable operating costs are a function of the production levels and include electrical, natural gas and consumables. Raw materials required for the production of potash are water, fuel, energy and reagents as discussed in the previous sections.

The primary component in the consumable category is reagents. For the process described, the reagents used are flotation amine, flotation oil, frother, and depressant used in flotation; flocculant and dispersant used to enhance settling; and dedusting oil and anti-caking agents applied to finished product prior to shipping to prevent degradation during transportation. The reagent addition rates and prices are based on AMEC's work on other projects. The consumables also include an allowance for hydraulic and lube oils used in the process plant, diesel fuel for mobile equipment, screen cloths, baghouse bags, and hydrocyclone wear parts.

The water and seawater operating costs are included in the infrastructure costs.

The expected operating costs are shown in Table 19.2.3.1.

Table 19.2.3.1: Process Site Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	277,142	6.17
Fuel	41,128	0.92
Power	229,500	5.11
Consumables	357,580	7.96
Materials	109,664	2.44
Total	**1,015,012**	**22.59**

Ancillary Surface Facilities

Operating costs have been estimated for the ancillary surface facilities at process site previously described. The expected operating costs are shown in Table 19.2.3.2.

Table 19.2.3.2: Ancillary Surface Facility Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	24,502	0.55
Fuel	-	-
Power	6,275	0.14
Consumables	-	-
Materials	11,236	0.25
Total	**42,013**	**0.94**

19.2.4 Marine

Operating costs have been estimated for the marine facilities and transshipment previously described and are shown in Table 19.2.4.1.

Table 19.2.4.1: Marine and Transshipment Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	42,533	0.95
Fuel	24,903	0.55
Power	3,550	0.08
Consumables	-	-
Materials	4,845	0.11
Total	**75,831**	**1.69**

19.2.5 Solid Residue and Brine Disposal

19.2.5.1 Solid Residue

Operating costs have been estimated for the RSF previously described, and are shown in Table 19.2.5.1.1.

Table 19.2.5.1.1: Solid Residue Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	5,534	0.12
Fuel	23	0.00
Power	3,181	0.07
Consumables	-	-
Materials	4,796	0.11
Total	**13,534**	**0.30**

19.2.5.2 Brine Disposal

Operating costs have been estimated for the salt brine management facilities previously described, and are shown in Table 19.2.5.2.1.

Table 19.2.5.2.1: Brine Disposal Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	6,241	0.14
Fuel	-	-
Power	7,879	0.18
Consumables	-	-
Materials	16	0.00
Total	**14,136**	**0.31**

19.2.6 Employee Facilities

Operating costs have been estimated for the employee facilities previously described, and are shown in Table 19.2.6.1.

Table 19.2.6.1: Expected Employee Facility Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	13,805	0.31
Fuel	-	-
Power	2,618	0.06
Consumables	206,990	4.61
Materials	5,423	0.12
Total	**228,837**	**5.09**

19.2.7 General Infrastructure

The power costs account for the pumps for the sea water intake and the fresh water supply, and the power distribution losses.

The expected operating costs are shown in Table 19.2.7.1.

Table 19.2.7.1: General Infrastructure Operating Costs by Element

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Labor	8,812	0.20
Fuel	20,081	0.45
Power	-	-
Consumables	1,086	0.02
Materials	59,951	1.33
Total	**89,930**	**2.00**

19.2.8 Owner Costs

These General and Administrative (G&A) costs represent a recurring annual operating cost, to cover all owner's expenses during operations. These costs include a labour force of approximately 130 people to manage the following:

- Health, safety and security;
- Human resources;
- Social and community;
- Information technology (IT);
- Finances;
- Purchasing and warehousing;
- Sales and marketing;
- Warehousing; and
- Employee transport.

Operational G&A also accounts for office supplies and consumables, ongoing training costs, operations of the employee transport and general sales & marketing expenses. The costs are summarised in Table 19.2.8.1.

Table 19.2.8.1 Owner Costs

Item	LoM (US$ 000s)	Cost - (US$ / tonne MoP)
Owner Staffing	115,722	2.58
Office Supplies & Expenses	40,800	0.91
Training Costs	21,103	0.47
Employee Transport	8,273	0.18
General Maintenance & Engineering Department	1,279	0.03
Sales and Marketing	68,000	1.51
Communications Systems	363	0.01
Environmental	13,079	0.29
Total	**268,618**	**5.98**

19.2.9 Operating Cost Summary

Operating costs are estimated at US$79.71 / tonne of product free on board (FOB) and shown in Table 19.2.9.1.

Table 19.2.9.1: Life of Mine Operating Costs by Cost Center

Item	LoM (US$ 000s) (Q2, 2012)	Cost - (US$ / tonne MoP) (Q2, 2012)
Mining	1,243,410	27.67
Hauling and Road Trains	590,230	13.14
Processing	1,057,026	23.53
Marine	75,831	1.69
Solid Residue and Brine Disposal	27,671	0.62
Employee Facilities	228,837	5.09
General Infrastructure	89,930	2.00
Owner Costs	268,618	5.98
Total Operating Cost	3,581,553	**79.71**

20 Economic Analysis (Item 22)

The results of Economic Modelling performed by SRK are presented in Volume X (SRK, 2012e).

20.1 Principal Assumptions

The economic results were derived from a combination of monthly and annual inputs provided by each consultant. All economic inputs were estimated in Q2, 2012 US dollars. As the project timeline envisions implementation commencing in July, 2013, SRK brought the project capital and operating cost inputs to Q2, 2013 terms by applying a 2% escalation factor. This factor represents the high end of a compilation of surveys of economists as of September 7, 2012 (Wall Street Journal: Economic Forecasting Survey, September 2012). All costs presented in this section are therefore in Q2, 2013 terms, as shown in Table 20.1.1. Potash price projections were not escalated.

Table 20.1.1: Capital and Operating Cost Escalation Summary

Description	Initial Capital (US$'000)	Sustaining Capital (US$'000)	LoM Capital (US$'000)	Operating Costs US$/ t MoP
Costs - Q2, 2012	1,850,977	635,324	2,486,301	79.71
Costs - Q2, 2013	1,888,151	647,876	2,536,027	81.31

For the purpose of this economic analysis, all costs incurred by ELM through June 30, 2013 are considered sunk costs. These costs are approximately US$27.6M, as noted in Table 19.1.8.1.

An economic model was prepared on an unleveraged (all equity), post-tax basis. The basis and results are presented in this section. Key criteria used in this analysis are summarized in Table 20.1.2.

Table 20.1.2: Assumptions for Economic Modelling

Model Parameter	Technical Input
Project Start Date	1-Jul-2013
Pre-Production Period	3.1 years
Mine Life	23 years
Post Production Period	2 years
Potash Price - CFR Basis for the year 2016 (US$/tonne)	525.00
Potash Price - CFR Basis for the year 2017 (US$/tonne)	545.00
Potash Price - CFR Basis for the year 2018 (US$/tonne)	565.00
Potash Price - CFR Basis for the year 2019 (US$/tonne)	605.00
Potash Price - CFR Basis for the year 2020 and beyond (US$/tonne)	605.00
Deduct for Standard Quality Product – US$/tonne	15.00
Freight to Market - Average Life of Project – US$/tonne	24.84
Royalty	3%
Government Free Carried Interest	10%
Corporate Tax Rate	30%
Depreciation Period (years)	8
Discount Rate	10.0%
Tax Holiday	5 Years

The mine will have a 23 year life given the production plan described in this report. A two year post-production period is assumed during which infrastructure operations will be turned over to local operators, operating costs will cease and asset retirement obligations, or closure costs, will be incurred.

No difference between financial and tax depreciation is assumed. Depreciation is applied to all non-mining assets on a straight line basis over 8 years. Mine assets are recovered on a unit of production basis over the life of the reserve.

Rates for the costs of Engineering, Procurement, and Construction Management (EPCM) are assumed to be 12.5% of initial capital costs plus contingency. Based on comparable experience in central Africa, rates for insurance costs are set at 1.2% of capital costs.

Working capital changes assume 30 days outstanding on average for accounts receivable and payable and finished product inventories balances are determined from average sales price FOB Pointe Noire multiplied by the tonnes of ore stored.

A corporate tax rate of 30% applies with a maximum loss carry forward period of three years (following a five-year corporate tax exclusion period from production startup). The ROC allows for various depreciation rates. A review of Deloitte Consulting's report (Delliotte, 2012) indicates 8 years straight-line depreciation.

A government free carried interest in the project of 10% has been modelled.

A mining royalty of 3% is due from the date of commencement of effective exploitation. The mining royalty is calculated on EBITDA.

The project discount rate applied to the Sintoukola Project is 10%. Table 20.1.3 shows the exchange rates assumed for the economic analysis. All costs (capital and operating) were adjusted to US dollars.

Table 20.1.3: Exchange Rates per US Dollar

Currency	Exchange Rate per US$
Canadian Dollar	1.00
Congolese Franc	928.00
Euro Dollars	0.77
Great Britain Pound	0.62
South African Rand	7.88
Swiss Franc	0.93
Australian Dollar	0.98
US Dollar	1.00

20.2 Cashflow Forecasts and Annual Production Forecasts

Table 20.2.1 shows annual production and cash flow projections for the life of the project. All production projections, ore grades, process plant recoveries and other productivity measures were developed by independent consultants whose reports are contained earlier in this report. Table 20.2.2 shows summary of the production, cash flow and Economic results.

Table 20.2.1: Production and After Tax Free Cash Flows (US$ '000)

Period	Mine Production (tonnes)	MoP Sold (tonnes)	Cash Flow (US$)	Discounted Cash Flow (US$)	Cumulative Cash Flow (US$)
2013	0	0	(78,034)	(78,034)	(78,034)
2014	0	0	(531,081)	(482,801)	(609,116)
2015	0	0	(733,644)	(606,318)	(1,342,760)
2016	2,369	467	(377,822)	(283,863)	(1,720,582)
2017	6,342	2,053	541,604	369,923	(1,178,978)
2018	6,871	2,151	797,872	495,416	(381,106)
2019	6,874	2,149	826,855	466,738	445,749
2020	6,871	2,158	842,874	432,528	1,288,623
2021	6,875	2,078	797,658	372,113	2,086,281
2022	6,874	1,978	651,357	276,239	2,737,638
2023	6,877	1,993	658,806	253,998	3,396,444
2024	6,861	2,012	608,479	213,268	4,004,923
2025	6,856	1,963	582,305	185,540	4,587,228
2026	6,878	2,005	609,747	176,622	5,196,975
2027	6,882	1,987	576,670	151,855	5,773,645
2028	6,880	1,996	573,627	137,322	6,347,272
2029	6,881	1,974	589,678	128,331	6,936,950
2030	6,874	1,926	583,110	115,365	7,520,060
2031	6,880	1,973	572,477	102,965	8,092,536
2032	6,872	1,990	599,843	98,079	8,692,380
2033	6,862	2,041	609,625	90,617	9,302,004
2034	6,858	2,021	609,179	82,319	9,911,183
2035	6,883	2,008	615,364	75,595	10,526,547
2036	6,817	2,086	639,617	71,431	11,166,164
2037	6,598	2,046	627,581	63,716	11,793,745
2038	5,242	1,678	529,466	48,868	12,323,211
2039	661	200	113,555	9,528	12,436,766
2040	0	0	51,715	3,945	12,488,481
2041	0	0	(2,865)	(199)	12,485,617

Table 20.2.2: Production, Cash Flow and Economic Results Summary (escalated to July 2013 US$)

Category	Units or Value		Pre-Production			Mine Start	Sales Commence					AVG				
		Year	2013	2014	2015	2016	2017	2018	2019	2020	2021	'22-'37	2038	2039	2040	2041
		Annual Period	-3	-2	-1	1	2	3	4	5	6	16 yrs	23	24	25	26
		Total or Average														
Production Statistics																
Ore Mined	kt	151,738				2,369	6,342	6,871	6,874	6,871	6,875	6,852	5,242	661	0	0
Ore Grade	% K_2O	20.0%				20.7%	20.5%	21.2%	21.2%	21.2%	20.4%	19.8%	19.2%	20.4%	0.0%	0.0%
Mined K_2O	kt	30,371				491	1,297	1,456	1,460	1,457	1,404	1,354	1,008	135	0	0
Ore Feed to Plant	kt	151,738				1,538	6,775	6,840	6,840	6,859	6,840					
K_2O to Plant	kt	30,371				315	1,387	1,452	1,450	1,456	1,404					
K_2O Grade to Plant	%					21%	20%	21%	21%	21%	21%					
Process Recovery	%	89.5%				89.6%	89.5%	89.6%	89.6%	89.6%	89.6%	89.5%	89.4%	89.5%	0.0%	0.0%
K_2O In Product	kt	27,183				282	1,242	1,301	1,300	1,305	1,257	1,210	1,015	121	0	0
Product Grade	% K_2O	60.5%				60.5%	60.5%	60.5%	60.5%	60.5%	60.5%	60.5%	60.5%	60.5%	0.0%	0.0%
MoP Product Sold	kt	44,931				467	2,053	2,151	2,149	2,158	2,078	2,000	1,678	200	0	0
Granular MoP Produced		39,404				247	1,410	1,914	1,912	1,920	1,849	1,780	1,493	178	0	0
Standard MoP Produced		5,527				219	643	237	236	237	229	220	185	22	0	0
Sales Volumes, Prices (CFR Brazil) and Delivery Costs																
Granular MoP Price	$/t-MoP	$597				$525	$525	$545	$565	$605	$605	$605	$605	$605	$0	$0
Tonnes Granular Sold	kt	39,404				247	1,410	1,914	1,912	1,920	1,849	$1,780	$1,493	$178	$0	$0
Standard MoP Price	$/t-MoP	$573				$510	$510	$530	$550	$590	$590	$590	$590	$590	$0	$0
Tonnes Standard Sold	kt	5,527				219	643	237	236	237	229	$220	$185	$22	$0	$0
Gross Revenue	$593.89	26,684,246				241,745	1,068,265	1,168,732	1,210,437	1,301,738	1,253,659	1,206,677	1,012,320	120,517	0	0
Freight to Market	$24.84	(1,115,997)				(11,593)	(50,995)	(53,425)	(53,367)	(53,588)	(51,606)	(49,674)	(41,672)	(4,961)	0	0
Royalty	$14.21	(638,303)				(5,313)	(24,980)	(27,532)	(28,750)	(31,394)	(30,142)	(28,928)	(24,450)	(2,898)	0	0
Revenues, Operating Costs, Capital and Economics																
Gross Income	$554.85	24,929,946				224,839	992,291	1,087,774	1,128,320	1,216,756	1,171,911	1,128,075	946,197	112,658	0	0
Total Operating Cost	81.31	3,653,167				47,724	159,640	170,035	169,980	170,295	167,186	163,815	131,190	16,071	0	0
Operating Margin	$473.54	21,276,779				177,114	832,651	917,738	958,340	1,046,461	1,004,726	964,260	815,008	96,587	0	0
Project Capital	$000s	(2,536,027)	(78,034)	(531,081)	(733,644)	(513,092)	(146,000)	(21,507)	(34,179)	(98,457)	(14,656)	(21,087)	(5,069)	(8,594)	(11,459)	(2,865)
Income Tax	$000s	(4,676,351)				0	0	0	0	0	(107,476)	(269,119)	(237,317)	(25,660)	0	0
Working Capital	$000s	(0)				(41,845)	(84,869)	(9,706)	(5,433)	(11,478)	3,693	75	15,674	63,839	68,920	0
Government Deduction	$000s	(1,578,785)	0	0	0	0	(60,178)	(88,652)	(91,873)	(93,653)	(88,629)	(67,413)	(58,830)	(12,617)	(5,746)	0
Free Cash Flow	$000s	12,485,617	(78,034)	(531,081)	(733,644)	(377,822)	541,604	797,872	826,855	842,874	797,658	606,717	529,466	113,555	51,715	(2,865)
Cumulative Free Cash Flow	$000s	0	(78,034)	(609,116)	(1,342,760)	(1,720,582)	(1,178,978)	(381,106)	445,749	1,288,623	2,086,281	7,249,106	12,323,211	12,436,766	12,488,481	12,485,617
Economic Returns Unlevered	IRR	29.3%														
NPV at 10%		2,971,105														
Payback - yrs		6.0														

20.3 Taxes, Royalties and Other Interests

Although the terms of the mining convention are currently in the process of being refined in preparation for the award of the exploitation permit, the main exonerations discussed below have been used in the PFS. These terms are based on terms previously accepted by the Congolese Government for similar mining projects.

20.3.1 Construction Phase

ELM expects to be provided relief from the following taxes:

- Personnel Income Tax for non-resident staff and Directors (20% on gross salary);
- Withholding tax for services performed by non-registered companies in Congo (20% of the contract price);
- Customs taxes and duties except IT royalty of 1%; and
- VAT.

20.3.2 Operational Phase

During a period of five years from the Date of Commercial Production (defined as the date of marketing the final product or the first MoP), SPSA expects to be exempt from the following taxes:

- Corporate income tax (30% on the mining companies' profits);
- Trading taxes (composed of various local taxes but mainly based on the rental value of the buildings);
- Personnel Income Tax for non-resident staff and Directors (20% on gross salary); and
- VAT on fuels and lubricants.

SPSA will benefit from the following tax advantages during the same period:

- Customs taxes and duties reduced to five 5% on importation of any equipment, material, raw material, big equipment, motors, spare parts, consumables necessary for exploitation; and
- Temporary admission regime in suspension of fees for the import and export of machine, large equipment etc. for the development of the project except IT royalty (1%).

Subject to certain conditions, the 5 year period special tax regime described above can be renewed for an additional 5 year period. Extension of the tax holiday has not been included in the economic model.

During the period of validity of the mining agreement, SPSA will benefits from a tax advantage of a 7.7% reduced rate regarding the withholding tax on services performed by non-registered companies in Congo.

20.4 Results

The economic analysis results, shown on Table 20.4.1, indicate a net present value (NPV) for the project at a 10% discount rate of US$2,971M with an internal rate of return (IRR) of 29.3% (after tax). The pay back period from Q3, 2013 is 6 years.

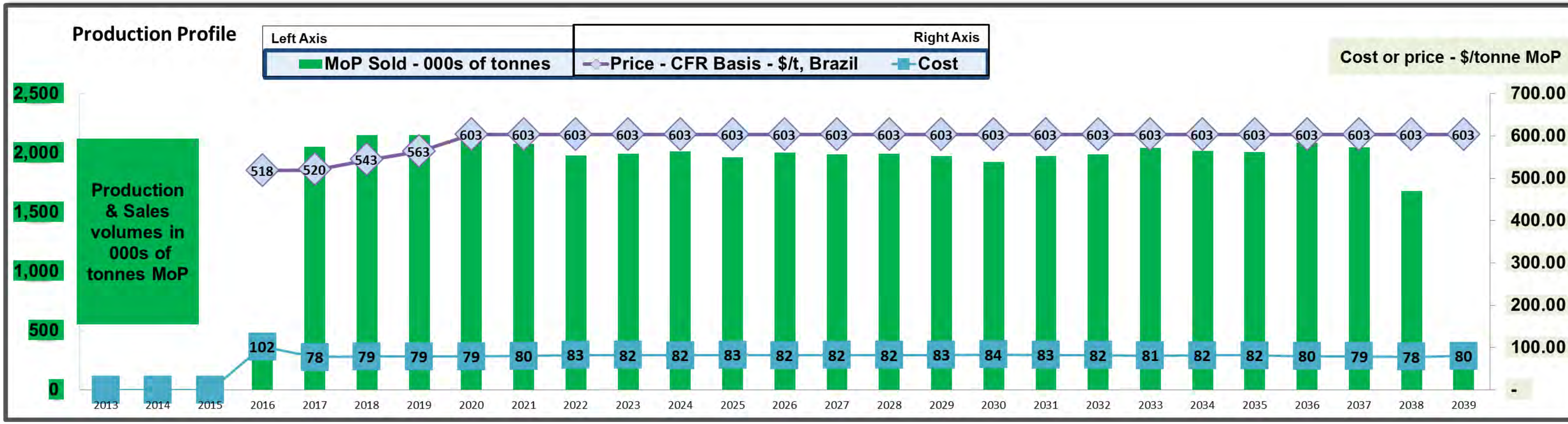
Table 20.4.1: Economic Analysis

Potentially Mineable Resources			Value	
Underground Mine				
	Resource Extracted		151,738	kt
	Ore Mined		151,738	kt
	K_2O Grade		20.02%	
	Mined K_2O		30,371	kt
Processing				
	Ore Delivered		151,738	kt
	K_2O Delivered		30,371	
	Process Recovery		89.50%	
	K_2O in Product		27,183	kt
	Product Grade		60.50%	
	MoP Product Sold		44,931	kt
Sales Volumes, Prices and Delivery Costs				
	Granular MoP Price		$597	$/t-MoP
	Standard MoP Price		$573	$/t-MoP
	Granular Product Sales		39,404	kt
	Standard Product Sales		5,527	kt
	Gross Revenue	$593.89 US$/t	26,684,246	000s
	Freight to Market	$24.84 US$/t	(1,115,997)	000s
	GSR Royalty	$14.21 US$/t	(638,303)	000s
	Gross Income	$554.85 US$/t	24,929,946	000s
Operating Costs				
	Mining	$28.23 US$/t	1,268,279	000s
	Haul Road & Road Trains	$13.40 US$/t	602,034	000s
	Process	$24.00 US$/t	1,078,154	000s
	Marine Loadout	$1.72 US$/t	77,347	000s
	Tailings Dispersion & Storage	$0.63 US$/t	28,224	000s
	Base Camp	$5.19 US$/t	233,411	000s
	Infrastructure	$2.04 US$/t	91,727	000s
	Environmental	$0.30 US$/t	13,340	000s
	Owner	$5.80 US$/t	260,651	000s
	Total Operating Costs	**$81.31 US$/t**	**3,653,167**	**000s**
	Operating Margin	**$473.54 US$/t**	**21,276,779**	**000s**
	Initial Capital		(1,888,151)	000s
	LoM Capital		(2,536,027)	000s
	Income Tax		(4,676,351)	000s
	Government Deduction		(1,578,785)	000s
	Free Cash Flow		12,485,617	000s
	NPV 10%		2,971,105	000s
	IRR		29.3%	

The production profile is shown in Figure 20-1 and the cash flow profile is presented in Figure 20-2.

20.5 Sensitivity Analysis

A sensitivity analysis for key operating and economic parameters is shown in Figure 20-3. The project NPV is most sensitive to changes in potash price, followed by changes to discount rate, tax rate, capital cost and operating costs. Additional sensitivities to power costs, labor costs, exchange rates and fuel costs were performed and the economic results of the project were only marginally affected by these changes.



srk consulting

elemental minerals limited

Sintoukola Potash Project, Republic of Congo

Source: SRK, 2012

Figure 20-1

Production Profile

srk consulting

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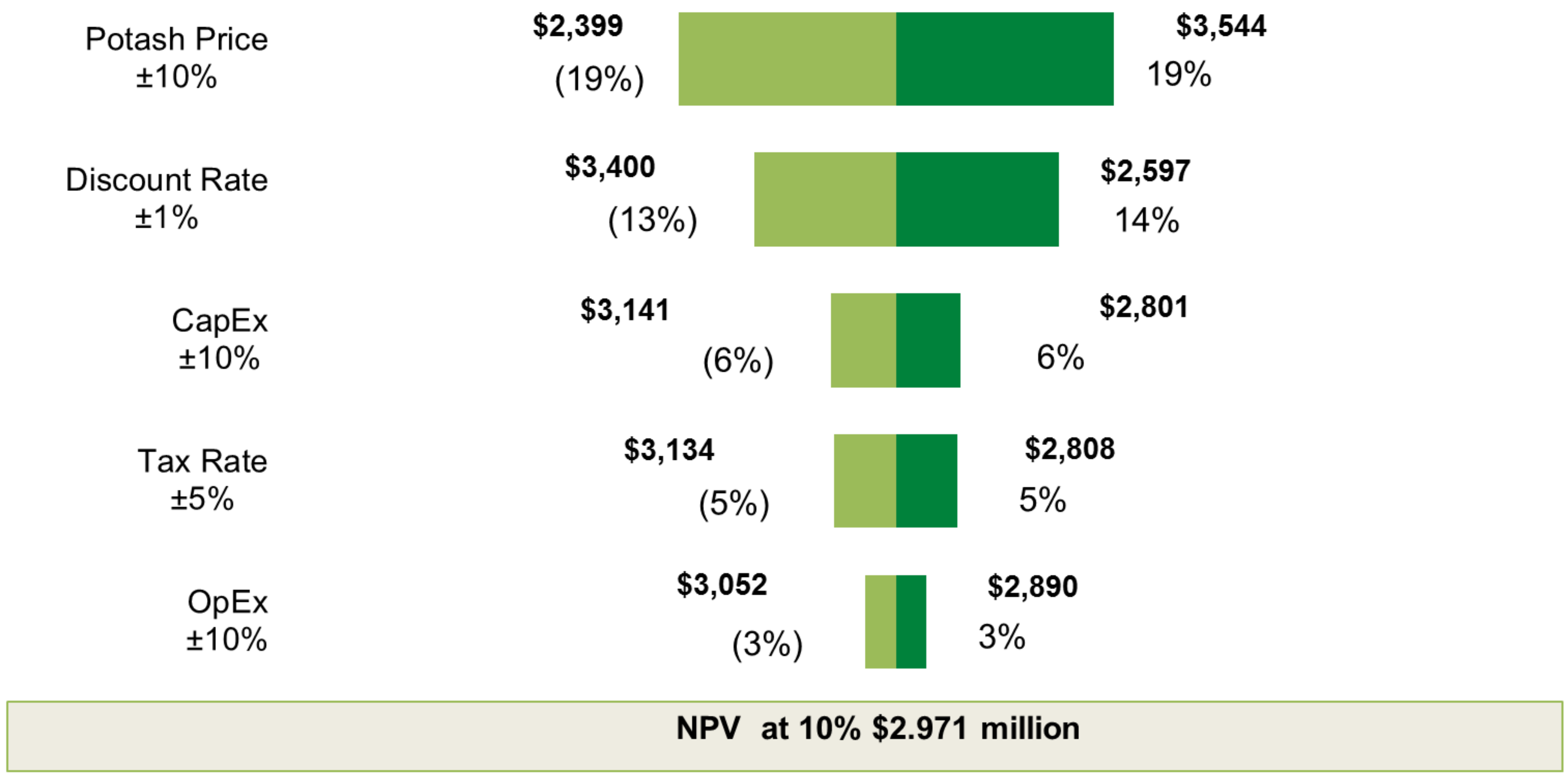
Sintoukola Potash Project,
Republic of Congo

Source: SRK, 2012

Figure 20-2

Cash Flow Profile



srk consulting

elemental minerals limited

Sintoukola Potash Project
Republic of Congo

Source SRK, 2012

Figure 20-3

Sensitivity Analysis Results

21 Adjacent Properties (Item 23)

Within the Congolese Basin there are several occurrences of potash mineralisation that are being explored by other companies. The properties of most significance to the Sintoukola Project are those located on the Makola Exploration Permit area (Rauche and van der Klauw, 2010) and the Mengo Exploitation Permit area (Rauche and van der Klauw, 2008), both of which are held by Mag Industries Ltd (TSX: MAA).

The Makola permit covers an area of 1,111 km^2 directly northeast of the city of Pointe Noire. The Mengo permit is within the Makola permit and covers an area of 136 km^2 surrounding the Mengo carnallite deposit (located approximately 20 km northeast of Pointe Noire). The Makola permit has a NI 43-101 compliant Mineral Resource of 9,197.8 Mt of carnallitite mineralisation at a grade of 12.1% K_2O in the Inferred Category and 70.1 Mt of sylvinite mineralisation at an average grade of 20.7% K_2O, also in the Inferred Category (dated November 18, 2010).

The Makola permit also includes the historic Holle Potash Mine located some 40 km northeast of Pointe Noire and approximately 70 km to the southeast of Sintoukola. The Holle Potash Mine was developed in 1964 by Mines Domaniales de Potasse d'Alsace. Shafts were sunk to intersect the potash deposits and some 21 exploration drillholes were drilled totalling 6,945 m. The Holle mine began production in 1968 and sylvinite was mined for nine years until 1977 when the mine flooded after very high water inflow was encountered during the development of an exploration adit. The mine produced a total of 7.4 Mt of sylvinite at a grade of 27.7% K_2O and produced approximately 3.2 Mt of KCl (de Ruiter, 1979). No further work has been completed at Holle.

The Mengo permit has a NI 43-101 compliant Mineral Resource of 146.7 Mt at 17.4% KCl in the Measured Category, 39.5 Mt at 17.7% KCl in the Indicated Category and 1,215.7 Mt at 17.2% in the Inferred Category (Updated Resource Estimate dated June 10th, 2009). The Mineral Resource estimate identified four mineralised horizons (#1, #2, #3 and #4) which ranged from 10 m to 26 m in thickness and from 44.1% to 90% carnallite (7.47% to 15.25% K_2O). Horizons #2, #3, and #4 are considered to have economic potential for solution mining of carnallitite via hot leaching in double well caverns. Following technical design and financial modelling a NI 43-101 compliant Mineral Reserve of 151.2 Mt at 17.3% KCl in the Proven Category and 40.3 Mt at 17.6% KCl in the Probable Category was estimated for the deposit (dated June 10, 2009).

CSA believes that the properties are significant to the Sintoukola Project as they demonstrate:

- The potential of the region for both sylvinite and carnallitite deposits;
- The opportunities and risks for exploitation by underground or solution mining methods; and
- The supportive nature of the Congolese government towards mining project development.

It should be noted that CSA has been unable to verify the information about the mineralisation on the adjacent properties and that the information is not necessarily indicative of the mineralization on the Sintoukola Project.

22 Other Relevant Data and Information (Item 24)

The results of studies performed by ELM that are discussed in this section are presented in Volume XI (ELM, 2012).

22.1 Staffing Estimate

The labour compliment for the Sintoukola Project was developed from individual estimates for each WBS area, based on first principles. The shift schedule assumed that there would be four shifts, with three shifts onsite at any one time to provide 24 hour coverage. This type of schedule requires a sequence of day shifts, swing shifts, night shifts and days off and is typical for an operating mine and process plant. Refer to Table 22.1.1 for the full labour compliment by level.

Table 22.1.1: Expected Labour Compliment for the Sintoukola Project

		Discipline							
Level	**Position**	**Mine**	**Hauling**	**Process**	**All Infrastructure**	**Owner's G&A**	**Total Labour**	**Total in Camp (1)**	**Total Off Camp**
Management	Site manager	2	0	1	1	3	**7**	7	0
	Engineer	13	0	30	5	0	**48**	48	0
	Manager	12	1	1	0	9	**23**	23	0
	Supervision	8	0	14	2	2	**26**	26	0
Technician/Team Chief	Superintendents	24	0	2	3	23	**52**	39	13
	Supervision (Foremen)	32	9	28	1	0	**70**	52	17
Skilled and semi-skilled workers	Skilled Labour (Artisan / Journeymen)	120	89	63	46	30	**348**	261	87
	Semi-skilled labour	287	98	42	0	34	**461**	346	115
	Clerical	0	3	8	0	27	**38**	29	10
Labourers (unskilled)	Unskilled labour	122	6	141	35	0	**304**	152	152
	Total	**621**	**206**	**330**	**93**	**128**	**1378**	**983**	**394**

It is expected that the Management will follow an expat rotation schedule and will need to be housed at the Employee Facility on a full time basis. The Technical Supervision through Laborer categories will follow the four shift rotation. Provision has been made for the bulk of these employees to be housed in the Employee Facility while on a working rotation. It has been assumed that a number of employees will be recruited from the surrounding villages and the nearby town of Madingo-Kayes and will therefore stay in private accommodation.

22.2 Training and Recruitment for Operational Staff

ELM engaged the Forhom Institude, a division of Egis International, to assess the Congolese labour market and available training facilities in the ROC in order to make recommendations for the recruiting and training of the required labour compliment. These recommendations are summarized below.

22.2.1 Congolese Labor Market

According to the International Labour Office (ILO) criteria, the unemployment rate in Congolese cities is up to 18%. The youth is particularly affected, with a rate of 29% for people under 30 (42% when taking into account a wider assumption of unemployment). Unskilled manpower is widely available. By contrast, skilled workers, technicians and engineers are a scarce resource in the ROC in all sectors, as most of the industrially qualified manpower is currently employed in the petroleum and para-petroleum industries.

22.2.2 Recruitment Strategy

ELM's recruitment strategy is based on the following objectives in terms of local employment:

- Management and supervisory level: 80% Congolese compliment at full operation;
- Skilled labour level: 85% Congolese compliment at full production;
- Semi-skilled and unskilled level: 100% Congolese compliment at full production; and
- An average annual labour turnover of 10% is assumed.

22.2.3 Local Recruitment Sourcing

Unskilled Manpower

ELM does not expect difficulty in recruiting unskilled staff near the mine site.

Skilled and Semi-Skilled Manpower

Due to the shortage of qualified artisanal skills in the Congolese labour market and considering the expected increase of industrial activities in the ROC, recruitment of skilled and semi-skilled workers will be the main challenge to be faced by the project in terms of human resource. ELM will pursue the following recruitment initiatives:

- Retention of skilled artisans from the construction phase;
- Local employment agencies;
- Develop partnerships with reliable vocational training schools in order to identify and recruit skilled graduates; and
- Make use of local media.

Technicians and Engineers

The main source of local recruitment for higher qualified staff will be technical and engineering schools. Given the shortage of new graduates (around 60 technicians and 60 engineers for the whole country) and the increasing demand from industry, ELM will develop partnerships with appropriate vocational schools within the ROC. Support will be provided in the areas of:

- Training infrastructure;
- Training and coaching capacity;
- Trainers' skills; and
- Management development potential.

22.2.4 International Recruitment

Due to the specialized nature of potash operations, ELM has planned for a group of expats as part of the construction and operational phases of the mine. In addition to managing the day-to-day operations, these expats will also be tasked with training local employees, so that over time the number of expats will be reduced.

22.3 Project Implementation Schedule

ELM intends to continue the FS upon completion of the PFS for the Sintoukola Project. The detailed PFS schedules for engineering and construction of the infrastructure, process plant and the mine have been combined into a master schedule, presented in Figure 22-1. The implementation strategies that will be pursued are discussed in the following sections.

22.3.1 Mine

A civil engineering approach has been selected for sinking the twin shafts. On completion of sinking, one shaft will be equipped with the materials handling system, while the other will be used to develop the shaft bottom and mine infrastructure. On this basis, mine production commences six months before the process plant has been commissioned. This early production will be stored on a RoM stockpile at the mine site.

22.3.2 Process Plant

The process plant schedule is currently the project critical path item. In order to optimize the construction schedule, a modular approach has been adopted. Front end and basic engineering will be done in parallel with the FS, followed by detailed engineering, early procurement of long lead items and initial construction set up work.

22.3.3 General Infrastructure

General infrastructure construction is not on the critical path. However, to support the mine, process plant and port construction, infrastructure early works are required, as follows:

- Employee Facilities : The construction camp used to house the construction labour and will be subsequently expanded as part of the Employee Facilities;
- Preparatory works: Some early earthworks are required to improve the current roads and prepare the process plant site; and
- Haul Road: Early engineering and tendering will start on the haul road towards the latter part of the FS to construct the road over the dry seasons.

22.3.4 Implementation Schedule – Key Dates

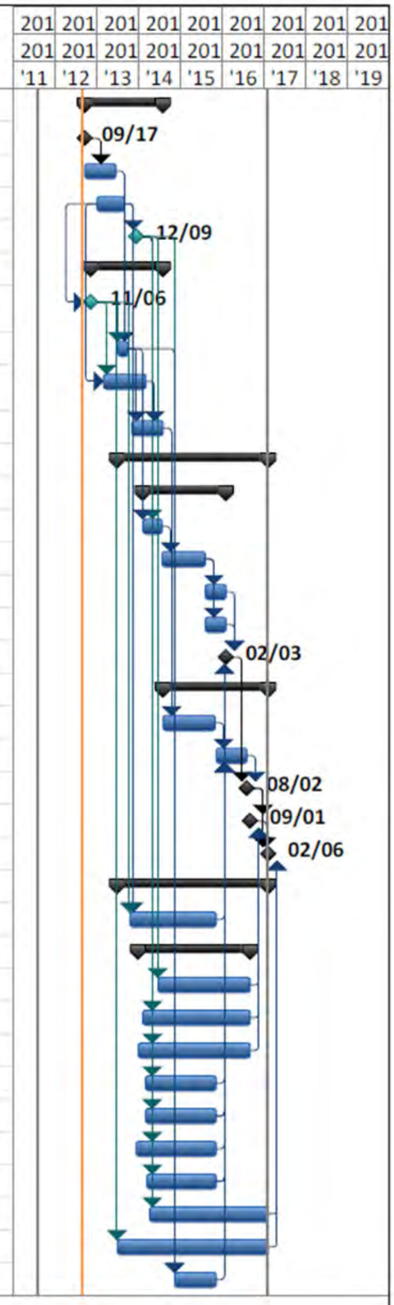
Key milestones are summarised as follows:

- Complete FS: Q3 2013;
- Full construction implementation commences: Q4 2013;
- Shaft sinking commences: Q1 2014;
- Mining commences:Q1 2016;
- First ore through the process plant:Q3 2016;
- First product shipped: Q3 2016; and
- Achieve nameplate capacity (2 Mtpa MoP): Q1 2017.

22.3.5 Implementation Strategy

In order to achieve the timeline described above, ELM will strengthen the Owner's Team during the FS and early works. The EPCM approach to be followed by ELM will be addressed during the FS.

ID	Task Name	Duration	Start	Finish
1	**Study Work**	**489 days**	**Mon 12/09/17**	**Thu 14/07/31**
2	Prefeasibility Study	0 days	Mon 12/09/17	Mon 12/09/17
3	SEIA and Permitting	193 days	Mon 12/09/17	Wed 13/06/12
4	Definitive Feasibility Study	174 days	Tue 13/01/01	Fri 13/08/30
5	Start full Project Implementation	0 days	Mon 13/12/09	Mon 13/12/09
6	**Early Work**	**453 days**	**Tue 12/11/06**	**Thu 14/07/31**
7	Start Early Engineering, Procurement & Construction	0 days	Tue 12/11/06	Tue 12/11/06
8	Egis Preparatory Works at Shaft and Plant Site	65 days	Mon 13/07/01	Fri 13/09/27
9	AMEC Basic Engineering, Procure Long Lead Items	265 days	Tue 13/02/26	Mon 14/03/03
10	AMEC Construct Temporary Facilities & Camp	196 days	Thu 13/10/31	Thu 14/07/31
11	**Project Construction**	**943 days**	**Thu 13/06/27**	**Mon 17/02/06**
12	**Mine**	**520 days**	**Wed 14/02/05**	**Wed 16/02/03**
13	Freezing, Secant Piles, Sheds	120 days	Wed 14/02/05	Tue 14/07/22
14	Sink both Shafts in parallel	270 days	Wed 14/07/23	Tue 15/08/04
15	Open Up Mine (through 1 shaft)	130 days	Wed 15/08/05	Tue 16/02/02
16	Mining Equipment (through second shaft)	130 days	Wed 15/08/05	Tue 16/02/02
17	Start Mining	0 days	Wed 16/02/03	Wed 16/02/03
18	**Process Plant**	**657 days**	**Fri 14/08/01**	**Mon 17/02/06**
19	Construction	325 days	Fri 14/08/01	Thu 15/10/29
20	Commissioning	192 days	Mon 15/11/09	Tue 16/08/02
21	First Feed through Plant	0 days	Tue 16/08/02	Tue 16/08/02
22	First Product Shipped	0 days	Thu 16/09/01	Thu 16/09/01
23	Plant at Steady State	0 days	Mon 17/02/06	Mon 17/02/06
24	**Infrastructure**	**943 days**	**Thu 13/06/27**	**Mon 17/02/06**
25	Haul Road & Trucks	540 days	Mon 13/10/14	Fri 15/11/06
26	**Marine Facilities**	**701 days**	**Thu 13/12/26**	**Thu 16/09/01**
27	Breakwater	577 days	Wed 14/06/18	Thu 16/09/01
28	Jetty & Loading Facility	671 days	Thu 14/02/06	Thu 16/09/01
29	Barges	701 days	Thu 13/12/26	Thu 16/09/01
30	Tailings Storage Facility	440 days	Mon 14/03/03	Fri 15/11/06
31	Power Supply	440 days	Mon 14/03/03	Fri 15/11/06
32	Gas Supply	500 days	Mon 13/12/09	Fri 15/11/06
33	Water supply & treatment, waste	432 days	Thu 14/03/13	Fri 15/11/06
34	Port & Mine Buildings	740 days	Tue 14/04/08	Mon 17/02/06
35	Camp	943 days	Thu 13/06/27	Mon 17/02/06
36	Construct Sea Water Intake	260 days	Mon 14/11/10	Fri 15/11/06



srk consulting

Sintoukola Potash Project, Republic of Congo

elemental minerals limited

Source: ELM, 2012

Figure 22-1

Summary of Sintoukola Implementation Schedule

23 Interpretation and Conclusions (Item 25)

23.1 Exploration, Geology and Resources

The estimated Mineral Resource for the Kola deposit is based upon reliable diamond drillhole and analytical data. The stratigraphic and structural interpretation of the drilling results is well supported by both down hole geophysical logs and 2D seismic data.

The geological framework indicated from historic exploration data was validated and the continuity of mineralisation was confirmed. The combination of grid drilling (1 km spaced in the centre of the deposit, up to 2 km spaced on the margins) and seismic data allowed for confident interpretation of the subsurface environment and the geometry of the host formation to mineralisation.

Diamond core drilling by ELM has confirmed that the bulk of the potash mineralisation occurs in four seams (FWS, LS, US & HWS) which are then further refined based on potash mineralogy i.e. sylvinite or carnallitite. Additional intervals of potash mineralisation occur beyond the four main seams but have not been modelled due to either limited extent, lack of data or mineralogical composition.

Whilst the mineralized seams occur at varying distances below the base of the anhydrite member, the relative thickness of halite between each seam is remarkably constant. For example the halite between the base of the HWS and the top of US is approximately 60 m thick, the interburden halite between the US and LS is generally 4 m thick and there is typically approximately 45 m of salt between the LS and the FWS.

Based on the higher density of seismic data the “fault model” of 2011 was revised to a “disturbances area model” in which the disturbance is likely to be caused by removal of salt in the evaporite sequence which has led to sagging of and locally disturbance of layering in the overlying strata and consequently to loss of seismic reflector continuity. The potash mineralisation within the footprint of these areas of geological uncertainty was not included in this Mineral Resource estimate and equates to approximately 7%.

The result of this work is a substantial increase in the Mineral Resource estimates and raised confidence in geological model which allowed an upgrade in the classification of the Mineral Resource estimates to Measured, Indicated and Inferred. The Kola deposit is estimated to contain potash resources (combined sylvinite and carnallitite) in Measured Mineral Resources of 559 Mt at an average grade of 16.01% K_2O, Indicated Mineral Resources 758 Mt at an average grade of 15.38% K_2O and Inferred Mineral Resources of 948 Mt at an average grade of 16.20% K_2O.

Within the four main potash-bearing seams discrete zones of sylvinite mineralisation have been identified and these are estimated to comprise Measured Mineral Resources of 264 Mt at an average grade of 21.32% K_2O, Indicated Mineral Resources 309 Mt at an average grade of 20.59% K_2O and Inferred Mineral Resources of 475 Mt at an average grade of 20.39% K_2O.

In CSA’s opinion the work completed by ELM over the past three years has substantially advanced the understanding of the Kola deposit resulting in improved confidence in the development potential of this significant potash project.

23.2 Mineral Reserve Estimate

23.2.1 Hydrogeology

The PFS hydrogeologic program was designed and carried out to characterize the groundwater system overlying the evaporite sequence at the Kola deposit area, and the effectiveness of aquitards lying between the aquifer and the mining horizon. Principal findings include:

- A permeable, vertically hydraulically connected groundwater system comprises all units between the laterite/ferruginous sandstone and the dolomitic limestone;
- The depth to the water table varies between 4 and 26 m within the Kola area of the Sintoukola Project area;
- Salinity increases with depth in the groundwater system, with up to 180 gpl present at Site A and 6 gpl present at Site B;
- Under pre-mining conditions the anhydrite sequence may be considered an aquitard where it is present, based on test results from Sites A and B; and
- Under pre-mining conditions the halite at the top of the evaporite sequence may be considered an aquitard, also based on test results from Sites A and B.

There will always be some inherent risk with mining in highly water-soluble country rock in the presence of a transmissive groundwater aquifer. At the Sintoukola Project, the principal risk is of groundwater infiltration to the mine from the overlying aquifer because of:

- Mining-induced fracturing of the intervening halite and anhydrite sequence (crown pillar); and
- Mining into an existing, natural fracture or other geological anomaly that connects the mine to the aquifer.

23.2.2 Geotechnical

A robust geotechnical study has been undertaken to support the Sintoukola Project. Because of issues related to the uncertainty regarding the anhydrite sequence being a continuous aquitard across the deposit, all pillars have been designed as permanent pillars. This is to ensure that a stiff mine structure is developed to inhibit movement or dislocation of the halite roof beam above the underground workings that may result in the development of any connectivity between the underground workings and the overlying aquifer. The recommended design parameters are therefore considered to be appropriate to mitigate the risk of groundwater infiltration to the mine.

23.2.3 Mineral Reserve Estimate

The Sintoukola Project underground mine design and production schedule presents an estimate of the ore that can be economically and safely extracted from the geologic model, using the site specific geotechnical and hydrogeological parameters, data and analysis noted previously. Only Ore Reserves determined from Measured and Indicated Resources are evaluated in the design.

The room and pillar mine design that is applied to the orebody is a widely used method in the industry and has been used successfully in other similar deposits. CMs are a proven technology and have been operating potash mines worldwide for many years.

The mine production schedule achieves a consistent production rate within a reasonable ramp up period. The schedule also delivers a tonnage and grade suitable to the process and material balance can be smoothed by using the surface stockpile when necessary.

Mineral Reserves are calculated to include 151.7 Mt of ore at an average grade of 20.02% K_2O.

23.3 Haul Road and Road Trains

Ore will be transported from the mine site to the process plant site using a dedicated road fleet of 21 road trains, over a distance of approximately 36 km. The road trains will be composed of articulated trailers and will be able to carry approximately 152 t of ore.

23.4 Metallurgy and Processing

Metallurgical testing was undertaken in 2011 on a composite sample to establish process design criteria and equipment sizing for the Sintoukola Project. Mineralogical analysis showed that the sample was composed of 38% sylvite (24.2% K_2O). Insoluble content was less than 1%, and consisted of anhydrite. The sample received was categorized as coarsely intergrown sylvinite. The low insoluble content and the ease of liberation size of the Sintoukola ore compare favourably with the best examples from Saskatchewan and other producing areas of the world. This allows for high recovery and competitive processing costs. Additional metallurgical testwork was undertaken in 2012 which supported the results from the 2011 testwork.

The results of the metallurgical test programs indicate that the Sintoukola ore can be effectively processed using a conventional potash flowsheet consisting of rougher/ cleaner flotation followed by regrind flotation. Flotation recoveries of up to 94.8% were achieved in the laboratory. The mass balance was developed using METSIM software. A combination of the SRC test results and AMEC's potash experience was used to determine the inputs into the model. This resulted in a process plant recovery of 91% at a feed grade of 25.0% K_2O (39.6% KCl).

The process plant is designed to produce 2 Mtpa of MoP at a grade of 60.5% K_2O. Based on a mine head grade of 19.8% K_2O (31.3% KC), the process simulation resulted in a recovery of 89.5%.

The metallurgical testwork program described was developed and performed under the supervision of AMEC who are satisfied that the testing program and results are at a level that supports a FS level of evaluation and design.

23.5 Marine

Bathymetric conditions support a transshipment solution that involves loading the potash from a jetty into barges, which transfer the product to Handimax and Panamax class vessels anchored approximately 6 nautical miles (11 km) from the coast. These barges are loaded from a 750 m long jetty, which supports the conveyor belt that delivers product from the process plant. The jetty will be protected by 250 m long breakwater structure.

23.6 Solid Residue and Brine Disposal

23.6.1 RSF

Laboratory test results indicate separation of the solids and liquids does occur and therefore decanting of the water should be possible.

The results of the capacity assessment confirm the RSF starter embankment can accommodate approximately 0.24 Mm^3 of insoluble residue and will reach full capacity after approximately 2 years of deposition. The RSF will be constructed in stages and the final RSF will accommodate the required total storage of 1.66 Mm^3 for the LOM. Results of the stability assessment show that the proposed perimeter embankments are stable under all conditions analysed. The results of the flood line assessment conclude that a 1 in 100 year storm event will require a bund to protect the downstream toe of the RSF embankment. Geochemical testing of the residue indicates that the impoundment will require a synthetic liner. A water balance model was performed that indicates there will be annual excess supernatant that will be used for process plant makeup water or pumped to the brine distribution tank to be discharged to the ocean.

23.6.2 Brine Management

The salt brine will be diluted with sea water, resulting in a maximum salt content of 125 gpl following dilution. The diluted sea brine will be pumped via a pipe attached to the jetty, and discharged into the ocean via diffusers installed in the seabed; this is a standard approach for desalination plants. Numerical modelling calculated that the salt concentration will be reduced to within ±2.4 gpl of the background ocean salt concentration within 250 m of the diffusers.

23.7 Employee Facilities

A 10 ha platform and accommodation complex sized to house 950 people has been designed. They will be transported by bus to the various working sites using the access roads and the Haul Road.

23.8 General Infrastructure

Electrical power will be supplied by the ROC national power grid. A 57 km long 220 kV transmission line will be constructed to supply power to the process plant sites. From the HV process plant site substation, a 35 km long 220 kV transmission line will be constructed to supply power to the mine site. The demand for electrical power is estimated at 24 MVA at the mine site and 32 MVA at the coastal sites.

AMEC estimated 20 Mm^3 of natural gas per annum were needed for the product drying process. EGIS sized a 150 mm nominal diameter pipeline that is 81 km in length. This pipeline will run from the gas treatment plant at Cote-Mateve, located 10 km south of Pointe Noire, to the process plant.

Fresh water will be supplied by wells at both the mine and process plant sites. Seawater will be supplied via a screened seawater intake installed 400 m from the shoreline.

The service road network will make use of a ten kilometer portion of the haul road and three other sections of road will be required prior to the construction phase, the existing tracks will be repaired seasonally (mainly drainage works and few earthworks) to allow for year round traffic. Once production starts, these roads will be upgraded along their entire length to allow for improved traffic conditions for the operational phase.

23.9 Environmental

A comprehensive SEIA that meets national and international requirements is being undertaken and the ToR for the SEIA has been approved by the Congolese Government. The SEIA is at an advanced stage, with social and biophysical baseline field studies completed and associated reports currently being drafted for review during September 2012. A comprehensive understanding of baseline conditions has been developed through the field studies and analysis of data; this will form the basis for the subsequent impact assessment and development of mitigation measures.

As the project will be located in a sensitive biophysical and social environment, opportunities to minimize negative impacts have been identified and integrated with project design throughout the PFS phase. At the current stage in the SEIA process (pre-impact assessment) it appears that the majority of potential environmental impacts identified can be readily managed through the implementation of standard environmental management plans. Several material negative risks, however, have been identified, including some related to social and community issues, which will warrant specific management measures in order to avoid project delays and reputational damage. The material risks relate to delays in the environmental permitting process; meeting the expectations of conservation NGOs and other stakeholders with respect to protecting biodiversity in the project area; delays in land acquisition, resettlement and compensation (all of which are government-led processes); the need to build trust and constructive relationships with key stakeholders (in particular local communities and indigenous peoples); effectively engaging and partnering with government authorities to manage the influx of people to the project area and effectively managing road safety on the upgraded service road and pedestrian access to the dedicated haul road.

ELM is aware of these material risks and is developing a range of approaches to address them and manage the potential impacts on the project, including enhanced stakeholder engagement and consultation. While the material negative risks are significant, with the implementation of appropriate mitigation measures and proactive management by ELM, they should not represent fatal flaws to the project.

23.10 Marketing and Economics

Potassium (K) is one of the three fertilizer nutrients essential for agricultural output and demand is therefore driven by projected global food consumption. ELM utilized market research from CRU and Fertecon to develop its potash marketing strategy. Both companies foresee a strong growth of both demand and supply in potash in the next decade.

CRU considered South Africa, India, SE Asia and Brazil as favourable markets based on forecasted growth in demand and competitive shipping rates from the ROC. For the purpose of the PFS, Brazil is considered to be the target market for product sales from the Sintoukola Project. Projected growth in the Brazilian potash demand is sufficient to absorb all the Sintoukola production. The preferred product in the Brazilian market is granular material, which will form the bulk of the production from the Sintoukola Project, at a targeted MoP grade of 60.5% K_2O. Fertecon provided CFR price projections for the Brazil market through 2020 where after the price was kept constant.

The economic analysis results indicate a net present value (NPV) for the project at a 10% discount rate of US$2,971M with an internal rate of return (IRR) of 29.3% (after tax). The pay back period from Q3, 2013 is 6 years.

24 Risks and Opportunities

24.1 Exploration, Geology and Resources

The exploration completed at the Kola deposit by ELM has greatly improved the understanding of the geological model; it has identified new areas of potash mineralisation; and has resulted in significant expansion to the estimated Mineral Resources. Opportunities remain to increase Mineral Resources via additional exploration as the deposit is open in several directions and significant Inferred Mineral Resources exist which may be able to be upgraded by additional work.

A fundamental geological risk relates to the dependence on 2D seismic data to underpin the geological interpretation in areas between drillholes (which are spaced up to 2km apart). Whilst this is common practice for the assessment of potash deposits (and even wider-spaced drilling grids are sometimes relied upon) it is accepted that differences between the current geological model and actual conditions are possible (and in fact likely). The materiality of any variances relate to the extent of the differences and to the potential for significant impact to the mine plan. Specific examples of geological risks which could affect the mine plan include: discrepancy in the locations of areas of actual or potential geological instability, identification of structures that have potential for water ingress, and unforeseen and abrupt changes to seam geometry or mineralogy.

During the exploration and interpretation work completed by ELM to date every care has been taken to ensure that any such issues are flagged and investigated as far as the data will allow. There has been close interaction between the exploration geologists, hydrogeologists, geotechnical and mining engineers to identify geological risks. Areas of geological uncertainty have been excluded from the Mineral Resources and will be avoided in the mine plan. Work programs have been recommended to improve the geological model by conducting additional drilling, sampling and analyses, acquiring further (2D and 3D) seismic data and enhancing the processing of the seismic to further develop the geological model. Mining, hydrogeological and geotechnical risks are discussed further below.

24.2 Mineral Reserve Estimate

24.2.1 Hydrogeology

Based on findings of the hydrogeological investigation, the principal risk to the mine is the potential for water to enter into the mine workings from the permeable overburden groundwater system through fractures in the upper part of the halite and anhydrite sequence induced by subsidence as a result of mining activities. Therefore, the underground mine should avoid the disturbance areas and should be designed with extraction ratios that do not result in any deformation (fracturing) to the upper part of halite and anhydrite sequence. Geotechnical studies should determine whether mining-induced stress could lead to fracturing of the halite above the mine, thus decreasing its effectiveness as a seal.

Currently, two mine shafts are planned (potential locations shown on Figure 14-4). Additional hydrogeological characterization is recommended in order to support development of production and service shafts. The aim of this work is to collect a detailed vertical profile of hydraulic conductivity, groundwater velocity, water chemistry, and temperature with depth.

24.2.2 Geotechnical

SRK has identified the following geotechnical risks associated with the Sintoukola Project. The mitigation of these risks will inform the requirements for the FS.

The overburden characterization is important as it acts as a dead load imparting stresses onto the mine elements. While the strength of these materials is not considered to be critical, it may have an impact on estimating the magnitude of subsidence at surface. Improved knowledge of the geotechnical characteristics of these materials will reduce this uncertainty.

Uncertainty exists regarding the composition, thickness and spatial variability of the anhydrite sequence which may impact its ability to provide an effective aquitard over some areas of the deposit.

An incorrect selection of evaporate strength parameters may change the stability results modeled and corresponding mine design parameters. Additional drilling, sampling and testing will increase the confidence of the selected strength parameters.

Some uncertainty will remain regarding interpretation of the geologic model. Possible deviations in interpretation of geophysical data may impact adversely on geotechnical mine stability conditions.

The evaporite structures identified in the drillhole logs appear to be sedimentary clay layers in roof and floor of deposit. The impact will be limited due to relatively narrow panel rooms and a slightly greater impact in wide permanent rooms. Systematic roof supports will be required.

In areas where the roof beam thins out, misinterpretation of the roof beam thickness may result in an increase in the risk of roof beam failure and breakthrough into the overlying water bearing materials

Evaporite creep will affect the performance of mine elements, resulting in long term deformation. This may affect the magnitude of ground movement, at surface, activate structures and may impact on overlying aquifers.

Opportunities exist to fine tune pillar dimensions as a result of more rigorous 3D creep modelling which may result in improved panel extraction ratios.

24.2.3 Mining and Mineral Reserves

SRK has identified the following Sintoukola Project opportunities:

- Upgrading of additional mineralization currently classified as inferred within the current mining area due to low drill density. Additional drilling should be planned to confirm this ore as well as additional seismic coverage;
- Underground exploration will improve resource definition without the penalty of resource sterilization that is associated with surface drilling;
- Grade variability in the CSA model appears globally smoothed. It is expected once underground a high variability may be seen over short distances that cannot be detected from surface. This would influence thickness, grade, and mineral type (sylvinite vs carnallitite) and could be a risk or an opportunity;
- Currently the mining schedule is based on a CM cutting at a rate of 2,600 tpd. There is an opportunity to increase this assumption once operating experience is gained; and

- Ventilation requirements can be lessened once more detailed mine design and scheduling is undertaken to minimize distances between active mining areas. This will potentially reduce ventilation operating costs as well as cooling costs; and
- Potential inclusion of other mining seams currently classified as indicated (FWS/HWS).

SRK has identified the following Sintoukola Project risks:

- Grade control will be challenging in the early stages of the mine's life as systems are implemented and personnel are learning about the orebody. To control this risk it is recommended that permanent geological staff are brought in early and that detailed grade control systems are developed prior to panel mining operations starting;
- Grade variability in the CSA model appears globally smoothed. It is expected once underground a high variability may be seen over short distances that cannot be detected from surface. This would influence thickness, grade, and mineral type (sylvinite vs carnallitite) and could be a risk or an opportunity. To control this risk underground exploration drilling will take place to define mineralization characteristics in the immediate mining areas;
- The presence of water in overlying units that are hydraulically connected with high conductivity have potential to flood the mine. Maximizing distance from mining to the base of anhydrite will minimize this risk by providing a thicker salt beam between the water and mining. Underground drilling ahead of mining will also minimize this risk. Additionally, if a catastrophic water event were to occur, an emergency mine dewatering plan should be in place;
- Disturbance zone have been identified in the closely spaced seismic areas. There is risk in the interpretation of these areas and potential of smaller disturbances between the lines. Areas classified as indicated need additional delineation of disturbances zones either from seismic or underground drilling information;
- There is a geotechnical risk in the stability of the pillars and back and a constant ground control program should be in place during operations to minimize this risk; and
- Shaley interbeds could cause local ground control problems.

24.3 Haul Road and Road Trains

A fleet of road trains will haul the ore from the mine to the process plant, which poses several safety risks. While safety precautions will be taken, this activity is subject to accidents particularly during rainfall events and at night. To mitigate this risk, the design of the haul road accounted for wider roads, larger curve radius, shallower slopes and security devices. Operator training will be required including regular review. Spare road train, parts and particular mobile equipment (mobile crane, tow truck, loader, etc.) have been included to minimize delays due to downtime.

There are risks associated with the integrity of the haul road, such as shallow instability in the cut slopes and embankment. To mitigate this risk, the design included an allowance for slope protection using Vetiver grass.

The actual estimate for hauling operation is based on an integrated owner operation. There is an opportunity to reduce the initial investment if contract hauling was considered.

24.4 Metallurgy and Processing

Subsequent to the completion of the metallurgical test program, it was determined that some of the material included in the sample is from seams that are currently not part of the mine plan. There is a risk that the metallurgical performance of the material in the mine plan would not match the results achieved in the test program. However, notwithstanding that the material from outside of the mining plan has slightly more insoluble content than the material included in the mining plan, the mineralogical composition and granulometry of the ore and its impurities are almost identical. Therefore the metallurgical performance of the material included in the mine plan should be comparable, if not better than that achieved for the metallurgical sample.

As many of the seams from the Sintoukola Project have a similar mineralogical composition and insoluble content, these seams have a relatively limited potential to have an impact on metallurgical performance. However, it must be noted that the seams from the Sintoukola Project have a much lower insoluble content when compared to those in the other major potash basins in the world. Therefore, there is the opportunity that the process plant design will efficiently process material from multiple seams.

There is an opportunity to filter the insolubles and dry stack them as opposed to discharging a slurry to the RSF, which will result in a smaller footprint for the RSF. Testwork should be conducted to determine the feasibility of filtering the insolubles.

There is an opportunity to reduce the height of the process plant building resulting in lower capital costs. AMEC is conducting a study to investigate this and the findings will be incorporated into the FS.

There is an opportunity to optimize the design of the process plant foundations to reduce the capital cost. Once the geotechnical report is available for the process plant area, AMEC will conduct an investigation to optimize foundation design. This will be incorporated into the FS.

24.5 Marine

There is a risk that the breakwater length may increase, if the navigational and ship-mooring simulations do not satisfy the navigational safety requirements. These simulations will be performed, and assumptions confirmed, as part of the FS.

There is an opportunity to reduce the length and the number of piles of the jetty, and this will be addressed in the FS as the results of the off-shore geotechnical investigation are made available.

24.6 Solid Residue and Brine Disposal

24.6.1 RSF

SRK assumed that the insolubles will settle to an in situ dry density of 0.9 t/m^3 under controlled deposition and form a beach angle of approximately1%. Settlement to a lower density is not expected, but has been evaluated. If the achieved density was as low as 0.6 t/m^3, an additional approximate 3 m embankment raise would be required. This additional raise would not impact the overall layout and operating philosophy of the RSF and the cost for this raise would absorbed by the contingency allowance. Although the final placed density remains uncertain, there are no significant risks that will have an impact on the design.

If the proposed deposition method for the insolubles does not result in beaching, alternative deposition methods can be investigated in the next design phase, such as deposition from the head of the valley. This will promote thin layer deposition, allow more time for water / solids separation and potentially improved densities.

Foundation conditions and construction material properties have been assumed based on limited testing done in proximity to the proposed RSF site. Geotechnical testing could potentially produce improved shear strength parameters and corresponding steeper construction slopes. Groundwater conditions at the proposed RSF site foundations are unknown.

Hydrological assessment of the flood lines has been based on a similar site. Therefore, actual storm vents may differ as some site-specific factors may not have been considered when comparing the sites.

A small insoluble sample was attained during the PFS and the geotechnical testing required to fully characterise the residue could not be conducted. Additional geotechnical testing could yield improved geotechnical parameters.

Dewatering of the insoluble could be considered, as this could increase brine recovery, potentially result in improved settling rates, an increased settled density and a corresponding potential decrease in RSF size.

24.6.2 Brine Management

If a suitable large groundwater source is identified and characterized as part of the FS hydrogeology investigation, the salt brine could be diluted using freshwater, reducing the volume of brine discharged into the sea.

24.7 Employee Facilities

An opportunity exists to further reduce the size of the employee facilities if the SEIA shows that unskilled labor do not need to be accommodated in the employee facilities can live in the local villages.

24.8 General Infrastructure

The reliability of the national power grid poses a potential risk to the Sintoukola Project. While the current power distribution reliability is low, the Société Nationale d'Electricité (SNE) has recently launched several projects to improve its reliability. In addition, the CEC power plant which will supply power is targeting to support the industrial development of the Pointe Noire area. During various meetings with the CEC, several solutions were discussed, such as a dedicated network to the industrial customers.

An opportunity exists to reduce the length of the gas pipeline if a closer source of natural gas is available. A trade off study during the FS will address this matter comparing lengths and gas characteristics.

An opportunity exists to change the alignment of the access road (ET2) from the employee facilities to another location closer to the process plant and reduce the road length by approximately 2 km.

24.9 Environmental

Refer to Section 18.5.

24.10 Marketing and Economics

The risk from a marketing perspective is that SPSA is unable to sell its product at projected market prices because of depressed future demand, over supply which may drive down the price of potash, and / or SPSA being unable to penetrate the Brazilian market.

The project NPV is most sensitive to changes in potash price, followed by changes to discount rate, tax rate, capital cost and operating costs. Additional sensitivities to power costs, labor costs, exchange rates and fuel costs were performed and the economic results of the project were only marginally affected by these changes.

ELM has held discussions with infrastructure funds, technical partners and operators with a view to finding a partner to build own and operate some or all of the defined infrastructure components. To date, a number of formal non-binding expressions of interest have been received from multiple parties which ELM believes are technically and financially credible. Two of the parties are high profile and credible global logistic operators who are have expressed an interest in funding, managing and operating the infrastructure project. If this initiative proves successful, the infrastructure outsourcing would result in significant capital savings, but with a commensurate increase in operating costs, which ELM deems manageable giving the low operating cost of the Sintoukola project.

24.11 Risk Assessment and Mitigation

A project wide risk assessment was performed for the Sintoukola Project based on an analysis of the risks identified during the June, 2012 Sintoukola Project Plenary session held in Paris that was attended by all major project contributors and led by ELM. Eighty four risks were identified during the session and mitigation measures were identified. A detailed summary of all identified risks and their mitigations can be found in Section XI.

24.11.1 Assessment Matrix

Risks were categorized as either construction related or operational. Construction risks deal with the feasibility study, early works, funding, project implementation and ramp up, while operational risks deal with ongoing activities once the project has achieved nameplate capacity. Some risks fall into both categories, in which case they are not duplicated.

Once identified, each risk was rated according to the likelihood and consequence of occurrence. In terms of likelihood, risks were rated as either almost certain, likely, moderate, unlikely or rare. Consequence ratings included catastrophic, major, significant, moderate or minor. By defining the likelihood and the consequence, the severity was then defined.

Both inherent risks, which are the risks before any mitigation, and residual risks, which are the risks after mitigation, were ranked and discussed for each category. Figure 24-1 summarises the quantum of inherent and residual risks with different severities. It can be noted that the mitigation actions effectively reduce the risk severity.

Only risks ranked "high" or "extreme" after mitigation are considered in detail, while the reader is referred to Volume XI for a full list of the identified risks.

24.11.2 Construction Risks

In total, 38 inherent risks were identified in the construction category. Prior to mitigation, one risk was classified as low, 17 as moderate and 20 were regarded as high risks. After mitigation, three risks were still considered to be high and are discussed below in terms of impact, drivers or triggers, timing and the proposed controls:

- Logistics: delays in delivering materials to site will delay project construction. This is exacerbated by the need to use barges to cross the Kouilou River while the bridge is under repair. The control for this risk is the implementation of expediting systems. This control reduces the likelihood of this risk from likely to moderate, as road and sea freight will be outsourced and therefore many challenges that may arise will remain beyond the project's control;
- Import delays: delays in clearing materials and equipment through customs will result in construction delays and failure to reach production targets. The control is to send equipment specifications, waybills, etc. to customs officials in advance and to ensure that the Owner's Team is sufficiently staffed to expedite import procedures. Similar to the logistics risk, this control reduces the likelihood of this risk from likely to moderate, as customs processes and procedures are beyond the project's control; and

Influx management: an uncontrolled influx of work seekers would result in social ills and environmental degradation. A lack of adequate government control would drive this risk, which may be realised during both the construction and operational phases. Government engagement and adequate project design and stakeholder communication may mitigate the impact of this risk, while the likelihood of it occurring remains high.

24.11.3 Operational Risks

In total, 46 operational risks were identified. Prior to mitigation, 21 risks were classified as moderate and 25 were regarded as high risks. After mitigation, eight high risks remain. The high residual risks are discussed below, in terms of impact, drivers or triggers, timing and the proposed controls:

- Light vehicle accidents on the way to the mine and Pointe Noire: accidents would result in injuries and possibly fatalities. Control measures include driver training, vehicle maintenance and public awareness campaigns, as well as strict enforcement of rules and procedures. These control measures are considered to reduce the likelihood from likely to moderate as light vehicle accidents are often the result of a human error in judgment and this is beyond the project's control;
- Accident on HV road: people crossing the road, or trucks colliding could result in serious injuries and even fatalities. Even though the risk likelihood will be reduced as pedestrian traffic will be controlled by fences and ongoing awareness campaigns and drivers will be trained, the frequency of the truck traffic ensures that this risk remains high;
- Night driving on haul road: accident risk is increased at night, which could result in injuries or fatalities. Poor lighting, pedestrian traffic and unsafe driving may trigger this risk during both the construction and operation phases. The control is the implementation of strict traffic

safety controls, adequate lighting and access control points on the haul road, wherever possible. These measures reduce the likelihood of the risk;

- Fiscal stability of the ROC: changes in royalties or taxes could result in fiscal instability. Political and economic developments in the ROC could drive this risk during both the construction and operation phases. The control is to engage with the government, however, the effectiveness of the control is relatively ineffective, as the ROC's political and economic climate is beyond the project's control;
- Reliability of grid power supply: As Sintoukola's power is drawn from the grid, lack of maintenance at the power plant, action by the power supplier to cut supply or damage to the power line could all lead to power interruptions. This leads to a loss of production and has a safety risk. The provision of backup generators reduces the consequence and an ongoing liaison with the power company reduces the likelihood of this risk, while the consequences of an interruption remain high;
- Salt back thickness: mine design parameters have been developed to ensure the integrity of the salt back based on modeled salt back thickness. Uncertainty in the actual thickness may cause these parameters to be exceeded. The saltback could break if geotechnical stability criteria are exceeded. The control is an increase in the level of detail in the geological model through underground drilling and conservative geotechnical design criteria. Provision of emergency pumping infrastructure also facilitates inrush management. Although these controls are effective, the potential consequence of this risk is catastrophic. The residual risk therefore remains high;
- Uncertainty of hydrogeological connectivity, water inrush and disturbance zones: any of these risks may cause a catastrophic inrush of water. Control measures for this risk include the use of tomography, ground penetrating radar (GPR) and long hole-drilling to ensure accurate geological and hydrogeological models. Provision of emergency pumping infrastructure also facilitates inrush management. This control reduces the likelihood of the event occurring. Risk occurrence, however, may result in the loss of the mine and therefore the residual risk remains high; and
- Risk of a CM mining into historic oil exploration drillholes: if a CM intersects an historic oil drillhole, this could result in a catastrophic water inrush. The control is accurate location and avoidance of the oil exploration drillholes. This control reduces the likelihood of the event occurring. Risk realization, however, may result in the loss of the mine and therefore the risk remains high.

24.11.4 Conclusion

The risk assessment performed for the Sintoukola Project was based on an analysis of the risks identified in the Sintoukola Project plenary session. All of the identified risks were subsequently rated according to the likelihood and consequence of risk occurrence. Risks were divided into two categories, either construction or operations and were examined separately.

After mitigation, three high construction risks remained that relate to logistics, imports and influx management and will be mitigated by good planning and expediting and ongoing liaison with the relevant parties. Operational risks were also evaluated and after mitigation, eight high risks remain. Three of these risks relate to possible road accidents, while another three relate to events that could lead to mine flooding. The other two risks relate to the stability of fiscal policy and the reliability of grid power. Mitigation measures for the road accidents include driver and community training and awareness campaigns and compliance with traffic rules. The risk of mine flooding is mitigated by improving geological understanding through underground drilling and good mapping and by applying a conservative approach to mine planning. Provision of emergency pumping infrastructure also facilitates inrush management.

25 Recommendations (Item 26)

25.1 Exploration, Geology and Resources

During the Phase III exploration program it is recommended that ELM complete the following:

- Improve the processing of the seismic data to enhance the imaging of the reflectors and areas of discontinuity (potential geological disturbance areas), to allow refinement of the geological interpretation generally. Specifically, this requires generation of improved velocity models to develop more reliable and integrated depth-migrated models;
- Acquire and process new 2D reflection seismic data across the existing grid to reduce the spacing to 150 m between dip-lines as well as additional tie lines. Some lines should be extended outside the current grid to allow coverage of proposed drillholes;
- Complete a trial 3D seismic survey over an approximate 1 to 2 km^2 area of the Kola deposit to determine the variance between the geological model from 2D and 3D seismic data. The trial 3D survey could make use of one of the naturally occurring savannah features to reduce land clearing;
- Review the economic potential of areas of Inferred Mineral Resources to define parameters and determine priorities for resource definition / expansion drilling;
- Conduct resource definition and expansion drilling in the areas of high-priority e.g. the HWS sylvinite mineralisation and other areas identified as having greater likelihood to have a significant positive influence on the mine plan. The drilling should be completed on seismic lines to allow tie-in to the existing geological model;
- Continue exploration on regional targets e.g. Dougou;
- Collect additional samples for density, petrographic, mineralogical and geochemical test work to better define the mineralisation particularly in those areas where upgrading of resource classification is sought; and
- Undertake a program of umpire analyses using classical analytical techniques to compare to the assay results returned from routine analysis using instrumental (ICP) techniques.

25.2 Mineral Reserve Estimate

25.2.1 Hydrogeology

Based on the listed above hydrogeological observations and conclusions, the mine should be designed to avoid:

- Areas of geological disturbance;
- Areas around previously-drilled drillhole; and
- Areas where the halite is projected to be less than the minimum thickness required in the crown pillar after accounting for potential uncertainty in the geologic model.

25.2.2 Geotechnical

Based on the assessment of geotechnical risks presented in Section 24.2.2 further geotechnical work will be targeted to mitigate and reduce these risks. Some of this will be carried out as part of the proposed Phase 3 geological investigation defined in Section 25.1 above.

As the PFS has developed other potential potash seams above and below the originally targeted US and LS have been identified. It is intended to bring some of these into reserve during the feasibility study. In order to assess the geotechnical impact of concurrent multiple seam extraction, the following will be required:

- Investigation of the geotechnical characteristics of the additional seams (FWS and HWS) and interburden units by sampling and testing core, if available, or by correlating existing test results with the new materials;
- Carry out additional 2D finite element sensitivity modelling of multiple seams to determine appropriate pillar dimensions, halite interburden thickness and halite roof beam thickness;
- Carry out creep modelling of pillars using FLAC3D to determine time dependent deformation of mine elements;
- Use the time dependent behaviour of the pillars to inform a more detailed assessment of surface subsidence; and
- Using the results of both the 2D and 3D numerical modelling fine tune and optimise pillar dimensions to maximise extraction ratio whilst maintaining long term stability of the mine.

25.2.3 Mining

The following recommendations are made regarding future work:

- Proceed with a more detailed design phase (FS into detailed design), including:
  - Shaft Design,
  - Panel layout planning,
  - Ventilation and cooling design,
  - Conveyor design and optimization, and
  - Ground support and pillar design.
- Shaft sinking contractor should be brought is to minimize timeframe to sinking;
- Prior to shaft sinking, pilot drillhole should be drilled recovering core for the length of the hole to ensure geology is suitable and as expected and confirm geotechnical design parameters;
- As the FS nears completion, long lead time capital equipment should be ordered on a priority basis; and
- Key underground technical and management staff should be recruited and hired to facilitate the mine design and detailed design phase.

25.3 Haul Road and Road Train

Specific recommendations for the haul road and road trains include the following tasks:

- Consult with local authorities, stakeholders and local land owners to confirm that the road meets expectations;
- Perform testing to support the feasibility of treating the in situ sand with cement to determine if the soil performance can be improved at a cost savings; and
- Consult with potential contract hauling providers to optimize haul road configurations based on operational experience in the ROC.

25.4 Metallurgy and Processing

It is recommended that the following metallurgical testwork be conducted on material from the Sintoukola Project:

- Complete phase 2 (XRD analysis and insoluble removal) and phase 3 (rougher and regrind flotation) of the proposed test program for the LSS, FWS and HWS;
- Conduct a test program on a composite sample consisting of material from the US, LS, HWS and FWS once the mine plan has been developed in further detail. The program should include liberation testing, XRD analysis and insoluble removal and rougher and regrind flotation testing; and
- Conduct a test program to investigate the feasibility of filtering the insolubles.

25.5 Marine

Specific recommendations for marine facilities include the following:

- Improve and refine the design of the jetty foundations through near shore geotechnical investigations. Due to the ocean conditions, these investigations need to be carried out in the first quarter of 2013;
- Re-run the wave transfer modelling study with the additional reef bathymetry survey data;
- Conduct navigational and ship-mooring simulations to verify if navigational safety requirements are achieved with the proposed attached breakwater layout; and
- Carry out shoreline evolution modelling to confirm the influence of the breakwater on coastal sediment dynamics.

25.6 Solid Residue and Brine Disposal

25.6.1 RSF

SRK has the following recommendations for the FS:

- A larger sample of the insoluble residue should be obtained so that particle size distribution (PSD), consolidated density and shear strength testing be carried out to estimate an in-place dry density of the residue material and beach angle estimation more accurately;
- Dewatering of the slimes should be investigated;
- A site investigation should be carried out to confirm the foundation conditions assumed in the PFS. Suitable sources of construction material need to be identified and tested to derive parameters for the stability assessment;
- Seismic register of earthquakes in the area should be obtained to support deformation analysis and confirm if steeper embankment slopes will be required;
- Intensity Frequency Duration (IFD) charts or temporal distributions for the project area for the 1 in 100year storm event be obtained from the nearest weather station. Alternatively, a calibrated method of flow analysis of the valley should be carried out on site; and
- Design criteria on erosion control should be obtained.

25.6.2 Brine Management

Specific recommendations for the brine management include the following:

- Re-run the brine dissolution and dilution model; and
- Develop a 3D model of the brine dispersion, including all elements contained in the salt brine to confirm the compliance with IFC Environmental and Health and Safety effluent discharge guidelines.

25.7 Employee Facilities

Specific recommendations for employee facilities include the following:

- Review the employee facilities location to reduce the road access length;
- Integrate synergies with the construction camp; and
- Review the number of people to be accommodated by categories and to check the possibility of reducing the number of local people to be accommodated on site and of providing bussing facilities.

25.8 General Infrastructure

Specific recommendations for infrastructure include the following:

- Investigate if closer sources of natural gas are available;
- Perform a hydrogeological investigation at the coastal site to confirm the potential of the ground water sources. If the hydrological survey does not provide satisfactory results, further investigations would need to be performed to determine the salinity levels of the surface raw water sources; and
- Engage discussions with national and local authorities to obtain agreement on the level of road improvements expected. If higher standards are required, changes to the width and alignments could impact the road interaction with certain villages.

25.9 Environmental

The SEIA should be completed as originally programmed, with submission of the national SEIA report in December 2012 and the international report in February 2013. The estimated cost for completion of the SEIA is US$1.9M, comprising US$0.8M for completion of the baseline studies and reporting, US$0.8 for the impact assessment, development of mitigation measures and preparation of the national and international SEIA reports and US$0.3M for post-submission stakeholder engagement during the government-led review of the national SEIA report.

The impact assessment process and national and international SEIA reports should be updated on completion of the FS; the cost of updating will depend on the nature and extent of any significant changes to the project description.

Effective stakeholder engagement must remain a core element of SPSA's mitigation and management plans throughout the FS and the subsequent project lifecycle (from construction to closure). SPSA must also continue to develop and implement management measures to address the material negative environmental and social risks (identified in Section 18.5).

25.10 Marketing and Economics

It is recommended that a detailed product marketing strategy be developed by ELM to evaluate the optimal method to penetrate the various product markets. Sales and marketing costs will also be evaluated in more detail.

25.11 Recommendations and Future Work Program

Based on the field programs, testwork, design work and economic results, SRK recommends that the Sintoukola Project advances to the FS level of design. Recommended work programs have been presented in the previous sections and the associated cost estimates are provided in Table 25.11.1.

Table 25.11.1: Cost Estimates for Recommended Programs

Item	Costs(US$'000)
Project Management	2,384.6
Plenary Meeting	70.4
DFS report writing	291.2
Hydrogeology	612.6
Geotechnical appraisal	611.4
Mining	678.8
Metallurgy/Processing	4,516.6
Infrastructure	6,108.4
Tailings Storage Facility	265.0
Environment and Community	2,721.9
Capital Cost Estimate	218.6
Operating Cost Estimate	11.4
Economic (Financial) Evaluation	89.0
Reimbursables	106.8
Total Study Program	18,686.8

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27 Glossary

27.1 Mineral Resources

The Mineral Resources and Mineral Reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 27, 2010). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Mineral Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

27.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility

Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

27.3 Abbreviations

The following abbreviations may be used in this report.

Table 26.4.1: Abbreviations

Abbreviation	Unit or Term
2D	two-dimensional
3D	three-dimensional
AAE	Alan Auld Engineering
AMEC	AMEC Americas Limited
amsl	above mean sea level
ASX	Australian Stock Exchange
API	American Petroleum Institute
°C	degrees Centigrade
CAPEX	capital expenditures
CDNP	Conkouati-Douli National Park
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CoG	cut-off grade
CM	continuous miner
cm	centimeter
cm^3	cubic centimeter
CoG	Cut-off grade
CS	check samples
CSA	CSA Global Pty Ltd
CRU	CRU International Ltd
°	degree (degrees)
DEM	digital elevation model
DGPS	differential global positioning system
dia.	Diameter
EBITDA	Earnings before tax depreciation amortization
EDS	energy dispersive spectroscopy
EGIS	EGIS International
EIA	Environmental Impact Assessment
ELM	Elemental Minerals Ltd
EMP	Environmental Management Plan
FS	Feasibility Study
FWS	Footwall Seam
FWSS	Footwall Sylvinite Seam
g	gram
g/cm^3	grams per cubic centimeter
gpl	gram per liter
GPS	Gloval positioning system
g/t	grams per tonne
ha	hectares
HAC	high angle conveyor
HDPE	Height Density Polyethylene
hp	horsepower

Abbreviation	Unit or Term
HWS	Hangingwall Seam
HWSS	Hangingwall Sylvinite Seam
IBH	interburden halite
ID2	inverse-distance squared
ID3	inverse-distance cubed
IFC	International Finance Corporation
IRR	Internal Rate of Return
JORC	Australian Joint Ore Reserves Committee
kA	kiloamperes
kg	kilograms
kg/m^3	kilograms per cubic meter
km	kilometer
km^2	square kilometer
K	Potassium
K^+	potassium ion
K_2O	potassium oxide
KCl	potassium chloride
kt	thousand tonnes
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LIDAR	Light detection and ranging
LoM	Life-of-Mine
LS	Lower Seam
LSC	Lower Seam Carnallitite
LSS	Lower Seam Sylvinite
m	meter
m/s	meters per second
m^2	square meter
m^3	cubic meter
$m^{3/}s$	cubic meters per second
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
Mg	magnesium
Mg^{+2}	magnesium ion
$MgCl_2$	magnesium chloride
mgpl	milligrams per liter
ml	
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
MoP	muriate of potash
MPC	mobile power centers
Mt	million tonnes
Mtpa	Million tonnes per annum
mpm	meters per minute
MTW	measured true width
MW	million watts
m.y.	million years
Na^+	sodium ion
NaCl	sodium chloride

Abbreviation	Unit or Term
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
NPV	Net Present Value
OSC	Ontario Securities Commission
%	percent
PFS	Prefeasibility Study
PLC	Programmable Logic Controller
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
PVC	polyvinyl chloride
PWC	Pricewaterhouse Coopers
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
RC	rotary circulation drilling
ROC	Republic of Congo
RoM	Run-of-Mine
rpm	revolutions per minute
RQD	Rock Quality Description
RSF	Residue Storage Facility
SEC	U.S. Securities & Exchange Commission
sec	second
SEIA	Social and Environmental Impact Assessment
SEM	Scanning Electron Microscopy
SF	seam floor
SG	specific gravity
SPSA	Sintoukola Potash, S.A.
SPT	standard penetration testing
SR	seam roof
SRC	Saskatchewan Research Council
ST	seam thickness
t	tonne (metric ton) (2,204.6 pounds)
TEM	Technical Economic Model
tph	tonnes per hour
tpd	tonnes per day
t/m^3	tonnes per cubic meter
TSP	total suspended particulates
TSX	Toronto Stock Exchange
TWTT	two way travel time
µm	micron or microns
US	Upper Seam
US$	U.S. dollars
USC	Upper Seam Carnallitite
USS	Upper Seam Sylvinite
UTM	Universal Transverse Mercator
V	volts
VFD	variable frequency drive
VSP	vertical seismic profiling
W	watt
XRD	x-ray diffraction
y	year

Appendices

Appendix A: Certificates of Authors

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CONSENT OF QUALIFIED PERSON

I, Andrew Scogings, BSc, PhD, MAIG, MAusIMM do hereby certify that:

1. I am an Associate Consultant with:
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   West Perth, Western Australia, 6005
2. This certificate applies to the technical report titled NI 43-101 Technical Report, Sintoukola Potash Project, Republic of Congo, with an Effective Date of September 17, 2012 (the "Technical Report").
3. I am a professional geologist having graduated with a PhD from the University of Durban-Westville, South Africa in 1990. I am a Member of the Australian Institute of Geoscientists and a Member of the Australasian Institute of Mining and Metallurgy. I have worked as a Geologist for a total of 22 years since my graduation from university. My relevant experience includes more than 15 years completing resource models of stratabound mineral deposits.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Sintoukola property.
6. I am responsible for the preparation of Mineral Resource estimate in the Executive Summary and Section 12of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is as a consultant responsible for mineral resource modelling.
9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of September 17, 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

"Signed"

Dr Andrew Scogings
CSA Global Pty Ltd

egis International

CERTIFICATE OF AUTHOR

I, J. HECTOR, Doctor in Geology, do hereby certify that:

1. I am Senior Geologist of:

   Egis International
   Place des Frères Montgolfier
   78286 Guyancourt Cedex

2. I graduated with a doctorate in geology from Montpellier University in 1978 (Ph.D).

3. I am a member of the European Federation of Geologists (EFG), International Association of Engineering Geology (IAEG), and International Society of Rocks Mechanic (ISRM).

4. I have worked as a Geologist for a total of 35 years since my graduation from University. My relevant experience includes many expertises for Infrastructure (Roads, viaducts, foundations, retaining walls, civil works, buildings and tunnels) in Kenya, Cyprus, Malaysia, Iraq, FYROM, Greece, Canada, South Africa, Portugal, Romania, Kosovo, Albania, Croatia, Marocco, Italy and Congo.

5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.

6. I am responsible for the preparation of marine, brine management and general infrastructure in the Executive Summary, Sections 3.2, 3.5, 16.1, 16.2, 16.3, 16.4, 16.5, 16.6 (excluding 16.6.4), 16.7.2, 19.1.2, 19.1.4, 19.1.5.2, 19.1.6, 19.1.7, 19.2.2, 19.2.4, 19.2.5.2, 19.2.6, 19.2.7, 23.3, 23.5, 23.6.2, 23.7, 23.8, 24.3, 24.5, 24.6.2, 24.7, 24.8, 25.3, 25.5, 25.6.2, 25.7, and 25.8

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

11. As of September 17, 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

J. Hector, Doctor in Geology (Ph.D.)

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

P:\U4457 Sintoukola Potash ESIA\Project\Reports\PFS\QP letter

17 September 2012

CERTIFICATE OF AUTHOR

I, Jane Joughin, Pr.Sci.Nat., MSc do hereby certify that:

1. I am Principal Environmental Scientist of:

   SRK Consulting (UK) Ltd
   5th Floor, Churchill House
   17 Churchill Way
   Cardiff
   CF10 2 HH

2. This certificate applies to the technical report titled NI 43-101 Technical Report Sintoukola Potash Project Republic of Congo with an Effective Date of 17 September 2012 (the “Technical Report”).

3. I have a BSc Honours degree from the University of Natal, South Africa, obtained in 1986. I also have an MSc for the University of the Witwatersrand, South Africa, obtained in 1988.

   I am a professional environmental scientist, registered with the South African Council of Natural Scientific Professions (Registration Number 400057/94).

   I have worked as an environmental scientist for a total of 20 years since my graduation from university. For most of this time I have worked on the environmental and social impact assessments (ESIAs) for new developments as part of project feasibility studies and applications for environmental authorisations, permits and/or licences. This work has included preparation of environmental and social management plans. Most of my experience has been gained on large-scale mining and mineral processing projects. To date, I have worked on over 35 ESIAs and I have managed half of these. My ESIA experience is complemented with work on existing operations, including environmental compliance auditing and due diligence reviews. I also have experience working on projects involving development of new dams, roads and railways.

4. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
5. I have not visited the property that is the Sintoukola Potash Project property.
6. I am responsible for the preparation of Sections 3.3, 3.4, 16.7.1, 18, 23.9 and 25.9 of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101
8. I have had prior involvement with the Sintoukola Potash Project that the subject of the Technical Report. I am the reviewer of the ESIA for the Sintoukola Potash Project and have been actively involved in the ESIA process since its inception in 2011.

9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

10. As of 17 September 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

Jane Joughin, Principal Environmental Scientist, Pr.Sci.Nat., MSc

SRK Denver
Suite 3000
7175 West Jefferson Avenue
Lakewood, CO 80235

T: 303.985.1333
F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF QUALIFIED PERSON

I, Johan Boshoff MEng, P.Eng. do hereby certify that:

1. I am a Principal Consultant of:

   SRK Consulting Australasia
   10 Richardson Street
   West Perth, WA, Australia, 6005

2. This certificate applies to the technical report titled NI 43-101 Technical Report, Sintoukola Potash Project, Republic of Congo, with an Effective Date of September 17, 2012 (the "Technical Report").

3. I graduated with a degree in Civil Engineering from the University of Pretoria in 1993. In addition, I have obtained a Masters Degree in Geotechnical Engineering (1998, University of Pretoria). I am a Member of the Engineering Council of South Africa, Engineers Australia and the Australian Institute for Mining and Metallurgy. I have worked as a Geotechnical Engineer for a total of 18 years since my graduation from university. My relevant experience includes over 18 years of experience in site investigations, site selection, design, construction, reclamation and project management of engineered structures for mine waste disposal including tailings dams, heap leach pads and mine waste rocks dumps. I have managed multi-disciplinary studies for a broad range of tailings projects related to more than 25 tailings dams, including upstream, centreline and downstream raised impoundments. I have specific experience with the site selection, design, commissioning and operation of various tailings dams in Southern Africa, Central and West Africa, Asia, Australia and Latin America.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Sintoukola Project property on 11 March 2012 for 4 days.

6. I am responsible for the preparation of the Reside Storage Facility (RSF) in the Executive Summary and Sections 16.7.3, 19.1.5.1, 19.2.5.1, 23.6.1, 24.6.1 and 25.6.1 of the Technical Report.

7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.

8. I have not had prior involvement with the Sintoukola Project that is the subject of the Technical Report.

9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

10. As of September 17, 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

"Signed" *"Sealed"*

Johan Boshoff

U.S. Offices:
Anchorage 907.677.3520
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F: 303.985.9947

denver@srk.com
www.srk.com

CERTIFICATE OF AUTHOR

I, Neal Rigby, CEng MIMMM, PhD do hereby certify that:

1. I am Principal of:

   SRK Consulting (U.S.), Inc.
   7175 W. Jefferson Ave, Suite 3000
   Denver, CO, USA, 80235

2. This certificate applies to the technical report titled NI 43-101 Technical Report, Sintoukola Potash Project, Republic of Congo, with an Effective Date of September 17, 2012 (the "Technical Report").

3. I graduated with a BSc degree in Mineral Exploitation (Mining Engineering) with a first class honors in 1974 and a PhD in Mining Engineering in 1977 both from the University of Wales, UK. I am a member of the Institute of Materials, Mining and Metallurgy and a Chartered Engineer through the Council of Engineering Institutions. I have worked as a mining engineer for a total of 36 years since my graduation from university.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Sintoukola Property

6. I am responsible for mining and reserves, and marketing and economics in the Executive Summary, Sections 1, 2.4, 2.5, 2.6, 2.7, 5.10, 8.3.7, 13, 14, 16.6, 16.8, 17, 19.1, 9.1.1, 19.1.8, 19.1.9, 19.1.10, 19.2, 19.2.1, 19.2.8, 19.2.9, 20, 22, 22.1, 22.2, 23.2, 23.10, 24.2, 24.10, 24.11, 25.2, 25.10, 25.11, 26 and 27 of the Technical Report.

7. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my previous involvement was in the preparation of the reports titled, "NI 43-101 Technical Report, Sintoukola Potash Project, Republic of Congo" with a report date of August 1, 2011, and "Trade Off Study, Sintoukola Potash Project, Republic of Congo" with a report date of October 7, 2011

9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.

10. As of September 17, 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

"Signed"

Dr. Neal Rigby, CEng, MIMMM, PhD

CERTIFICATE OF QUALIFIED PERSON

Paul O'Hara, P.Eng.
AMEC Americas Limited
301 – 121 Research Drive
Saskatoon, Saskatchewan S7N 1K2

I, Paul O'Hara, P.Eng., am employed as the Manager, Process with AMEC Americas Limited, Saskatoon Office.

This certificate applies to the technical report entitled "NI 43-101 Technical Report, Sintoukola Potash Project, Republic of Congo" with an effective date of September 17, 2012.

I am a member of the Association of Professional Engineers and Geoscientists of Saskatchewan (Member No. 11687). I graduated from the University of British Columbia at Vancouver, British Columbia, with a Bachelor of Science degree in Mining and Mineral Process Engineering in May 1986.

I have practiced my profession for 26 years. I have been directly involved in the operation of copper, gold, and potash processing plants in Canada. I have been involved in process design for gold and potash process plants in Canada, England, and the Republic of Congo. As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have visited the Sintoukola Property from September 2-4, 2011.

I am responsible for the Metallurgy and Process in the Executive Summary, Sections 11, 15, 19.1.1, 19.1.3, 19.2.1, 19.2.3, 23.4, 24.4 and 25.4 of this Technical Report.

I am independent of Elemental Minerals as independence is described by Section 1.5 of NI 43–101.

I have been involved with the preparation of the Prefeasibility Study for the Sintoukola Project prepared by AMEC Americas Limited since May 2011.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Signed and sealed

Paul O'Hara, P.Eng.

Dated: September 17, 2012

CSA Global
Resource Industry Consultants

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ABN 67 077 165 532
www.csaglobal.com

17 September 2012

CERTIFICATE OF QUALIFIED PERSON

I, Simon Dorling, MSc, PhD, do hereby certify that:

1. I am Principal Consultant Geologist of:

   CSA Global Pty. Ltd.

   Level 2, Ord Street

   West Perth, WA, 6005

2. This certificate applies to the technical report titled "NI 42-101 Technical Repost Sintoukola Potash Project, Republic Of Congo" , with an Effective Date of 17th of September 2012 (the "Technical Report").
3. I graduated with a degree in Geology from University of Bonn, Germany in 1991. In addition, I have obtained a PhD in 1995, from the University of Western Australia, Perth in Economic Geology. I am a Member of the MAIG (Reg. No. 3808). I have worked as a Geologist for a total of 21 years since my graduation from university. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Sintoukola property several times. Most recently I visisted the property on 1st of September , 2012 for 7 days.
6. I am responsible for the preparation of Sections 2.1, 2.2, 2.3, 2.3.1, 3.1, 4, 5 (excluding 5.10), 6, 7, 8 (excluding 8.3.7), 9, 10, 21, 23.1, 24.1, and 25.1) of the Technical Report.
7. I am independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is as an advisor / consultant on geological and exploration aspects of work undertaken by the issuer.
9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of 17th of September 2012, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 17th Day of September, 2012.

Simon L. Dorling

[Seal or Stamp]

Appendix B: Laboratory Certifications

DAP Deutsches Akkreditierungssystem Prüfwesen GmbH

Unterzeichner der Multilateralen Abkommen von
EA und ILAC zur gegenseitigen Anerkennung

vertreten im

Deutschen AkkreditierungsRat

Akkreditierung

Die DAP Deutsches Akkreditierungssystem Prüfwesen GmbH bestätigt hiermit, dass die

K-UTEC AG Salt Technologies

Am Petersenschacht 7
99706 Sondershausen

die Kompetenz nach DIN EN ISO/IEC 17025:2005 besitzt, Prüfungen in den Bereichen

physikalische, physikalisch-chemische und chemische Untersuchungen von Wasser, Abwasser, Schlamm, Sedimenten, Abfall und Stoffen zur Verwertung, Klärschlamm, Böden und kontaminierten Böden; Analytik von Salzen und Salzlösungen; chemische Parameter im Rahmen der Trinkwasserverordnung: 2001 ohne radiologische Parameter; Probenahme von Roh- und Trinkwasser; Probenahme von Wasser und Abwasser

gemäß den in der Anlage aufgeführten Prüfverfahren auszuführen. Die Anlage ist Bestandteil der Urkunde und besteht aus 30 Seiten.

Die Akkreditierung ist gültig vom 2008-02-01 bis 2012-03-19.

DAR-Registriernummer: **DAP-PL-3044.00**

Berlin, 2008-02-01

i. V. Valbuena

Univ.-Prof. Dr.-Ing. habil. K. Ziegler
Geschäftsführer
DAP Deutsches Akkreditierungssystem
Prüfwesen GmbH

Siehe Hinweise auf der Rückseite.

1. Ausfertigung

Die DAP Deutsches Akkreditierungssystem Prüfwesen GmbH (im folgenden DAP genannt) ist Unterzeichner des Multilateral Agreement for Testing Laboratories (MLA) der European co-operation for Accreditation (EA) und der Mutual Recognition Arrangement (MRA) der International Laboratory Accreditation Co-operation (ILAC). Für Prüflaboratorien wurden von EA weitere bilaterale Abkommen zur gegenseitigen Anerkennung abgeschlossen.
Die Unterzeichner dieser Abkommen aus den nachfolgend aufgeführten Staaten erkennen ihre Akkreditierungen von Prüflaboratorien gegenseitig an:

Ägypten – Argentinien – Australien – Belgien – Brasilien – Volksrepublik China – Costa Rica – Dänemark – Deutschland – Estland – Finnland – Frankreich – Griechenland – Großbritannien – Hongkong – Indien – Indonesien – Irland – Israel – Italien – Japan – Kanada – Republik Korea – Kuba – Lettland – Litauen – Malaysia – Mexico – Neuseeland – Niederlande – Norwegen – Österreich – Philippinen – Polen – Portugal – Rumänien – Schweden – Schweiz – Singapur – Slowakei – Slowenien – Spanien – Südafrika – Taiwan – Thailand – Tschechien – Türkei – USA – Vietnam.

Der aktuelle Stand der Mitgliedschaft kann der jeweiligen website entnommen werden:
EA - http://www.european-accreditation.org
ILAC - http://www.ilac.org

Die Akkreditierung erfolgt aufgrund einer Begutachtung und des mit dem DAP abgeschlossenen Vertrages über die Akkreditierung eines Prüflaboratoriums nach den Regeln und Verfahren des Deutschen Akkreditierungssystems, gemäß den Normen DIN EN ISO/IEC 17025 und DIN EN ISO/IEC 17011.

Die materiellen und personellen Voraussetzungen nach DIN EN ISO/IEC 17025 für die in der Akkreditierungsurkunde angegebenen Prüfgebiete sowie für die in der Anlage zur Akkreditierungsurkunde beschriebenen Verfahren sind erfüllt.

Angaben über den Umfang der Akkreditierung (Prüfgebiete, Verfahren und Spezifikationen) sind in der Anlage zu dieser Akkreditierungsurkunde aufgeführt.

Die Anlage sowie die eingereichten Unterlagen sind Bestandteil der Akkreditierung. Änderungen bedürfen der Schriftform.

Die Akkreditierung wird unter dem Vorbehalt des jederzeitigen Widerrufs bei Wegfall der im Vertrag sowie in der Anlage zu dieser Akkreditierungsurkunde festgelegten Voraussetzungen erteilt.

NATA Accredited Laboratory

National Association of Testing Authorities, Australia
(ABN 59 004 379 748)

has accredited

Intertek Genalysis
Trading as Genalysis Laboratory Services Pty Ltd

following demonstration of its technical competence
to operate in accordance with

ISO/IEC 17025

This facility is accredited in the field of

Chemical Testing

for the tests shown on the *Scope of Accreditation* issued by NATA

Jennifer Evans
Chief Executive Officer

Date of accreditation: 20 September 1991
Accreditation number: 3244

NATA
WORLD RECOGNISED
ACCREDITATION

NATA is Australia's government-endorsed laboratory accreditor, and a leader in accreditation internationally. NATA is a signatory to the international mutual recognition arrangements of the International Laboratory Accreditation Cooperation (ILAC) and the Asia Pacific Laboratory Accreditation Cooperation (APLAC).