THE KOIDU VERTICAL PIT – SIERRA LEONE

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ABSTRACT
The Koidu Kimberlite Project is situated in the Kono District of Sierra Leone, approximately 330 km east of the capital city, Freetown. The mining lease area is 4 km² and comprises 2 kimberlite pipes, 4 dyke zones and a number of blows off the dykes. Koidu Holdings established on the property and began dewatering and the removal of silt from the existing 30 m excavation of the No.1 Pipe in 2003. While the original planning for the pipes included conventional open pit operations, mitigating factors such as the proximity of the town, the presence of houses close to the perimeter of the pipe, the size of the pipe and the planned open pit depth with relevant stripping ratios, as well as uncertainties on the grades in respect of an underground operation, lent itself to the vertical pit concept.

The vertical pit was established using an A-frame headgear for the hoisting arrangement, with sidewall support comprising a combination of 20 m and 40 m long cable anchors, rock bolts, wire mesh and pneumatically applied concrete. The pit reached steady state production in January 2007 and produced some 26,000 tonnes of head feed per month until August 2007 when mining reached a depth of 74 m below the pit collar (approximately 126 m below surface). Mining was suspended following a sidewall failure that disrupted operations. Now that the grades in No.1 Pipe have been validated, focus has been switched to an underground operation taking account of both No. 1 Pipe and No. 2 Pipe.

While the mining costs for the vertical pit have been higher than for a conventional open pit, the concept did circumvent the social implications, the issue of grade consistency and diamond value at depth, as well as providing the time frame to carry out a full underground mining study.

1. INTRODUCTION – Koidu Holdings S.A. is wholly owned by BSG Resources (Pty) Ltd and holds the rights to the Koidu Kimberlite Project in the Kono District within the historically productive diamond fields of Sierra Leone. After the civil war ended in 2002, the previous owners of the rights to the property, DiamondWorks Ltd (renamed Energem Mining), entered into a joint venture with a subsidiary company of the Beny Steinmetz Group (BSG) for the evaluation and development of operations at the defunct Koidu Kimberlite Project. Between 2002 and 2004, Koidu Holdings was established and the shareholding changed progressively from the initial 50/50 to a 60/40 arrangement, where BSG Resources assumed management control of the Company. Since February 2007, when Energem’s 40% shareholding was acquired, Koidu Holdings has been wholly owned and operated by BSG Resources.
The Koidu Kimberlite Project is situated within the Tankoro Chiefdom of the Kono District in the Eastern Province of Sierra Leone, approximately 2 km south of the district capital, Koidu, and approximately 330 km east of Freetown, the capital city (Figure 1).

When the Company returned to the Koidu Kimberlite Project Mining Lease Area in 2002, No. 1 Pipe was flooded and was being used by the local community. After dewatering of the 30 m deep open pit excavation made by Sierra Leone Selection Trust and the National Diamond Mining Company in the 1970s and 1980s, removal of the remaining silt and mud commenced. Between March 2003 and November 2003, the No. 1 Pipe was dewatered, cleaned and prepared for the conventional open pit mining phase. No. 1 Pipe was mined by conventional open pit methods from January 2004 to May 2004, after which mining shifted to No. 2 Pipe. During this time and having assessed the scale of the disruption to the community and the impact of blasting, the Company considered either switching to standard underground mining methods or attempting the innovative and pioneering vertical pit mining technique on the No. 1 Pipe. Continuing with the open pit at No. 1 Pipe was ruled out because, apart from the negative social and environmental impacts posed by the close proximity to the town (Figure 2), the small size of the pipe would have limited the maximum depth of mining below the existing pit floor to a further 80m (120 m below surface) before the high stripping ratio would have made the deposit uneconomical to mine. At that stage, insufficient information regarding grade and diamond value was known to justify establishing an underground mining operation from the outset. The vertical pit option appeared to hold the most merit, requiring limited capital investment for establishment, and minimizing the social impacts and environmental footprint of the operation. SRK Consulting was approached to determine whether vertical pit mining would be appropriate from a geotechnical perspective and, if so, to undertake a vertical pit design for No. 1 Pipe.

In this paper we discuss the various geotechnical, operational and economic factors that played a role in the selection of the vertical pit mining method for the No. 1 Pipe kimberlite, providing a brief overview of the decision making process from identification of geological and geotechnical constraints, through the design and support specification phase, to the site establishment, implementation and mining phases.

Vertical pit mining is a relatively new mining technique, which has been successfully implemented only once before, at the Nyala chrome deposit in Zimbabwe in the late 1990s (Redford & Terbrugge, 2000). By supporting the vertical sidewalls of the pit by means of cable anchors, rock bolts, wire mesh and wetcrete, a large open air shaft roughly the size of the ore body is established. We have refined the technique and shown that it can be employed successfully, even in very difficult environments. Having attained a depth of
126 m from surface and 74 m from the headgear collar established at the level of the open pit floor, the Koidu vertical pit essentially represents the largest diameter “shaft” to have reached such depths.

Mining activities were brought to a halt in August 2007 after the Vertical Pit experienced a significant rock fall on the southern margin of the pipe. Fortunately there were no injuries to personnel and no equipment was damaged. The cause of the failure was attributed to the unusually high rainfall measured during the season and its destabilising effect on the kimberlite in the sidewall, in addition to the large size of the rock wedge. After investigation by SRK Consulting and concern for the safety of those working inside the pit, it was decided to implement further support measures and to wait until the end of the rains before resuming mining activities in the vertical pit.

2. GEOLOGY – Sierra Leone is situated on the Man Craton of the Southern West African Shield. The Archaean granitic shield contains elements of early sedimentary and mafic formations and a group of supracrustal greenstone belts with banded ironstone and detrital sediments. The basement granite and rocks of the younger Kambui Group have been deformed and metamorphosed together with the underlying gneisses and intruded by late and post orogenic granites. The granitic rocks are cut by several fracture systems which are widely believed to have controlled the emplacement of kimberlite and dolerite dykes. Foliations and faults in the basement granites are almost parallel, trending roughly north-south to northeast-southwest. Second order fractures are developed in northeast orientations.

The Koidu Kimberlite Project Mining Lease Area contains two kimberlite pipes (No. 1 Pipe and No. 2 Pipe), four dyke zones (Dyke Zones A, B, C and D), the Ring Structure, a small blow on Dyke Zone A and another blow on Dyke Zone B. The country rock in the Koidu area is granodiorite gneiss, containing metamorphic inclusions of amphibolite, ultrabasic schists, quartzites and granulites, among others. The kimberlite dyke systems at Koidu have strike orientations of between 070° and 074° and are restricted to the area between two parallel regional scale 010° trending faults.

Both kimberlite pipes consist of multiple massive volcanioclastic kimberlite (diatreme facies), magmatic kimberlite dykes (hypabyssal facies) pre- and postdating pipe formation, as well as transitional kimberlites. The kimberlite pipes have been eroded down to the root zone, having irregular shapes, particularly in the case of No. 1 Pipe.
3. VERTICAL PIT DESIGN – The Koidu vertical pit mining project was characterized by a unique set of circumstances comprising political, social, geographical and administrative challenges. A highly detailed design process with large numbers of boreholes and multiple sets of laboratory tests, followed by of the required design work was not financially feasible, nor practically possible. The decision was made to base the design on the information already collected during the earlier open pit feasibility study, with an additional site visit to audit the core logs and give the designers a better feel for the geology of the country rock surrounding the pit.

The design of the vertical pit excavation was carried out using various calculation approaches to ensure design integrity. The material parameters were obtained by back analysis of the existing highwall at the No. 1 Pipe, excavated during previous mining attempts, and using the Hoek-Brown failure criteria based on the laboratory tests and core logs. The design calculations comprised limit equilibrium, 2-dimensional and 3-dimensional numerical modeling. The results obtained from each of these methods were ranked according to stability and a final design decision made.

3.1 ROCK MASS STRENGTHS – The design rock mass strengths were selected based on the back analysis of the existing highwall and the Hoek-Brown failure criteria (Hoek et al., 2002). In addition to the normal factors incorporated by the Hoek-Brown failure criteria, the following factors were considered subjectively:
• Additional confinement provided by the constrained geometry of the vertical pit sidewalls;
• The small scale of the vertical pit in terms of cross-sectional area and height; and
• Time related degradation of the rock mass strength through creep and strain softening.

The final design parameters based on the above evaluation were:

<table>
<thead>
<tr>
<th>Material</th>
<th>Cohesion (kPa)</th>
<th>Friction Angle (°)</th>
<th>Tensile Strength (kPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Granite Rock Mass</td>
<td>800</td>
<td>41</td>
<td>70</td>
</tr>
<tr>
<td>Granite Joints</td>
<td>0</td>
<td>41</td>
<td>0</td>
</tr>
</tbody>
</table>

As these parameters included many subjective components that may have changed with time once mining commenced, these parameters were based on the following design assumptions:

• Effective pit limit blasting;
• Effective dewatering of the pit sidewalls;
• Ongoing monitoring of the pit sidewalls to give warning of unacceptable sidewall displacement and possible sidewall collapse;
• A maximum mining life of 3 years from start to completion to limit the time dependant changes in rock strength;
• Vertical sidewalls or flatter, no overhangs allowed;
• A constant pit footprint to avoid deviation from the original assumptions.

3.2 GEOLOGICAL STRUCTURES – For geotechnical design purposes, the structures in the granitic country rock were summarized as follows:

• The basic structural fabric consisted of three joint sets: the first set consisted of sub-vertical joints striking roughly parallel to the sidewalls, the second set roughly perpendicular to the sidewalls and the third set sub-horizontal. This resulted in roughly cube shaped rock blocks with sides ranging from 0.3 m to 1 m (Figure 3).
• In addition to these there were randomly orientated joints with continuieties in excess of 100 m. These were typically dipping at angles of between 25° and 50° and if orientated correctly, had the potential to create wedges that could not practically be supported.
3.3 GROUNDWATER – The groundwater level was close to the ground surface before site establishment began and consequently would exercise a negative influence on the stability of the excavation. It was considered that due to the blocky nature of the rock mass, the ground water would be concentrated in joints and fissures. This would simplify dewatering as it was a matter that could be addressed by drilling and pumping. For this reason, it was decided to assume a design water level that renders the pit effectively dry to reduce water pressure destabilizing the vertical pit walls.

To achieve dry pit sidewalls, vertical wells were recommended through which pumping could be achieved. Dewatering was achieved through horizontal drainage holes, vertical and inclined wells and pumping of water from an old exploration shaft near the vertical pit with some galleries into the orebody.

3.4 FAILURE MECHANISMS – Site inspection and analysis of the available information indicated that the following failure mechanisms may be expected:

- Rock mass failure;
- Toppling failure along the sub vertical joint planes;
- Complex wedge failure (multiple cut off planes with some rock mass boundaries); and
- Ravelling.
Support elements were selected to counter each of these mechanisms. Through the JPLOT analysis it was established that a small chance did exist of a kinematic mechanism large enough such that it could not be supported economically.

3.5 ANALYSIS – The analysis was undertaken to show that each of the failure mechanisms was either not feasible, or to determine what was required to ensure that each mechanism was sufficiently supported. In the case of the kinematic analysis it was shown that mechanisms could exist, that could not be economically supported. Consequently, there was an element of risk that had to be accepted before mining could commence. As mining progressed validation analyses were carried out to ensure that the design remained reliable.

3.5.1 Rock Mass Failure – Three different numerical modeling software packages were used to determine the vertical pit’s resistance to rock mass failure. These were: Examine 3D™, Phase™ and FLAC™.

Examine 3D is a 3 dimensional boundary element package which is capable of modelling one elastic material only. Joints or other discontinuities as well as support elements could not be included in the model. It was therefore decided to use a lower bound allowable stress criteria to indicate rock mass failure. The allowable stress was determined using the strength factor output provided by the programme. The result showed some minor yielding near the sidewall surface but these were put down to numerical effects. An example output figure is presented as Figure 4.

Figure 4 Contours of Maximum Shear Stress

Phase™ is a 2 dimensional finite element software package which is capable of modeling elastic-plastic materials as well as support elements. Phase™ was used to model plane strain sections through the pit as well as horizontal plane strain slices. Plane stress slices would have been preferred but the package did not support this option. The results of the plane strain analysis indicated that the rock mass would be stable in plane strain without support although some plasticity was evident. The results of the axisymmetric analysis showed that the model was stable without support with very little plasticity.
Due to the limitation in Phase²TM in not being able to model plane stress, it was decided to include FLAC™ calculations in the design. The FLAC™ models showed that a section through the pit was stable in plane stress without support with some plasticity. If active support was included the degree of plastic yielding was dramatically decreased. The horizontal plane stress slices confirmed this result while also accounting for the pit plan geometry.

Based on the analysis carried out, it was decided that the rock mass would require minimal support and that the support would be required to prevent kinematic failure.

3.5.2 Toppling along sub vertical joint planes parallel to the sidewall – Toppling failure as one could expect for a sub vertical highwall is difficult to analyze correctly as it is more a combination of buckling and toppling. The decision was taken that the regular pattern of tensioned anchors and grouted dowels would provide adequate support for the toppling/buckling that may occur.

3.5.3 Complex wedge failure – Statistics were compiled for the joint length, spacing and orientation parameters based on data collected during the site visit. The statistics were used as input for an in house SRK software program (JPlot) to generate artificial joint networks. The geometry of the vertical pit was superimposed on the joint networks and the likely sizes of potential wedges calculated. The statistics describing the depth (Figure 5) and volume, and therefore weight, of the potential wedges were used to determine the design length, spacing and capacity for the tensioned anchors, grouted dowels and wetcrete as the function of these support elements was primarily to support kinematic failures. The analysis also showed that the possibility existed that a wedge could exist that was so large that it could not be supported.

![Figure 5](image-url)  
**Figure 5**  Histogram of complex wedge depth based on JPlot analysis
3.5.4 **Falling of loose ground** – Falling of loose ground was not covered specifically in the analysis but the specification of wetcrete, wiremesh and grouted dowels were considered adequate support.

3.6 **LATERAL SUPPORT** – Based on the analysis and other design work, the following support elements (Figure 6) were recommended:

- Wetcrete with a compressive strength of 30 MPa and minimum thickness of 75 mm;
- Weldmesh with wire diameter of 4 mm and aperture size of 150 mm;
- Fully bonded grouted dowels with an ultimate tensile strength of 14 tonnes at a 1.5 by 1.5 m spacing;
- 20 m long cable anchors tensioned to 40 tonnes at a 5 m by 7 m spacing. The fixed end was 5 m long leaving 15 m free for tensioning which was grouted after some time. Provision had to be made to decrease the spacing to 5 m by 5 m for 30 % of the time as well as for 30 % of the anchors to have a 40 m length as required.

During mining it became necessary to install 40 m anchors near the collar elevation to prevent a kinematic mechanism developing beneath the hoist platform. The actual anchor system used had to be varied several times during mining due to difficulties sourcing the required wire rope but the total support effort remained constant. Difficulties were also experienced with a fully grouted fixed end as this caused delays to the mining schedule. Experiments with mechanical expansion shells were only partly successful as they failed to grip in some instances. Doubts were also expressed about their reliability due to the generally low confinement of the rock mass near surface.

![Figure 6 Vertical Pit – schematic support pattern.](image-url)
Equipment for the installation of the support system included:

- Drilling – Furukawa PCR 200 (Compressed Air Top Hammer) & Seco S23 Jackhammers;
- Tensioning - 50 tonne hydraulic jack;
- Grouting - Ictus CM5001 Mixer and M60 Grout Pump (for ropes), Lategan Pump (for rock bolts);
- Wetcreting - Chryso Wetcreter 200.

The support system (Figure 7) was supervised and recorded by the Shift Boss and quality inspected by a qualified and experienced civil engineer. SRK Consulting undertook regular site inspections to audit implementation and compliance with design.

4. IMPLEMENTATION – Following the positive outcome from SRK’s geotechnical analysis and vertical pit design study, the decision was taken to proceed with the development of a vertical pit at the Koidu No. 1 Pipe.

4.1 SITE ESTABLISHMENT - With the steep sided open pit already at 52 m below surface at the time, establishing the vertical pit collar at that level meant that the preparation work and time required for the development of the collar infrastructure and erection of the headgear and winder was significantly reduced. Nevertheless, certain modifications to the open pit profile were necessary to provide sufficient working space and suitable access for the ore transport vehicles (Figure 8).

Based on the available geological information, the maximum extent of the kimberlite pipe at depth was projected to surface, forming the perimeter of the vertical pit. The limited drillhole data suggested that the pipe plunged gently towards the south and the design work had taken that into account. However, during collar preparation, evidence was found of kimberlite infiltrating fractures in the country rock at the intended location of the headgear.
Further investigations revealed a bulge in the pipe, typical of the irregularities commonly observed in the root zone of kimberlite pipes. The headgear had to be moved to a new location, requiring substantial additional waste stripping. The enlarged perimeter of the vertical pit made reworking of the geotechnical analyses and design work necessary. The increase in the pit floor area from approximately 3,480 m$^2$ to 4,388 m$^2$ impacted negatively on the final design depth, reducing the maximum depth from 180 m to approximately 100 m.

![Aerial view of No. 1 Pipe open pit prior to establishment of the vertical pit.](image1)

![Aerial view of No. 1 Pipe pit after waste stripping and establishment of the vertical pit collar.](image2)

![Commissioning of the headgear and winder in November 2005](image3)

![Vertical pit headgear and collar infrastructure](image4)

**Figure 8** Stages in the development of the Koidu vertical pit.

The Koidu Kimberlite Project reached steady state production from the No. 1 Pipe vertical pit during the third quarter of 2006, despite setbacks due to the delayed arrival of capital items required to meet the production schedule, breakdowns of the hoisting arrangement, and variable stripping ratios encountered due to the irregular geometry of the pipe. The vertical pit as at March 2008 is shown in Figures 9 and 10.

Production from the No. 1 Pipe Vertical Pit progressed satisfactorily during the first three quarters of 2007, with the headfeed target of 26,000 tonnes per month being exceeded on three occasions. In addition to the higher tonnages mined and processed an increase in the average grade of the orebody was observed.
In comparison with the fairly consistent average grade of approximately 0.53 carats per tonne recovered from 2004 to 2006, the average grade for No. 1 Pipe increased by 20\% to 0.63 carats per tonne in 2007.

**Figure 9** Satellite image of the Koidu No. 1 Pipe vertical pit draped over the 3D topographic surface.

![Satellite image of the Koidu No. 1 Pipe vertical pit draped over the 3D topographic surface.](image)

**Figure 10** E-W slice through the Koidu No. 1 Pipe vertical pit, showing the narrow benches of the open pit above the vertical pit collar and the vertical sidewalls forming a large shaft on the kimberlite pipe.

![E-W slice through the Koidu No. 1 Pipe vertical pit](image)

### 4.2 REGULATORY FRAMEWORK –

Vertical pit mining combines various open pit and underground mining concepts and, therefore, required regulations related to both. The Mines and Minerals Act (1994) of the Republic of Sierra Leone does not address the technical control and safety regulations specifically related to this mining method and, hence, Koidu Holdings adopted the Mining Regulations contained in the Minerals Act No. 50 (1991) of the Republic of South Africa, as provided for in Part XV of the Sierra Leone Mines and Minerals Act, 1994.
4.3 MANAGEMENT – The Head of the Mining Department was responsible for the overall management of the vertical pit, being a qualified Mining Engineer and holder of a Mine Manager’s Certificate of Competency. Shaft Sinkers (Pty) Ltd from South Africa were contracted to supply and service the hoisting arrangement. Shaft Sinkers appointed a site manager, who took full responsibility for the hoisting arrangement.

Mining activities were overseen and coordinated from the vertical pit collar by the Mine Captain, who was responsible for all administrative, logistical and ad hoc support to the mining team on shift in the vertical pit. Activities in the vertical pit (including drilling in preparation of blasting, drilling of anchor holes, installation of anchors, installation of wiremesh and wetcreting, loading & hauling and pumping) were supervised and coordinated by an expatriate Shift Boss (a qualified miner and holder of both open cast and underground blasting certificates). The Shift Boss was assisted by a national mining supervisor and a foreman who supervised each of the activities in the pit.

The technical support team, which included the Mine Surveyor and Geologist, reported to the Head of the Mining Department and were responsible for daily, weekly and monthly surveying and monitoring, as well as assisting with the monthly mine planning.

The vertical pit operated on a 24 hour basis, with three eight hour shifts, each shift consisting of a Shift Boss and staff compliment for each activity.

Due to the loss of man hours related to tropical diseases and the 3-month-on-duty, 3-week-off-duty expatriate leave cycle, the Company chose to appoint 3 additional expatriate Shift Bosses in order to ensure consistency and uninterrupted production.

4.2 PLANNING – The mine plan was updated monthly by the Head of the Mining Department, assisted by the Mine Surveyor, Geologist, Shaft Sinkers Site Manager and Chief Engineer. Whilst the LOM plan and annual depletion plan formed the basis of the monthly planning, a number of issues had to be taken into consideration during the detailed monthly planning sessions. One of the most important of these was the restriction on the time and frequency of blasting. All households within the 250 m blast envelope had to be evacuated before each blast, requiring a well coordinated effort between the Company and local authorities to ensure the safety of the community and the protection of property during the time before and after each blast. The Company reached an agreement with the Government and the local community that blasting would be limited to twice per week, taking place between 15H00 and 16H00. Careful and detailed planning was required to ensure that the sequencing of the various activities within the vertical pit were coordinated with the permitted blasting schedule. Furthermore, adherence to the schedule was paramount, because slippage in any one of the activities would impact on those down the line.
For example, the support installation, including the drilling of holes for cable anchors and rock bolts had to be completed before the meshing and wetcreting could be applied, all of which had to be completed before the next scheduled blast in a particular area. The cycle time for installing one cable anchor was four days starting from the moment the anchor was installed into the hole until it was tensioned and signed off. The planning had to allow for this standing time and focus activities in other parts of the pit while ensuring that the drill rigs and support teams were working from a safe and comfortable footwall. The support activities and mining production activities, therefore, had to be carefully synchronized to ensure continuous operation of the vertical pit.

Another critical factor in the mine planning was sustaining the production target of at least 26,000 tonnes ROM ore to keep the 50 tonne per hour DMS processing plant operating at optimum capacity. Hoisting capacity was limited to 1,728 tonnes per day, with the average hoisting achieved being approximately 1,400 tonnes per day. Downtime necessary for hoist examinations and maintenance every 10,000 tonnes or at least once per week had to be accounted for in the planning, as well as the loss of shifts due to blast preparation.

Other factors that had to be considered in the planning included the maintenance of a sump for drainage and dewatering purposes, as well as creating and maintaining a free face for optimal blasting results. In cases where problems encountered during the month set the schedule out of synch, these activities had to be incorporated into the work programme for the following month.

4.5 INFRASTRUCTURE – The following infrastructure was established at the vertical pit collar:

- A-frame headgear with traverse car;
- Winder house, 111 KW Single Drum Winder;
- Generator house with two 550 KVA generators synchronised (one primary and one standby) to provide power to the winder, surface infrastructure and pumps. One 35 KVA generator provided power for emergency lights;
- Compressor house with three 1,050 Cfm compressors – providing compressed air to two drill rigs, two wetcreters (two primary and one standby);
- 5,000 Gallon diesel storage tank and distribution system to supply fuel to equipment in the bottom and on the collar;
- Access and egress security control systems;
- Offices for the Mine Captain, Mining Clerk and Shaft Sinkers Site Manager;
- Stores for support consumables, mechanical consumables, electrical consumables, oils and lubricants and general;
- Small workshop for maintenance and minor repairs to pumps, wetcreters, hoisting equipment and welding shop;
• Banksmen’s cabin and signal room;
• First aid station;
• Dam with stage pumping arrangement;
• Lighting plant;
• Safety wall;
• Drainage system;
• Survey beacons.

Infrastructure (Figure 11) within the pit included:

• A ladderway in compliance with the requirements of the South African mining regulations;
• Service pipes:
  — 150mm diameter for compressed air;
  — 100mm diameter for pumping;
  — 50mm diameter for industrial water;
  — 25mm diameter for fuel;
• Two power cables; one 525 volt and one 380 volt.

The following equipment remained inside the pit for the full duration:

• 1 Volvo EC360BLC Excavator;
• 1 CAT 318 Excavator;
• 1 Volvo A25C ADT;
• 1 Volvo L90 FEL;
• 2 Grindex Submersible Pumps;
• 2 Chryso Wetcreters;
• 2 PCR 200 Drill Rigs.
4.6 DRILLING AND BLASTING – Due to the small floor area of the vertical pit and the fact that withdrawing the earthmoving, drilling, pumping and wetcreting equipment from the pit prior to each blast was not practically feasible, much emphasis was placed on the blast design and the type of explosives used in order to avoid damaging the vertical pit sidewalls or in-pit equipment.

An additional difficulty arose sourcing suitable explosives and importing them into the country. AEL Ghana was the only supplier able to provide the required explosives and accessories. Working in conjunction with AEL’s blasting experts and with input from SRK Consulting, an optimal blast design was found for the vertical pit. AEL’s initial blast design involved the use of 45 mm and 54 mm diameter holes. These diameters were based on a bench height of between 2 m and 3 m, which was limited in attempts to minimize damage to the pit walls. However, due to logistical and operational reasons, the mine standardized on 76 mm diameter for both anchor and production holes. In view of the sequencing and support parameters, a 3.2 m bench height, with a 1.5 m x 1.5 m pattern was maintained. Drill/blast operations are shown in Figure 12.
Each blast layout consisted of a combination pre-split, narrow “trim”, and production blasts. The pre-split was fired instantaneously, followed by the narrow “trim” blasts with a maximum of 4 rows adjacent to the final wall i.e. approximately 6m wide with a burden of 1.5 m and a pre-split standoff of 0.8 m and a buffer row with half the normal spacing (i.e. 0.85 m), and half the normal charge mass per hole.

The explosive prescribed by AEL was R100G emulsion which was transported and charged using an AEL U117E charging unit. The U117E was an electrically operated emulsion pump. The unit was provided with a quick-fit electric coupling to connect to the 525 volt power available in the pit. The pump was mounted on a skid so that it could be dragged into position on the footwall of the vertical pit. A delivery hose was long enough to reach the holes to be charged. A compressed air double diaphragm pump was used to pump the R100G emulsion from the ISO – container into the U117E charging unit, which has a capacity of 1,500 kg of emulsion.

The mine was unable to use electronic detonators due to restrictions on the importation of explosives and accessories.

4.7 LOADING AND HOISTING – Loading from the muck pile was undertaken by excavator onto a 25 t Volvo ADT then hauled from the bench face to a position adjacent to the kibble. A second excavator loaded the material into a kibble which was hoisted to the collar elevation (Figure 13).

An A-frame headgear was used for hoisting the material from the excavation. The A-frame had four legs supporting a beam which extended 2 m beyond the edge of the pit, allowing a kibble to be lowered into and raised from the pit footwall, and then traversed away from the pit to an area where the material was dumped into a parked truck. The A-frame was supported on a competent concrete foundation, which was anchored to the footwall, thereby securing the cantilever away from the edge of the pit.
A 111 kW single drum winder was positioned 20 m away from the headgear, with the rope leading directly to a deflecting sheave and then to a multi sheave system allowing four falls of rope to handle the weight of the kibble and payload. Power for the winder was provided from a 550 kVa diesel generator which was sited near to the winder.

The kibble used for hoisting material was 2250 mm in diameter x 2000 mm high and had a payload capacity of 12.5 tonnes. It was attached by chains to the hoist rope via an annealed hook. The material was discharged from the kibble by attaching a lazy chain to the kibble bottom, and slacking off the hoist rope. In order to optimize cycle time, two kibbles were used, one being loaded at the bottom of the pit whilst the other was being hoisted. The following table indicates hoisting capacity from various depths:

<table>
<thead>
<tr>
<th>Depth (m)</th>
<th>Cycle time (min)</th>
<th>Tonnes / hour</th>
</tr>
</thead>
<tbody>
<tr>
<td>30</td>
<td>9.3</td>
<td>80</td>
</tr>
<tr>
<td>60</td>
<td>10.4</td>
<td>72</td>
</tr>
<tr>
<td>90</td>
<td>11.7</td>
<td>64</td>
</tr>
</tbody>
</table>

### 4.8 SLOPE MONITORING PROGRAMME

The purpose of the monitoring programme was to detect early signs of impending fall of ground and to give early warning of such an event to evacuate personnel in the bottom of the pit. The design monitoring programme consisted of the following:

- Regular visual inspections of the wetcrete and surrounding rock mass to detect any signs of movement or deterioration of the support systems;
- Regular inspection of the weepholes to monitor the rate of groundwater flow and level of the phreatic surface on the sidewall face; and
- Regular survey monitoring of targets on the sidewall face as well as on the collar of the vertical pit.

To this end, ten survey beacons were installed on surface in the bank/ winder house area in order to monitor displacements. The Mine Surveyor was responsible for monitoring these targets on a daily basis and results were
recorded for analysis and future reference. Further to these installations, a series of anchor heads on all walls were measured. Three columns of targets/anchor heads on all walls at the 325 m level, the 300 m level and the 275 m level were monitored.

In addition to providing regular survey on selected targets in and around the vertical pit, and due to the nature of the operations and exposure of personnel in the pit, SRK recommended that the Company introduce a Ground Probe Radar slope stability monitoring system.

The Ground Probe Radar system is an effective real time monitoring system which would reduce the exposure of personnel in the pit. Monitoring data from the radar, is displayed in real time with unit cell size being defined as required, and output depicting displacements registering yellow through red for movements into the pit, and blue through green for displacements out of the pit. Further to the visual display, specific critical cell/pixels can be identified with real time, time versus displacement or rate of displacement plots generated as required. The system can be monitored on a 24/7 basis and is ideal for high risk / exposure levels.

### 4.9 DEWATERING

The significant volumes of ground water flowing into the vertical pit along the fractures and dyke zones impacted negatively on the stability of the vertical pit, particularly in areas where kimberlite stringers and veins occurred behind the sidewalls. A geohydrological investigation was undertaken by Digby Wells and Associates in order to determine the ground water flow regime and required pumping rates for the vertical pit, as well as to provide information on the ground water conditions likely to be encountered at depth. A total of eleven boreholes were drilled, with depths ranging from 91 m to 214 m. Total volumes to be pumped ranged from 26 to 33 litres per second, depending on the level of drawdown required (Figure 14).

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**Figure 14**

a) Inclined holes to intersect major joint sets, fractures and dyke zones.  
b) Ground water intersection.
ECONOMICS — Among the deciding factors in the selection of the vertical pit mining method for the exploitation of the Koidu No. 1 Pipe was the cost of establishing the vertical pit and the associated ongoing operating costs. A trade-off study between the conventional open pit option and the vertical pit option was undertaken, comparing capital requirements, operating costs and the time frame in which each of the options could be realistically implemented.

In order to maintain optimum processing capacity, the earthmoving fleet required to achieve the waste mining targets for open pit mining of No. 1 Pipe would have included 6 ADTs, 2 excavators, 1 dozer, 5 drill rigs and service infrastructure, at a total cost at the time of US$ 4.3 million. In comparison, the capital expenditure for the vertical pit was US$ 3.1 million, which included a smaller earthmoving fleet (4 ADTs, 2 excavators, 1 dozer, 2 drill rigs), as well as the construction of the headgear and winder foundations and the purchase of generators and support equipment over and above what would have been required for conventional open pit mining. The additional waste stripping for the relocation of the headgear and the enlargement of the vertical pit perimeter was also capitalized.

Therefore, the capital expenditure required for the vertical pit equipment and development, whilst being higher than for the open pit, was well below what would have been called for if underground mining had been the only other viable option.

<table>
<thead>
<tr>
<th>Item</th>
<th>Open Pit</th>
<th>Vertical Pit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Geotechnical/Mining Design</td>
<td>US$ 25,000</td>
<td>US$ 42,133</td>
</tr>
<tr>
<td>Foundation Design</td>
<td></td>
<td>US$ 27,136</td>
</tr>
<tr>
<td>Headgear &amp; Winder Foundations</td>
<td></td>
<td>US$ 300,672</td>
</tr>
<tr>
<td>Drilling Equipment</td>
<td>US$ 1,755,000</td>
<td>US$ 755,550</td>
</tr>
<tr>
<td>Earthmoving Equipment</td>
<td>US$ 1,880,000</td>
<td>US$ 505,967</td>
</tr>
<tr>
<td>Services</td>
<td>US$ 300,000</td>
<td>US$ 375,623</td>
</tr>
<tr>
<td>Pumps</td>
<td></td>
<td>US$ 185,555</td>
</tr>
<tr>
<td>Support Equipment</td>
<td></td>
<td>US$ 118,432</td>
</tr>
<tr>
<td>Generators</td>
<td></td>
<td>US$ 195,256</td>
</tr>
<tr>
<td>Geohydrological Drilling Programme</td>
<td>US$ 340,000</td>
<td>US$ 395,000</td>
</tr>
<tr>
<td>Shipping and Transport</td>
<td></td>
<td>US$ 273,362</td>
</tr>
<tr>
<td><strong>Total Capex</strong></td>
<td><strong>US$ 4,300,000</strong></td>
<td><strong>US$ 3,174,686</strong></td>
</tr>
</tbody>
</table>

Development stripping: US$ 2,114,000

**Total Capex**

US$ 5,288,686
An indication of the actual operating costs per month for the vertical pit is given below:

<table>
<thead>
<tr>
<th>Vertical Pit Activity</th>
<th>Cost/month</th>
</tr>
</thead>
<tbody>
<tr>
<td>HR</td>
<td>US$ 136,000</td>
</tr>
<tr>
<td>Shaft Sinkers – Hoist Service &amp; Supply</td>
<td>US$ 117,500</td>
</tr>
<tr>
<td>Drill &amp; Blast</td>
<td>US$ 79,200</td>
</tr>
<tr>
<td>Load &amp; Haul</td>
<td>US$ 193,000</td>
</tr>
<tr>
<td>Support</td>
<td>US$ 83,600</td>
</tr>
<tr>
<td>Pumping</td>
<td>US$ 18,000</td>
</tr>
<tr>
<td>Consultants</td>
<td>US$ 5,000</td>
</tr>
<tr>
<td>Ground Probe – Radar Monitor</td>
<td>US$ 32,000</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>US$ 664,300</strong></td>
</tr>
</tbody>
</table>

Tonnes mined per month (ore + waste) 42,000

Cost per tonne mined  US$ 15.82

On a cost per tonne basis, vertical pit mining (US$ 15.82) is more expensive than hard rock open pit mining in Sierra Leone (US$ 2.50). However, the benefit is realized when considering the negligible waste mining involved in vertical pit mining versus open pit mining. The stripping ratio for the vertical pit is only 0.65 whereas the stripping ratio for the open pit would have increased from 6.2 on the first cut to 13.6 on the final cut (Figure 15). The total open pit waste tonnes to be mined down to 270 m amsl would have amounted to over 5.7 million tonnes, compared with approximately 1.6 million waste tonnes actually mined over the life of the vertical pit down to 265 m amsl (including the additional 845,600 tonnes of capitalized waste development for headgear relocation and collar establishment).

Analyzing the total mining cost over the life of an open pit to 270 m amsl and what was actually achieved with the vertical pit down to 265 m amsl, the vertical pit was marginally more expensive than the open pit would have been due to the additional waste development that became necessary for the vertical pit collar establishment. Vertical pit vs open pit profiles are shown in Figure 15.
From a timing perspective, the initial work required for the establishment of the vertical pit collar and infrastructure was estimated to take only two months. However, the relocation of the headgear and the subsequent additional stripping delayed the work programme by six months. With the additional tonnages to be mined, the fleet of earthmoving equipment at No. 2 Pipe was reassigned to No. 1 Pipe and mining of No. 2 Pipe placed on hold. The mine plan for the No. 1 Pipe was modified to create a temporary ramp into the No. 1 Pipe orebody to allow for continued production while the vertical pit collar was being established. In October 2005, the hoist and winder were commissioned and the transition to full-scale vertical pit mining was made.

6 CONCLUSIONS – Vertical pit mining has been shown to be an economically viable alternative to open pit and underground mining methods on small kimberlite pipes. The factors critical to the successful implementation of this technique are the following:

- Geology – The geometry of the orebody must be known;
- Host rock – An imperative for the design is a full geotechnical appreciation of the host rock and the groundwater conditions;
- Size and Shape – Variances in size and shape can be problematic particularly with the development of overhangs;
- Economic factors – required depletion rate, cost per tonne, revenue per tonne;
- Implementation – Design specifications should be strictly adhered to with efficient quality control; and
- Geotechnical – Ongoing geotechnical input including mapping and logging is a pre-requisite for success.

Production from the from No. 1 Pipe from February 2005 until August 2007 provided sufficient information on the geology, grades, diamond values, operating costs and other economic factors in order to complete the underground mining feasibility studies. The Koidu Kimberlite Project is about to enter the next phase of development for the establishment of underground
mining operations, exploiting both No. 1 Pipe and No. 2 Pipe, as well as the dyke zones and small blows that have been identified.

7 REFERENCES

