TECHNICAL REPORT

On the

KUMTOR GOLD PROJECT

KYRGYZ REPUBLIC

for

CENTERRA GOLD INC.

Effective Date
September 30th, 2012

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Report Filing Date
December 20th, 2012

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1 SUMMARY

1.1 Kumtor Gold Project

The Kumtor Gold Project (Kumtor Project) located in the Kyrgyz Republic originated in 1992 when Cameco Corporation (Cameco), while pursuing uranium prospects in the Kyrgyz Republic, was presented with an opportunity to follow up on the discovery of gold mineralization at Kumtor in 1978 and subsequent extensive exploration work by the USSR Ministry of Geology when the Kyrgyz Republic was part of the former Soviet Union. Centerra Gold Inc. (Centerra), which became a public company in 2004, holds a 100% interest in the Kumtor Project through its wholly-owned subsidiary, Kumtor Gold Company CJSC (KGC).

The Kumtor Project is comprised of a Central Deposit (historically often referenced as the Central Pit or Kumtor Pit) and a series of smaller satellite deposits, the Sarytor Deposit, Southwest Deposit and Northeast Deposit. Open-pit mining since 1996 has concentrated on the Central Deposit, with a smaller tonnage mined from the Southwest Deposit. Mineral reserves to be extracted by open-pit mining have been estimated for the Central, Southwest and Sarytor Deposits as presented in this report, and additional mineral resources that may be mined by open pit have been estimated for the Central, Southwest, Sarytor and Northeast Deposits. Within the Central Deposit, parts of the high-grade Stockwork and SB Zones are located directly beneath the Life of Mine (LOM) pit design of the Central Pit and are considered potential underground mining targets by Centerra.

Since late 1996, the Kumtor mill has produced approximately 8.5 million ounces of gold from 85 million tonnes of ore with an average gold head grade of 3.9 grams per tonne (g/t) over a sixteen-year period.

As at September 30, 2012, the estimated Kumtor Project proven and probable mineral reserves from the Central, Southwest, and Sarytor pits and ore stockpiles were 93.1 million tonnes containing 9.7 million ounces of gold at an average gold grade of 3.3 g/t. Based on these mineral reserves, the new open-pit life-of-mine plan has been updated with open-pit mining extending to 2023 and milling operations concluding in 2026.

As at September 30, 2012, the estimated Kumtor Project measured and indicated open pit mineral resources in addition to the proven and probable mineral reserves for the Central, Southwest and Sarytor deposits are 34.1 million tonnes containing 2.5 million ounces of gold at an average gold grade of 2.3 g/t tonne; the estimated open pit inferred resources for the Central, Southwest, Sarytor and the Northeast deposits are 9.7 million tonnes containing 746 000 ounces at an average gold grade of 2.4 g/t.

The Stockwork Zone accounted for an estimated additional indicated mineral resource potentially mineable by underground mining methods of 351 000 tonnes containing 121 000 ounces of gold at an average gold grade of 10.7 g/t. Additional inferred mineral resources potentially mineable by underground methods in the Stockwork and SB Zones are estimated to amount to 5.2 million tonnes containing 1.9 million ounces of gold at an average gold grade of 11.1 g/t.
At the Kumtor Project, mineral reserves have increased by approximately 58% compared to the 2011 year-end as a result of:

- Continued successful exploration drilling which has extended the known mineral resources of the SB Zone over the last years;
- Expansion of the open pit to recover a larger proportion of the increased mineral resources of the SB Zone by open-pit mining. This includes the incorporation into the new LOM plan of low-grade resources below the cut-off grade for underground mining;
- The pit expansion is supported by the necessary early mining of additional ice and waste in 2012;
- The higher gold-price assumption of $1,350 per ounce compared to $1,250 per ounce assumed for the December 31, 2011 estimate had minimal impact on the expansion but did help offset increasing operating costs, and
- The expansion results in an increased project net present value (NPV) compared to the NPV of the LOM plan formulated at the end of 2010.

This most recent estimate of the Kumtor Project mineral reserves continues the trend over the last number of years whereby the successful in-pit exploration combined with a higher gold price has resulted in annual reserve increases since 2007, despite the annual production of more than 0.5 million tonnes of ore. In the southwestern part of the Central Deposit, further additions to the reserves will no longer be possible due to adverse topography, and the additional mineral resources potentially mineable by the Central Pit are now located in the central and northeastern parts of the Central Deposit.

1.2 Arrangements with the Government of the Kyrgyz Republic

Kumtor Gold Company (KGC), holds Centerra’s 100% interest in the Kumtor Project which is operated by the Kumtor Operating Company CJSC (KOC). Both KGC and KOC are incorporated in the Kyrgyz Republic and are wholly-owned subsidiaries of Centerra.

Centerra became a publicly-listed company on the Toronto Stock Exchange in June 2004 following the transfer to Centerra of certain gold assets, including the Kumtor Project, previously held by the Government of the Kyrgyz Republic (the Government) and Cameco Gold Inc., a wholly-owned subsidiary of Cameco.

In April 2009, the Government, Cameco and Centerra, entered into an Agreement on New Terms (ANT) for the Kumtor Project. As a result, the parties entered into restated project agreements (including the Restated Concession Agreement, the Restated Investment Agreement, the Restated Gold and Silver Sales Agreement and the Restated Shareholders Agreement) to govern the Kumtor Project, which agreements incorporated the provisions of the ANT and settled
certain outstanding disputes related to the Kumtor Project. Pursuant to the terms of the ANT, Centerra issued 18,232,615 shares from its treasury to Kyrgyzaltyn JSC (Kyrgyzaltyn), a state-owned entity of the Kyrgyz Republic.

The Restated Concession Agreement gives KGC the exclusive rights to all minerals within an area of approximately 26,000 hectares centered on the Central Deposit and with an expiry date of December 4, 2042 (Concession Area). All of the deposits and prospects outlined in this report, the current and future waste dumps and the processing plant and current tailing management facility are located within the Concession Area.

The Restated Investment Agreement provides that the Government will support further and additional exploration activity by Centerra in the Kyrgyz Republic by inviting it to consider opportunities to acquire additional exploration and mining licenses. As of June 6, 2009, when the Restated Concession Agreement came into effect, the mining and exploration licenses and associated agreements then in existence terminated and were superseded by the Restated Concession Agreement.

On December 30, 2009, Cameco disposed of 88,618,472 common shares of Centerra by means of a public offering through a syndicate of underwriters. On the same date, Cameco also transferred an additional 25,300,000 shares of Centerra to Kyrgyzaltyn pursuant to the terms of the Restated Shareholders Agreement. With the completion of these transactions, Kyrgyzaltyn now owns approximately 33% of Centerra with the balance being held by public shareholders.

1.3 Property Location and Description

The Kumtor Project is located in the Kyrgyz Republic, one of the independent successor states of the former Soviet Union, some 350 kilometres to the southeast of the Kyrgyz capital of Bishkek and about 60 kilometres to the north of the international boundary with the People's Republic of China, in the Tien Shan Mountains, at 41º 52' N and 78º 11' E.

Access to the Kumtor Project site is by the main road from Bishkek to Balykchy, a distance of 180 kilometres. Balykchy is located on the western shore of Lake Issyk-Kul at an elevation of 1,600 metres and hosts the marshalling yard for the project. The next leg follows a secondary road along the south shore of the lake to the settlement of Barskaun, a distance of 140 kilometres. The final 100 kilometres into the Tien Shan Mountains to reach the Kumtor Project site is on a narrow winding road that climbs to an elevation of 3,700 metres through 32 switchbacks of the Sary-Moynuk Pass before proceeding eastward on a plateau through which the Kumtor river and other seasonal rivers flow. The entire road trip takes approximately 5 hours to complete.

The mill facility is situated in alpine terrain at an elevation of 4,016 metres, while the highest mining activity occurs above 4,400 metres. The main camp, administration and maintenance facilities are at approximately 3,600 metres elevation. Local valleys are occupied by active glaciers that extend down to elevations of 3,800 to 3,900 metres, and undisturbed permafrost in the area can reach a depth of 250 metres. The region is seismically active as a result
of the continuing convergence between the Indian and Eurasian plates, but the Concession Area has a relatively sparse history of seismic activity. All facilities, including the process plant and tailings storage dam, have been designed in accordance with recommended seismic standards for the area.

1.4 Kumtor Project Geology and Mineralization

The Kumtor Project deposits occur in the middle Tien Shan metallogenic belt, a Hercynian fault and thrust belt in Central Asia that extends from Uzbekistan in the west through Tajikistan and the Kyrgyz Republic into northwestern China and hosts a number of important gold deposits, among them Muruntau, Zarmitan, Jilau and Kumtor.

The mine geology is dominated by several major thrust slices and fault zones which strike northeasterly and dip to the southeast at varying but moderate angles. Each thrust sheet contains older rocks than the sheet it structurally overlies. The slice hosting the gold mineralization is composed of meta-sediments of Vendian age (youngest Proterozoic or oldest Palaeozoic) that are strongly folded and schistose. In most areas, the Kumtor Fault Zone (KFZ), a dark-grey to black, graphitic gouge and schist zone forms the footwall of this structural segment. The KFZ has a width of up to several hundred metres. The adjacent rocks in its hanging wall are strongly affected by folding, shearing and faulting for a distance of up to several hundred metres. The rocks in the structural footwall of the KFZ are Cambro-Ordovician limestone and phyllite, thrust over Tertiary sediments of possible continental derivation which in turn rests, with apparent unconformity, on Carboniferous clastic sediments.

The structural geology has evolved through four main deformation events pre-Carboniferous to Tertiary in age. Recent advances in understanding of the structural geology have improved the understanding of some of the geotechnical issues affecting the Central Pit.

Gold mineralization occurs where the Vendian sediments have been hydrothermally altered and mineralized, an event that has been dated as late Carboniferous to early Permian. Gold mineralization is developed at varying intensities over a distance of more than twelve kilometres, with the Central Deposit being the most important. Other known occurrences along the mineralized trend are the Southwest Deposit which was first mined in 2006 to 2008, the Sarytor Deposit, for which a mineral reserve was estimated for the first time at the end of 2006, and the Northeast Deposit for which an estimate of inferred mineral resource was first produced in February 2011. Additional centers of mineralization are known from the Akbel, Muzdusuu and Bordoo prospects, but no mineral resources have yet been estimated in these areas.

Mineralization took place in four main pulses with the mineralization being most intense, and the gold grade the highest, where the metasomatic activity was continuous through phases two and three. Substantial volumes affected by such activity are represented by the Stockwork and SB Zones of the Central Deposit, which contain the most significant accumulations of high-grade mineralization. Native gold and gold-bearing minerals such as tellurides occur as very fine inclusions in pyrite, with an average size of only 10 microns, which accounts for the partly refractory nature of the ore. However, the fine grain size of the gold also renders assay results for this mineralization relatively reliable, with only a small nugget effect. Post-ore faults, in addition
to being of geotechnical significance, often carry significant quantities of graphite, and other
carbonaceous components which constitute the source for the preg-robbing character of some of
the mineralization.

1.5 Geotechnical Issues

1.5.1 Bedrock Slope Stability

Operations at the Central Pit have been negatively affected as a result of two substantial
failures of the bedrock high wall that forms the northeastern limit of the Central Pit as well as less
severe deformations that occurred in other parts of the pit.

The first high wall failure on July 8, 2002 resulted in the loss of a life, and a temporary
suspension of operations, and led to a shortfall in 2002 production because the high-grade
Stockwork Zone was rendered temporarily inaccessible. A second failure of similar magnitude
occurred on July 13, 2006, in an area above the Stockwork Zone that was planned to be mined in
2006 and 2007. Mining from the area has since been deferred and was concentrated on the
southern part of the Central Pit to exploit the high-grade SB Zone discovered in 2005.

Following the second ground wall movement, KOC, Golder Associates and Centerra
continued to assess the causes of the pit wall failure and have developed remedial and long-term
pit slope design criteria that would reduce the possibility of a recurrence. This work has provided
insight into why the high wall failures occurred.

In the 2009 year-end mineral reserve estimate and life-of-mine plan, the northeastern high
wall design was revised from a slope angle of 36° to a slope angle in the order of 30°. The
current design assures that the known wedges that gave rise to the two failures will be mined out.
The current design is also expected to prevent the exposure of the next set of wedges. Since 2006,
the inactive high wall has been stable based on the monitoring data collected from approximately
100 prisms. The safety of the highwall design depends on the state of its depressurization. If the
highwall is not or cannot be sufficiently depressurized and proves to be unstable at the current
slope angles, the mineral reserves and LOM plan for this part of the Central Pit would be
adversely affected. However, that part of the Lysii glacier providing melt water to the northeast
highwall will be mined out in 2014 and 2015 according to the new LOM mine plan, mitigating
against most surface water entering the northeastern high wall and KOC currently plans a drilling
depressurization program in the area.

The southern part of the Central Pit which exploits the SB Zone has undergone several
revisions to its design bedrock slope angles. The slope angles of 36° originally specified in 2006
were revised to 30° for most sectors, with only a few retaining an angle of 36°. The revisions
were required as a result of ravelling and deformation of the rock slopes during previous mining
activities and were determined using a substantial amount of geotechnical drilling completed
after 2006. This has shown that the structural features causing slope instability dip into the pit at
relatively shallow angles (more or less parallel to the pit slopes) in two major sectors
(northwestern and eastern walls). The pit walls are now designed to avoid undercutting of these
structures. The safety of the walls depends on the accuracy of the structural geological model,
which is being continuously refined and updated, as well as the ability to depressurize water-bearing faults and structures. Additional geotechnical drilling will cover substantial portions of the southeastern and eastern walls into which the pit will expand, and more slope-angle revisions may be required. It is important to note that one degree change in the overall slope angle results in a 15-metre vertical move of the bottom elevation of the pit. Any downward revision of the slope angles will impact the reserves and/or economics of the project.

1.5.2 Historical Waste Dumps, Glacial Ice and Moraine

A substantial amount of waste rock from the southern part of the Central Pit had been deposited directly onto the Davidov glacier prior to the discovery of the SB Zone in 2005. Starting in 2007, operations at the SB-Zone section of the Central Pit have been adversely affected as a result of significant creeping into the pit of glacial ice and waste dumps riding on the ice, with these movements continuing to date. The section of the pit impacted is approximately 1 100 metres long. Creep rates are being monitored and are generally above 5 mm/h and have averaged 20 mm/h (15 m/month) during the summer months. This movement began to accelerated late in 2011 with movements increasing to as high as 80 mm/h in March 2012. This forced the suspension of waste-rock mining above the SB Zone to allow removal of the moving ice and loose waste. As a result, the access to ore was delayed and milling operations were shut down for a seven-week period in 2012 from late July to late September after all existing stockpiles had been consumed.

The situation is a unique geotechnical problem and the main effect has been the delay of mining of some of the high-grade parts of the SB Zone to allow removal of the waste and ice, and an increased overall stripping ratio of this part of the Central Pit. After extensive investigations, it is now accepted that the ice/waste dump mass is essentially sliding on the upper surface of the basal till below the Davidov Glacier. Most of the melt water is also running along this surface and likely contributes to the creep rate by providing lubrication. The slope angle of the basal till layer and the weight of the waste on the ice are other factors contributing to the creep movement while the till itself, which has proved to be dry, is not.

Current information indicates that creeping of ice and some remaining waste in this section of the pit wall will continue for the next several years. This will be managed by the continued unloading of the waste dump material and the positive impacts of increased dewatering efforts along the ice-till contact in this area of the Central Pit. To manage the future anticipated creep movement, the updated LOM plan allocates significant mining capacity to four “unloading cutbacks” for the years 2012 to 2015 for the removal and management of historical waste dump and glacial ice in this section of the Central Pit. Extra mining capacity has been allowed to deal with the moving ice in the future. This remediation plan results in a more stable and achievable gold production profile over the life-of-mine for the Central Pit but will result in an overall extension in the timeline to complete mining the SB Zone.

Mining of the enlarged Central Pit that contains the majority of the mineral reserves of the Kumtor Project will require the removal of additional glacier ice (without waste heaped on it) which will continue to advance toward the pit. The advance rate of the ice is currently not well
known, but the LOM plan allows for additional mining equipment to address this issue. An initial pushback of the glacier ice by 185 metres involving a significant volume of ice is planned as an initial step, followed by ice mining annually the amount of which will depend on the glacier advance rate. Ice advance monitoring installations have been updated and improved.

### 1.6 Mining and Milling Operations

Mining operations use conventional open-pit mining methods. The Central Pit has been an active mining area since the start of the Kumtor Project. Total material mined between January 1, 2011 and September 30th, 2012 was approximately 259 million tonnes, or 400 000 tonnes per day. The overall waste to ore ratio during this period was 44 to 1; however this ratio is uncharacteristic for the historical pit production and the life of mine going forward as a result of the significant ice unloading that was required during 2012.

The Central Pit has had the benefit of a favourable topographical layout where the top mining elevation in the current ultimate pit design is at about 4 460 metres at the northeast high wall, while the deepest part of the final pit excavation will be at 3 500 metres in the bottom of the SB Zone located in the southwest end of the Central Deposit. The crushing plant to which the ore is delivered is at about 4 050 metres, and ore transport is thus downhill for the upper portion of the ore body, and has a maximum uphill vertical haul of 550 metres for the lowest portion of the ore body.

Mining is done on ten-metre benches at the Central Pit with a variety of large-scale mining equipment. Eight rotary-percussion rigs drill 300-mm holes on a six to seven-metre pattern. Blasting is with ammonium nitrate with fuel oil (ANFO) and special emulsion for wet holes. Loading is accomplished by 16 hydraulic shovels and three front-end loaders. The current haulage fleet of 96 trucks have capacities ranging from 90 to 200 tonnes and their movement is monitored and directed by an updated GPS-based tracking system. Starting in late 2012, an additional 27, two hundred tonne capacity haulage trucks will be added to the fleet before the smaller and older haulage trucks are decommissioned from mine production starting in 2014.

The mill facility was originally designed with a capacity to process 4.8 million tonnes of ore per year. The mill throughput for 2011 was 5.8 million tonnes or a nominal capacity of 15 900 tonnes per day and is scheduled to be increased and a result of a planned expansion to a nominal 18 400 tonnes per day starting in 2016. This expansion will require a capital expenditure of 33.5 million dollars.

The plant flowsheet reflects the fine-grained nature of the gold and its intimate association with pyrite, and consists of crushing, grinding, pyrite flotation, and double re-grinding of the flotation concentrate. Two separate carbon-in-leach (CIL) circuits extract the gold from the re-ground concentrate and from the flotation tails, with final gold recovery accomplished by electro-winning and refining. The historical gold recovery has been 79% which is expected to be achieved for the current LOM plan as well.
The mill feed is managed through a number of stockpiles that allow blending for grade smoothing and metallurgical character as forecast by the grade-control data. With the exception of the summer of 2012, the stockpiles have allowed milling operations to continue during mining disruptions caused by geotechnical issues in the open pit. The updated Life of Mine plan will begin to increase ore stockpiles in 2014 resulting in a reduced production risk and a more consistent annual and quarterly production profile.

1.7 Mineral Resources and Mineral Reserves

Mineral reserves, along with updated open-pit LOM plan and additional mineral resources for the Kumtor Project have been estimated by Dan Redmond, P. Geo., Director of Mining, Technical Services Department of Centerra and John Baker, Mine Manager, KOC as of September 30, 2012 and are outlined in Table 1.

Table 1 Kumtor Project Mineral Reserves and Resources
(As of September 30, 2012 – Tonnes and Ounces in Thousands)

<table>
<thead>
<tr>
<th>Mineral Reserves</th>
<th>Proven Mineral Reserves</th>
<th>Probable Mineral Reserves</th>
<th>Total Proven and Probable Mineral Reserves</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes</td>
<td>Grade Au (g/t)</td>
<td>Contained Gold (oz)</td>
</tr>
<tr>
<td></td>
<td>Tonnes</td>
<td>Grade Au (g/t)</td>
<td>Contained Gold (oz)</td>
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<tr>
<td></td>
<td>Tonnage</td>
<td>Grade Au (g/t)</td>
<td>Contained Gold (oz)</td>
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<tr>
<td></td>
<td>Gold (oz)</td>
<td>Gold (oz)</td>
<td>Gold (oz)</td>
</tr>
<tr>
<td></td>
<td>Mining</td>
<td>Method</td>
<td>Method</td>
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<tr>
<td></td>
<td>233</td>
<td>1.8</td>
<td>13</td>
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</tbody>
</table>

Additional Indicated Mineral Resources

<table>
<thead>
<tr>
<th>Measured Mineral Resources</th>
<th>Indicated Mineral Resources</th>
<th>Total Measured and Indicated Mineral Resources</th>
</tr>
</thead>
<tbody>
<tr>
<td>Tonnage</td>
<td>Tonnage</td>
<td>Tonnage</td>
</tr>
<tr>
<td>Grade Au (g/t)</td>
<td>Grade Au (g/t)</td>
<td>Grade Au (g/t)</td>
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<tr>
<td>Contained Gold (oz)</td>
<td>Contained Gold (oz)</td>
<td>Contained Gold (oz)</td>
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<tr>
<td>Mining</td>
<td>Mining</td>
<td>Mining</td>
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<tr>
<td></td>
<td></td>
<td>Method</td>
</tr>
<tr>
<td>21 709</td>
<td>12 426</td>
<td>34 135</td>
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<tr>
<td>2.3</td>
<td>2.3</td>
<td>2.3</td>
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<tr>
<td>1 600</td>
<td>929</td>
<td>2 529</td>
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<tr>
<td></td>
<td></td>
<td>Open Pit</td>
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<tr>
<td>-</td>
<td>351</td>
<td>351</td>
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<tr>
<td>-</td>
<td>10.7</td>
<td>10.7</td>
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<tr>
<td>-</td>
<td>121</td>
<td>121</td>
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<tr>
<td>-</td>
<td></td>
<td>Underground</td>
</tr>
</tbody>
</table>

Additional Inferred Mineral Resources

<table>
<thead>
<tr>
<th></th>
<th>Tonnage</th>
<th>Grade Au (g/t)</th>
<th>Contained Gold (oz)</th>
<th>Mining Method</th>
</tr>
</thead>
<tbody>
<tr>
<td>Central, Sarytor, NE &amp; SW Deposits</td>
<td>9 697</td>
<td>2.4</td>
<td>746</td>
<td>Open Pit</td>
</tr>
<tr>
<td>Stockwork Underground</td>
<td>2 002</td>
<td>11.0</td>
<td>705</td>
<td>Underground</td>
</tr>
<tr>
<td>SB Underground</td>
<td>3 193</td>
<td>11.2</td>
<td>1 150</td>
<td>Underground</td>
</tr>
</tbody>
</table>

Mineral resources do not have demonstrated economic viability. Additionally, it cannot be assumed that inferred mineral resources will be upgraded to a higher mineral resource category. The eventual conversion of additional underground resources into reserves will require trial mining followed by a feasibility study due to the challenging ground conditions.
For the estimation of the mineral resources and mineral reserves of the Central Deposit, a new block model identified as the KS-13 model was developed, including all data available as of September 30, 2012. For the other deposits, existing block models SW-2 (for the Southwest Deposit), SR-2 (for Sarytor) and NE 1 (for the Northeast Deposit) were used.

Details of the parameters used and assumptions made for the estimation of the mineral resources and reserves are detailed in Sections 14 and 15 of this report. The main assumptions are:

- The estimates assume a gold price of $1 350 per ounce;
- The mineral reserves and resources are reported at a cut-off grade of 0.85 g/t Au for open-pit mining at the Central Deposit and at a cut-off grade of 1.0 g/t Au for open-pit mining at the Southwest, Sarytor and Northeast Deposits; and
- Underground mineral resources are reported at a cut-off grade of 6.0 g/t Au.

The block models used for the resource and reserve estimates in Table 1 have been tested against actual production from the Central and Southwest Deposits for the period January 1, 2004 to September 30, 2012. The comparison shows that grade and tonnage variances for annual mining volumes have been in the order of ±5% to ±10%, and that there was no grade or tonnage bias in the reserve estimates.

1.8 Reconciliation with Year-End 2011 Mineral Reserve Estimate

After accounting for processing of approximately 3.2 million tonnes with an average gold grade of 1.7 g/t in the first nine months of 2012 (172 000 ounces of contained gold), the September 30, 2012 mineral reserve surpasses the December 31, 2011 estimate by 36.6 million tonnes containing an additional 3.6 million ounces of gold at an average grade of 3.1 g/t. The main components responsible for this increase are:

- Exploration drilling in 2012 at the Kumtor Project has continued to add both mineral reserves and resources in the southwest extension to the SB Zone. A sizeable portion of this additional mineralization (4.7 million tonnes at 1.6 g/t gold) falls within the new open pit design and has been included in the probable mineral reserves;
- The expanded Central Pit has also captured approximately 3.0 million tonnes of high-grade underground resources with an average grade of 13.8 g/t gold. The large majority of this addition resides in the SB Zone, a small tonnage is in the Stockwork Zone; and
- The largest addition to the new final Central Pit was 28.9 million tonnes at an average gold grade of 2.2 g/t. These had previously been reported as additional measured and indicated mineral resources mineable by open pit. And were located below the previous pit bottom.
1.9 Kumtor Project Life of Mine Plan and Projected Cash Flow

The estimation of mineral reserves for the Kumtor Project as of September 30, 2012 included the development of a detailed life-of-mine (LOM) plan which demonstrates the technical feasibility of mining the stated mineral reserves and the ability of the mining operation to provide continuous feed to the mill. The LOM plan together with annual cost estimates is summarized in Table 2. Note that the line item “Revenue Based Taxes” constitutes 14% of gross revenue and is thus variable with the gold price.

<table>
<thead>
<tr>
<th>Table 2 Kumtor Project LOM Plan, Operating and Capital Costs Forecast</th>
</tr>
</thead>
<tbody>
<tr>
<td>PRODUCTION</td>
</tr>
<tr>
<td>Mining Operating (t x 1000)</td>
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<tr>
<td>Mining Capital Prestrip (t x 1000)</td>
</tr>
<tr>
<td>Mining Total (t x 1000)</td>
</tr>
<tr>
<td>Milling (t x 1000)</td>
</tr>
<tr>
<td>Gold Production (oz x 1000)</td>
</tr>
<tr>
<td>DIRECT OPERATING COSTS</td>
</tr>
<tr>
<td>Mining Operating ($ x 1000)</td>
</tr>
<tr>
<td>Milling ($ x 1000)</td>
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<tr>
<td>Administration (2) ($ x 1000)</td>
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<tr>
<td>Refining and Other Management Fee ($ x 1000)</td>
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<tr>
<td>Total Direct Operating Costs ($ x 1000)</td>
</tr>
<tr>
<td>Direct Cash Cost per Ounce ($/oz)</td>
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<tr>
<td>OTHER PRODUCTION COSTS</td>
</tr>
<tr>
<td>Mining Capital Pre-Strip ($ x 1000)</td>
</tr>
<tr>
<td>Revenue Based Taxes ($ x 1000)</td>
</tr>
<tr>
<td>Total Other Production Costs ($ x 1000)</td>
</tr>
<tr>
<td>Total Direct and Other Cash Costs Per Ounce ($/oz)</td>
</tr>
<tr>
<td>CAPITAL COSTS</td>
</tr>
<tr>
<td>Open Pit Sustaining Capital ($ x 1000)</td>
</tr>
<tr>
<td>Open Pit Growth Capital ($ x 1000)</td>
</tr>
<tr>
<td>Total Capital ($ x 1000)</td>
</tr>
<tr>
<td>UNIT COSTS</td>
</tr>
<tr>
<td>Mining Operating Costs $/t mined</td>
</tr>
<tr>
<td>Mining Capital Pre-strip $/t mined</td>
</tr>
<tr>
<td>Milling $/t milled</td>
</tr>
<tr>
<td>Administration $/t milled</td>
</tr>
</tbody>
</table>

(1) October 1 to December 31, 2012
(2) Includes significant direct logistic support costs such as camp costs and delivery of consumables to site.
The new LOM plan extends the open-pit mining to 2023 and milling operations of the Kumtor Project to the end of 2026 as a result of expanding the Central Pit to recover a larger portion of the expanded SB Zone resources. The LOM plan is based only on open-pit mineral reserves and has no provision for production from any underground mining activities.

Pre-stripping of the Stockwork-Zone high wall is scheduled to begin in December 2015 and will release significant levels of ore by mid-2018. Pre-stripping of the Sarytor pit will begin in 2018 and mining of the Sarytor and Southwest pit mineral reserves will be completed in 2023.
Overall mining production rates will increase from the current 400,000 tonnes of ore and waste per day to more than 500,000 tonnes of ore and waste per day in for the years 2013 to 2021 before starting to decline in 2021. The ramp-up in the mining production rate over the next two years has been partially met by purchases of new mining equipment in 2010-2012 but will require an additional $100 million of capital over the next two years for additional new larger mining equipment to expand the overall mining capacity and to replace older mining equipment that has now reached the end of the effective operational life.

### 1.10 Economic Analysis

At the assumed gold price of $1350 per ounce, the open-pit LOM plan has been used to project the net cash flow for the open pit operations for the period of October 1, 2012 to 2026. Total gross revenue from the sale of gold and minor silver credits amounts to $10.7 billion while direct operating costs are estimated to be $3.3 billion or $422 per recovered ounce of gold. After further amounts spent on capital pre-stripping of approximately 1.7 billion, capital investments of around $726 million, and the payment of revenue based taxes of $1.5 billion in accordance with the Restated Investment Agreement, the resulting undiscounted net cash flow for the Kumtor Project totals approximately $3.5 billion or $1.9 billion at a 8% discount rate. Total cost per ounce of recovered gold including all operating costs, capitalized pre-stripping, capital additions and revenue based taxes (at a $1350 gold price) amounts to $917 over the projected mine life.

The total estimated open-pit LOM capital costs of $2.4 billion include $1.7 billion for pre-stripping, $557 million for sustaining capital, mostly related to the maintenance or replacement of current open-pit mining equipment, and $169 million of expansion capital, mostly related to the purchase of additional mining equipment, relocation of some surface facilities, expansion of the processing plant and of the tailings management facility. Costs related to ongoing exploration estimated to be $16 million for 2012/2013 are not included in this cash flow projection.

A gold price of $843 per ounce is required to achieve neutral net cash flow under the current open-pit LOM while meeting all anticipated requirements for operation, capital expenditures and taxes, but excluding exploration expenditures. The 14% gross revenue taxes outlined in Section 22.2 have been included in this break-even net cash flow estimate but the amount payable for the break-even case is naturally lower than those outlined in Table 2 due to the lower gold price.

Other than the gold price, the only parameter that could have a significant impact on mine cash flow would be an increase or a decrease in mill head grade. A 10% gold-grade variance would impact the cumulative cash flow over the period of the LOM plan by plus or minus $634 million at a constant gold price of $1,350 per ounce. However, the experience to date has been that, over a sufficiently long period of time, the mine head grade will match the projected LOM grade within a few percent.
1.11 Environmental, Permitting and Social

The environmental performance of the Kumtor operation with respect to Kyrgyz and World Bank requirements is monitored by KOC annually and by outside consultants occasionally. These reviews indicate that no major infractions have occurred, but that improvements on a number of issues are possible. KOC has translated the latest such set of recommendations into an action plan that will be implemented over the next years.

The Kumtor operation requires a long list of licences and permits, many of which are issued for relatively short periods requiring frequent renewals. Included in this list are approvals of the annual mine plans, the development of an “Ecological Passport” with annual permits for emissions and discharge, three-year licences for the disposal of tailings and toxic wastes into the tailings management facility, annual to semi-annual permits for the importation and transport of hazardous substances such as blasting agents and cyanide, and an annual water use permit. While issuance of these permits is not always on time, the 16-year operating history of the mine shows that the permitting process does not interfere with the operations.

Conceptual closure provisions have to be updated every three years (the final closure plan is due two years prior to the event), and the next such update is due in 2013. For the much increased scope of the new mine plan extending the life of the operation to 2026, an opinion has been obtained from an outside consultant which concludes that the existing closure plan is not significantly impacted by the additional waste tonnages that will be produced. A trust fund has been set up for final reclamation measures. The fund currently has a balance of $11.3 million, representing nearly 40% of the current final estimated closure costs of $29.5 million.

The performance of the operation with respect to occupational health and safety is on par with Western similar mining operations, with lost-time accidents in recent years at a level of less than 0.1 per 200 000 man-hours.

The Kumtor operation currently provides employment for more than 2 700 Kyrgyz citizens at wage rates that are significantly higher than the country average. The operation is the largest private enterprise in the country and contributed nearly 12% of GDP in 2011. Direct and indirect contributions and expenditures within the Kyrgyz Republic amounted to $383 million in 2011.

1.12 Exploration

The remaining exploration targets that will be investigated during the fourth quarter of 2012 and 2013 as potential opportunities to provide mill feed to the existing operation in the future include the following:

- In-fill drilling (18 000 metres) is planned to upgrade the classification of the additional mineral resources in the SB Zone below the KS-13 design pit potentially mineable by underground mining methods;
• Deep drilling (15,000 metres) is planned for the purpose of determining the full extent of the entire SB Zone at depth for informed decision making about a possible mining approach (15,000 metres);

• Exploration possibilities exist to add to the open-pit resources in the Central Pit area (Saddle Zone) and along strike at the Southwest and Sarytor Deposits. A total of 9,000 metres of drilling is planned for these targets; and

• Additional surface drilling of 3,000 metres is planned for the Northeast Deposit.

The Karasay and Koendy licences acquired in 2009 and 2010 and located outside of the Concession Area have expired, and applications for re-issue are pending. Beyond the two concessions, Centerra commenced a regional assessment program in 2010. As a result of this work KGC submitted two license applications covering regional targets in 2010 and a third application in early 2012. Since the licensing commission of the Agency for Subsoil and Natural resources has taken no action on these applications, the 2012/2013 exploration budget has no provision for extensive work in these areas.

The total budget of $16 million for exploration in the last quarter of 2012 and all of 2013 has been set aside for these activities. This figure is not included in the expenditures used for the economic evaluation of the Kumtor Project.

1.13 Risk Factors

Based on the good reconciliation between tonnages milled and the block models used for the updated reserve estimate, there is little reserve risk for the new Kumtor LOM plan. Annual tonnage and gold grade estimates are interpreted to be predictable with a variance of ±5% to ±10% and no bias.

The understanding of the Kumtor geotechnical issues, which have resulted in repeated and costly interruptions of previous mine plans (but not in the loss of any mineral reserves), has improved significantly in the past years. However, the September 30, 2012 mineral reserves and LOM plan outlined in this report will create a much enlarged man-made excavation with wall heights reaching approximately 600 metres in many sections of the Central Pit. Additionally, the final pit wall will move to within 140 metres of the processing plant. The new pit design assumes the continued success of future remedial measures to provide slope stability consisting of the adherence to shallow pit-wall angles aided by efforts of depressurization and dewatering. Should these measures prove insufficient and if a lowering of the slope angles by only 2° is required, there would be a loss of mineral reserves and/or an increase of the waste-to-ore ratio for the remaining mine life, both with negative implications for the LOM plan and the economic performance of the project. If the grinding equipment of the processing plant were to be affected by ground stability issues, a major disruption of the LOM plan would occur.

The second geotechnical challenge facing the future Kumtor operation is the interaction of the Central Pit with the surrounding glaciers. The main tasks are to complete the unloading of the historical waste dumps and underlying ice, and the additional removal of a sufficient amount
of glacier ice to prevent the advancing Davidov glacier from entering the Central Pit until early 2018, when this part of the pit will be mined out. While the LOM plan has a contingency for additional ice tonnages to be mined, any substantial additional ice mining would negatively impact the mining of ore and the project cash flow; a loss of reserves is not anticipated. However, should Kumtor be prevented from continuing its practice of mining ice, the entire KS-13 mineral reserves and LOM plan would be at risk, leading to an early closure of the operation.

Petrov Lake, which is fed by the receding and melting Petrov Glacier and which is therefore increasing in size, is confined by a moraine which is currently stable due to its largely frozen state. It is forecast that the moraine will thaw out prior to 2050 due to continued global warming, and a sudden and potentially damaging glacial lake outflow will occur. This event has the potential to erode a section of the Kumtor tailings management facility and to damage other downstream installations. Any erosion of the tailings dam would have to be considered a serious threat. While the risk of this outflow occurring in the next ten years is considered low, this is a future event that needs to be considered for mine closure. An early warning system is currently being installed to safeguard people working in the path of a potential outflow, and remedial measures to armour and protect the tailings facility are under evaluation. The construction of a permanent spillway to lower the level of Petrov Lake by three metres is planned to commence in 2014.

A Kyrgyz State Commission was formed in July 2012 to “assess the environmental, industrial and social damage” caused by the Kumtor project, and to provide a “legal examination of agreements made on the Kumtor Project in terms of protection of the state interests.” (Government Decree as translated). There have also been four recent claims by the Kyrgyz State Inspectorate Office for Environmental and Technical Safety for a total payment of approximately $152 million relating to alleged environmental damages at the Kumtor Project.

Although both the Prime Minister and the President have indicated through the media that it is not an option, there continues to be voices within the Kyrgyz Republic demanding nationalization of the Kumtor Project. It is conceivable that the recommendations of the as yet unavailable State Commission report could include a recommendation to re-define the terms between the state of Kyrgyzstan and the Kumtor Project. The economic assumptions and projections in this report are based on the existing agreements concluded in 2009. Any changes would alter the economic outcomes of the project.

1.14 Conclusions

This review of the Kumtor Project and the September 30, 2012 mineral reserve estimate has resulted in the following main conclusions:

1. The open-pit mineral reserves at Kumtor have continued to expand since the discovery of the SB Zone in 2005. The total known Central Deposit (past production plus mineral reserves) has grown from 72 million tonnes with an average gold grade of 4.0 g/t at the end of 2004 to 164 million tonnes with an average gold grade of 3.6 g/t as of September 30, 2012;
2. The mineral reserves (tonnes and contained ounces) as of September 30, 2012 have increased by approximately 58% compared to the estimate at the end of 2011, with the average gold grade remaining constant at 3.3 g/t. Milling operations are now expected to continue until 2026;

3. A review of the production reconciliation to the latest version of the mineral reserve models shows that the models are reliable estimators of the mineral reserves (tonnes and grade) of the gold mineralization at Kumtor. The variances for annual production tonnages observed since 2005 are within industry standards. The mineral reserve estimation risk for the tonnage and grade predicted by the current LOM plan is low;

4. While mineral reserve increases over the last four years have been material, opportunities for further reserve expansions are now contained by the unfavourable topography in the southern part of the new ultimate Central Pit;

5. The Kumtor operation will continue to produce ore at a high strip ratio for most of its projected mine life, with the total annual tonnage mined in the range of 170 to 193 million tonnes for the period of 2013 to 2021 with an average waste-to-ore ratio of 19.2 to 1;

6. To accomplish this mining rate, the program of capital expenditures for additional mining equipment of the past years will continue until 2014. Re-location of surface installations due to the expansion of the Central Pit and because of the encroachment by the Davidov Valley waste dump requires additional capital expenditures until 2015. Other capital projects include the continuing expansion of the tailing facility and the expansion of the mill capacity in the years 2014 and 2015;

7. Gold production from the Central Pit has been negatively impacted on several occasions from 2002 to 2012 by geotechnical issues related to the poor quality of the host rocks resulting from the intensive and complex structural deformation in the area, and the gradual movement of sections of the historical waste dumps that had originally been placed on parts of the adjacent Davidov glacier. While the understanding and resulting remedial plans for these issues have progressed significantly, geotechnical issues remain the most significant technical risk to achieving the gold production and associated cash flow as outlined in the LOM plan;

8. There are no indications of geotechnical issues for the smaller Sarytor and Southwest pits that will be mined in the years 2018 to 2023;

9. Since 2007, Centerra has undertaken a sizeable program of underground development and drilling to evaluate the potential for a possible underground mining operation at Kumtor that would exploit high-grade ore shoots below the final pit bottom to augment the open-pit mining. As of September 30, 2012, approximately $190 million had been spent on the underground project which included costs related to the establishment of portal facilities, underground mining equipment and the completion of approximately 3 000 meters of ramp development in two declines. No underground test mining has yet been completed
in what are very difficult ground conditions. Without the planned additional drilling and a comprehensive test mining program, followed by a feasibility study, conversion of these resources into reserves will not be possible.

10. Kumtor has been the object of a State Commission formed in July 2012 with the intention to “assess the environmental, industrial and social damage” caused by the Kumtor project and to provide a “legal examination of agreements made on the Kumtor Project in terms of protection of the state interests.” It is possible that the State Commission final report expected to be issued in January 2013 will recommend changes to the existing agreements that could put additional economic constraints on the operation. These potential constraints are currently unknown and are not reflected in the economic projections made in this report which is based on the existing agreements concluded in 2009.

1.15 Recommendations

The authors of this technical report make the following recommendations:

1. The authors endorse the plans by Kumtor to implement additional geotechnical drilling and studies to ascertain that the pit-slope assumptions used for the KS-13 Central Pit are sufficient to prevent any large slope failures in general, and to ensure that the processing plant will not be impacted by the expanding pit;

2. The program of geotechnical analysis of the creeping area of the historical waste dumps deposited on glacier ice and the glacial ice itself should be continued with the goal of further improvements in the understanding of the mechanisms for the movement. These investigations should continue to include the analysis of the movement of the existing and accumulating waste dumps;

3. Monitoring of the hydraulic pressure of all of the Central Pit walls and particularly of the high walls as recommended by the geotechnical consultants should be continued and completed, including the drilling of depressurization wells where necessary;

4. With the physical limits of open-pit mining having been reached in the southern part of the Central Pit with the new LOM plan based on the KS-13 block model, Centerra should complete studies to determine the economic conditions required for the conversion of all or some of the additional mineral resources mineable by open pit into mineral reserves. These investigations would include the determination of incremental strip ratios for the additional resources, capital requirements (the cost of additional waste mining as well as of any additional equipment purchases), and the gold price that would support such conversion. Only additional mineral resources mineable by open pit that satisfy these conditions should be publicly reported in future;

5. Given the success rate of previous exploration programs at the Kumtor Project and given the remaining exploration possibilities, the authors support the ongoing exploration efforts with a financial commitment of $16 million for 2012/2013 for surface and underground
drilling exploration. Further exploration expenditures will likely be required beyond 2013 on the additional new exploration licenses in subsequent years, but details of these, and their justification, are contingent on the results of the 2012/2013 program;

6. The future conversion of mineral resources mineable by underground methods into reserves and subsequent inclusion in the Kumtor LOM plan should be made dependent on the successful completion of the additional drill program and on a comprehensive underground test mining program that determines the technical and economic parameters of underground mining for a dedicated feasibility study; and

7. The authors endorse the recommendations made by Bloom (2012) for a modest capital program to improve the space and the equipment of the mine assay laboratory.
2 INTRODUCTION

2.1 Background

A technical report compliant with National Instrument 43-101, Standards of Disclosure for Mineral projects (NI 43-101) for the Kumtor Project has been commissioned by Centerra Gold Inc. (Centerra) to be completed by Strathcona Mineral Services Limited (Strathcona) and Centerra. The current technical report provides an update of the technical report dated March 22, 2011 (Redmond et al., 2011) to document important changes to the Kumtor Project mineral resources and reserves. This technical report also provides an update to and supersedes the Kumtor Project mineral resources and reserves as estimated by Centerra as of December 31, 2011 which were published in a Centerra press release dated February 9, 2012.

Kumtor Gold Company CJSC (KGC), a wholly-owned subsidiary of Centerra, holds Centerra’s interest in the Kumtor Project which is operated by Kumtor Operating Company CJSC (KOC). KOC is incorporated in the Kyrgyz Republic and is also a wholly-owned subsidiary of Centerra. Centerra became a publicly-listed company on the Toronto Stock Exchange in June, 2004 following the transfer to Centerra of certain gold assets, including the Kumtor Project, previously held by the Government of the Kyrgyz Republic (the Government) and Cameco Gold Inc., a wholly-owned subsidiary of Cameco Corporation (Cameco).

In December 2003, the Government, Cameco and Centerra, among others, entered into various project agreements setting out the terms and conditions applicable to the operation and development of the Kumtor Project. These agreements replaced the original agreements concluded in 1995 as part of the original feasibility activities. In August 2007, the Government, Cameco and Centerra entered into framework agreements to provide for the amendment and restatement of such project agreements, but the framework agreements were not ratified by the Parliament of the Kyrgyz Republic within the time frame agreed to by the parties and therefore expired.

In April 2009, the Government, Cameco and Centerra, among others, entered into an Agreement on New Terms for the Kumtor Project (ANT) as a result of which the parties entered into restated project agreements to govern the Kumtor Project, which agreements incorporated the provisions of the ANT and settled certain outstanding disputes related to the Kumtor Project. Pursuant to the terms of the ANT, Centerra issued 18,232,615 shares from its treasury to Kyrgyzaltyn JSC (Kyrgyzaltyn), a state-owned entity of the Kyrgyz Republic.

On December 30, 2009, Cameco disposed of its remaining 88,618,472 common shares of Centerra by means of a public offering. On the same date, Cameco also transferred an additional 25,300,000 shares of Centerra to Kyrgyzaltyn. With the completion of these transactions, Kyrgyzaltyn now owns approximately 33% of Centerra with the balance held by public shareholders.
Following the completion of the transactions contemplated by the ANT, the Kumtor Project is now governed by:

1. A Restated Investment Agreement entered into as of June 6, 2009, among the Government, on behalf of the Kyrgyz Republic, Centerra, KGC and KOC (Restated Investment Agreement) setting out the terms and conditions applicable to the operation and development of the Kumtor Project including those relating to taxation which are summarized in Section 22.2 of this report;

2. A Restated Concession Agreement entered into on June 6, 2009 among the Government, on behalf of the Kyrgyz Republic, and KGC (Restated Concession Agreement), which provides for the expansion of KGC’s then-existing concession to include the full area of its previous exploration and development licenses into the Concession Area as described in Section 4 of this report;

3. A Restated Shareholders Agreement entered into on June 6, 2009 among Kyrgyzaltyn, Cameco and Centerra (Restated Shareholders Agreement) which sets out the rights and obligations of Cameco and Kyrgyzaltyn with respect to their respective ownership of shares of Centerra. As a result of its disposition of Centerra shares in December 2009, Cameco is no longer a party to this agreement; and

4. The Restated Gold and Silver Sale Agreement dated June 6, 2009 among KOC on behalf of KGC, Kyrgyzaltyn and the Government under which Kyrgyzaltyn has agreed to purchase all of the gold produced by the Kumtor Project at market prices for reprocessing at its refinery in the Kyrgyz Republic as described in Section 19 of this report.

In addition to the Restated Investment Agreement, the Restated Concession Agreement, the Restated Shareholders Agreement and the Restated Gold and Silver Sale Agreement, there are a number of other important legal documents with respect to the Kumtor Project, which are briefly noted below:


2. The Priority Power Supply Agreement dated May 22, 1995 among the State Joint Stock Energy Holding Company of the Kyrgyz Republic and KGC, under which the Kumtor Project is guaranteed an uninterrupted source of electricity;

3. The Reclamation Trust Deed dated January 25, 1996 among the Government, KOC and Rothschild Trust Corporation Limited (the Trustee) establishing the reclamation trust described in Section 20.3 of this report; and

4. On November 16, 2010, Centerra entered into a three-year $150 million revolving credit facility with the European Bank for Reconstruction and Development (EBRD) as sole lender. This facility is for general corporate purposes, permitted acquisitions, working capital, capital expenditures and intercompany loans and/or capital contributions to finance
the development of Centerra’s existing properties in the Kyrgyz Republic and Mongolia, and for future investments in other EBRD countries of operation. The terms of the facility require KGC to pledge certain mobile equipment as security and Centerra to maintain compliance with specified covenants including financial covenants.

On February 15, 2012, the Kyrgyz Republic Parliament established an interim Parliamentary Commission to inspect and review Kumtor’s compliance with relevant Kyrgyz operational and environmental laws and regulations and community standards, and the State’s regulation over Kumtor’s activities. The Parliamentary Commission report was issued on June 18, 2012 and made a number unfavourable assertions regarding the Kumtor Project as reported in a Centerra press release dated June 22, 2012.

Following the Parliamentary Commission Report, the Kyrgyz Parliament passed a resolution obliging the Government to form a state commission (the State Commission) to “assess the environmental, industrial and social damage” caused by the Kumtor Project, and to provide a “legal examination of agreements made on the Kumtor Project in terms of protection of the state interests.”

The State Commission was formed in July 2012 and is comprised of three working groups with responsibility for environmental and mining matters, legal matters (including a review of all prior and current agreements relating to the Kumtor Project), and socio-economic matters (including a review of financial, taxation, procurement and employment related matters). Following extensive discussions and reviews by the State Commission at the Kumtor site, the final reports of the working groups are expected to be released in January 2013. There have also been four recent claims by the Kyrgyz State Inspectorate Office for Environmental and Technical Safety for a total payment of approximately $152 million relating to alleged environmental damages at the Kumtor Project.

Further to the Parliamentary review of its report and resulting recommendations, the Government on July 5, 2012 cancelled a previously issued Government Decree that provided Kumtor with land use (surface) rights over the Concession Area for the duration of the Restated Concession Agreement. As a result, the related land use certificate issued by the local land office was also cancelled. The revocation has not impacted Kumtor operations as the Company is relying upon assertions by the Prime Minister that the revocation would have no impact on or limit Kumtor’s activities or operations, its rights under the Restated Investment Agreement and the history of constructive dialogue between Centerra, Kumtor and the Government. Pursuant to the Restated Investment Agreement, Kumtor is guaranteed all necessary access to the Concession Area, including all necessary surface lands as is necessary or desirable for the operation of the Kumtor project.

2.2 Terms of Reference

Strathcona has been retained by Centerra to provide a technical review and report on the mineral resources and reserves of the Kumtor Project as at the end of September 2012. The report
is to comply with the standards for a technical report as set forth in NI 43-101. The report was prepared in cooperation with technical staff of Centerra. The requirement for a new report is due to a number of items including:

1. Updated mineral reserves outlined within the expanded Central Pit design have risen materially from those published by Centerra for December 31, 2011. This represents a 58% increase of contained ounces before accounting for the processing of 172 000 contained ounces of gold in the first nine months of 2012;

2. Reflecting the increased mineral reserves, the updated LOM plan has been extended to 2026 from the production profile outlined as part of the 2011 year-end mineral reserves and by five years from the LOM plan presented in the previous technical report. The revised LOM plan decreases 2012 and 2013 gold production by 465 000 total ounces relative to the LOM plan projected in the previous technical report, due to the necessity of ice and waste unloading of the SB Zone area which resulted in a seven-week closure in 2012 of the Kumtor mill for a lack of ore;

3. The increased overall size of the Central Pit has resulted in the requirements for additional capital expenditures over the next three years over and above the substantial capital investment completed in 2011 and 2012. The additional capital expenditures relate mostly to new mining equipment, due to a longer duration of higher overall mining rates and the requirements of substantial glacier ice mining, and include a provision for the costs associated with the expansion of the nominal processing plant capacity from currently 15 900 tonnes per day (tpd) to 18 400 tpd;

4. The Central Pit has had several occurrences of bedrock pit slope failures and movement of the historical waste dumps above the SB Zone. These incidents have resulted in material and economically adverse changes to the LOM and short-term production plans. The main effects have been the delay of mining of some of the high-grade parts of the deposit, and an increased overall stripping ratio of the pit. The expanded Central Pit will still have to deal with these potential issues and will additionally cut completely across sections of the Davidov glacier which raises new geotechnical issues that have not been experienced to date but are being addressed.

Centerra has made significant progress in mitigating many of the geotechnical risks by lowering pit-slope angles and by removing the waste dumps and the unstable ice below the dumps. This process is ongoing, prompted by accelerated ice movement in the past 12 months that has made continued mining of ore in the pit below unsafe. Access to the SB Zone ore was delayed by five months. While the understanding of the mechanisms resulting in the geotechnical issues has evolved significantly, the current mineral reserves and LOM plan assume the success of future remedial measures. Should the remedial plans be unsuccessful, the effects on the gold production and resulting cash flow could be more severe than those experienced to date and at worst could impact a portion of the remaining Central Pit mineral reserves; and
5. Centerra has since 2006 conducted a major development and drilling program to evaluate the potential for possible underground mining of high-grade parts of the Stockwork and SB Zones below the Central Pit. As of September 30, 2012, approximately $190 million had been spent on the underground project. The new ultimate pit design incorporates much of the high-grade SB Zone previously considered for underground mining, and additionally a substantial tonnage of the surrounding lower-grade gold mineralization which had not been considered for underground mining. The development and evaluation of high-grade mineralization of the Stockwork Zone outside of the new pit outline is well advanced. Successful test mining is required to be able to convert any resources considered for underground mining to reserves. This report provides an update on these activities.

The authors of this technical report have a long association with the Kumtor Project. Dan Redmond, P.Geo. has been employed by Centerra for over eight years and has made many visits to the site between 2005 and the date of this report. Mr. Redmond last visited the Kumtor Project in October, 2012 for 7 days. He is a qualified person within the meaning of NI 43-101. As a result of his employment with Centerra, Mr. Redmond is not independent of Centerra, applying all of the tests in Section 1.5 of NI 43-101.

Tommaso Roberto Raponi, P.Eng. has been employed by Centerra for over seven years and has made many visits to the site between 2005 and the date of this report. Mr. Raponi last visited the Kumtor Project in September 2012 for 6 days. He is a qualified person within the meaning of NI 43-101. As a result of his employment with Centerra, Mr. Raponi is not independent of Centerra, applying all of the tests in Section 1.5 of NI 43-101.

Because of the importance of geotechnical aspects as described in more detail in Sections 15.3, 15.4, 18.2 and 18.3 the current report is co-authored by Victor Vdovin, P. Eng. of Centerra. Mr. Vdovin (corporate strategic planning engineer) also has a long association with the Kumtor project as he has been employed by KOC working at the site from 2002 to 2008 as geotechnical assistant to the chief engineer, moving to Canada in 2008 to work at the Centerra head office initially as corporate geotechnical engineer. In addition to the years spent working at Kumtor, Mr. Vdovin has made numerous visits to the site, with the most recent conducted in September, 2012 for 5 days. Mr. Vdovin is a qualified person within the meaning of NI 43-101. However, as a result of his employment with Centerra, Mr. Vdovin is not independent of Centerra, applying all of the tests in Section 1.5 of NI 43-101.

Strathcona is familiar with the Kumtor Project having served as the independent mining engineer on behalf of the original banking consortium from 1995 to 2002, and having prepared four technical reports compliant with NI 43-101. The first such report was in 2004 for the initial public offering (IPO) or share listing of Centerra, the others were in 2006, 2008 and 2011, the last two being joint efforts with BGC Engineering and Centerra. Henrik Thalenhorst, P. Geo. of Strathcona has visited the site on a number of occasions, initially from November 27 to December 2, 1998, again from January 8 to 14, 2006 in preparation of the 2006 technical report, from October 28 to November 3, 2007 in preparation for the 2008 technical report, and from September 6 to 10, 2010 as part of a project review on behalf of EBRD with respect to a contemplated $150 million revolving credit facility for Centerra. His last visit to the Kumtor site was from June 13 to 16, 2012 in preparation for the current report. Henrik Thalenhorst is a
qualified person within the meaning of NI 43-101, and is independent of Centerra, applying all of
the tests in Section 1.5 of NI 43-101.

The metric system of units is used throughout this technical report, deviating only to
report ounces of gold. The currency used is the United States dollar, unless otherwise indicated.

2.3 Sources of Information

Following the initial discovery of gold at Kumtor in 1978, the Central Deposit was
delineated by a Soviet-Kyrgyz geological expedition. Extensive drilling programs, surface and
underground sampling programs and studies related to the Central Deposit and its exploitation
were completed by various Soviet agencies. The data from those studies were evaluated and
verified by the Kilborn Feasibility Study (as defined in Section 6.1) initiated by Cameco in 1993.

Since commencement of production in late 1996, additional technical studies have been
carried out by KOC, Cameco, Centerra and consultants retained by them with expertise in the
fields of geology, geotechnical issues, mineral resource estimation, engineering, mining,
metallurgy, and environment as part of the ongoing mining operations. Such studies have
included the preparation of periodic mineral resource models and annual mineral reserve
estimates and the reconciliation of the mineral reserve estimates to mine production, all of which
have been made available to the authors of this report. Other sources of technical information
have included geological and engineering studies, sampling and assaying results, internal notes
and memoranda, computer models, and monthly KOC operating reports from December 1996
through September 2012.

The mineral reserves and resources of the Kumtor Project were estimated by Dan
Redmond, P. Geo., Director of Mining, Technical Services for Centerra and John Baker, Mine
Manager, KOC and were published in a Centerra press release dated November 7, 2012. The cut-off
date for technical information used for the estimate is September 30, 2012.

Considerable experience has been accumulated by Centerra on the Kumtor Project, with
mineral resource and reserve estimates being monitored by means of mineral reserve-production
reconciliation, the results of which are reviewed in Section 15.11.

Information with respect to actual historical and future estimated operating and capital
costs and to taxation issues pertaining to the Kumtor Project has been provided by Centerra and
KOC for inclusion in the economic evaluation of the mineral reserves presented in Section 22.
The information presented in Section 20 (Environmental Studies, Permitting and Social Impact)
has been prepared by Centerra staff and KOC department heads for inclusion in this report.

References used in the preparation of this report are listed in Section 28.
2.4 Report Contributions

Table 3 sets out the contributions by the co-authors to this report.

<table>
<thead>
<tr>
<th>Company</th>
<th>Primary Areas of Responsibility</th>
<th>Report Sections Authored</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dan Redmond Centerra Gold Inc.</td>
<td>Mineral resource and reserve estimation, pit optimization and life-of-mine plan, economic evaluation, capital and operating costs estimates</td>
<td>Section 1 (except 1.1, 1.4, 1.5, 1.12, 1.13, 1.14, 1.15) Section 4 Sections 6.2 and 6.3 Section 14 Section 15 (except 15.3, 15.4) Sections 16, 18.1, 19, 20, 21, 22</td>
</tr>
<tr>
<td>Victor Vdovin Centerra Gold Inc.</td>
<td>Geotechnical aspects of pit slopes, waste dumps, and tailings facility</td>
<td>Sections 1.5, 15.3, 15.4, 18.2 and 18.3</td>
</tr>
<tr>
<td>Tommaso Raponi Centerra Gold Inc.</td>
<td>Milling, Metallurgy and Mill Expansion.</td>
<td>Sections 13, 17</td>
</tr>
<tr>
<td>Henrik Thalenhorst Strathcona Mineral Services Limited</td>
<td>Overall responsibility for the report and its conclusions</td>
<td>All sections not completed by the other authors</td>
</tr>
</tbody>
</table>
3 RELIANCE ON OTHER EXPERTS

The authors have relied, and believe they have a reasonable basis to rely upon the following individuals who have contributed to the, environmental, legal, marketing and taxation information stated in this report, as noted below:

- Manuela Battello, Treasurer, Centerra, with respect to the agreement with EBRD, gold sales and project financing in Sections, 19 and 22.2;

- Frank Herbert, General Counsel and Corporate Secretary, Centerra, with respect to legal matters addressed in Sections: 1.2, 1.13, 2.1 and 22.2;

- Matt Bliss, Vice President Environment and Sustainability, Centerra, with respect to environmental, health and safety and permitting matters addressed in Section 1.11 and 20;

- Andrew Sazanov, Vice President of Government and Corporate Relations, KOC, with respect to permitting and licensing matters in Sections, 4 and 20.1.

The four authors of this report have reviewed the information provided by the other experts as listed above and, based on the authors’ review of this information, believe it to be reasonable and reliable.
4 PROPERTY DESCRIPTION AND LOCATION

The Kumtor Project is located in the Kyrgyz Republic, one of the independent successor states of the former Soviet Union, some 350 kilometres to the southeast of the Kyrgyz capital of Bishkek (Figure 1) and about 60 kilometres to the north of the international boundary with the People’s Republic of China, in the Tien Shan Mountains, at 41° 52’ N and 78° 11’ E (Figure 2).

The Kumtor Project is comprised of the Central Deposit including the high-grade Stockwork and SB Zones and three satellite deposits known as the Sarytor, Southwest and Northeast Deposits.

Under the Restated Concession Agreement, KGC has the exclusive rights to all minerals within an area of approximately 26 000 hectares centered on the Central Deposit and with an expiry date of December 4, 2042 (the Concession Area). The open pits, deposits and prospects outlined in this report, existing and future waste dumps, the processing plant and the tailing management facility are located within the Concession Area. The Restated Concession Agreement also provides that the Government will support further and additional exploration activity by Centerra in the Kyrgyz Republic by inviting it to consider opportunities to acquire additional exploration and mining licenses. As of June 6, 2009, when the Restated Concession Agreement came into effect, the then existing mining and exploration licenses and associated agreements terminated and were superseded by the Restated Concession Agreement.

The outline of the Concession Area (corners 1 to 5) are shown is Figure 3 and the coordinates are set out in Table 4.

<table>
<thead>
<tr>
<th>Table 4 Coordinates of the Concession Area</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Gauss Kruger Coordinates</strong></td>
</tr>
<tr>
<td><strong>North</strong></td>
</tr>
<tr>
<td>Corner 1 4 648 000</td>
</tr>
<tr>
<td>Corner 2 4 648 000</td>
</tr>
<tr>
<td>Corner 3 4 631 000</td>
</tr>
<tr>
<td>Corner 4 4 631 000</td>
</tr>
<tr>
<td>Corner 5 4 635 000</td>
</tr>
<tr>
<td><strong>East</strong></td>
</tr>
<tr>
<td>14 260 000</td>
</tr>
<tr>
<td>14 276 000</td>
</tr>
<tr>
<td>14 276 000</td>
</tr>
<tr>
<td>14 264 000</td>
</tr>
<tr>
<td>14 260 000</td>
</tr>
</tbody>
</table>

The coordinate system is Gauss Krueger (Pulkovo 1942) Zone 14.
CONCESSION AREA

Northeast Deposit

KS13 Ultimate Central Pit

Ultimate Sarytor Pit

Ultimate Southwest Pit

Gauss Kruger (Pulkovo 1942) Zone 14

CENTERRA GOLD INC.

Kumtor 2012 Technical Report

Concession Area and Site Map

Figure 3
The Restated Investment Agreement specifies that KGC will be guaranteed such access to the Kumtor Project site, including all necessary surface lands, together with access to water, power and other infrastructure, as is necessary or convenient for the operation of the Kumtor Project.

However, further to a Parliamentary review of its report and resulting recommendations, the Government on July 5, 2012 cancelled a previously issued Government Decree that provided Kumtor with land use (surface) rights over the Concession Area for the duration of the Restated Concession Agreement. As a result, the related land use certificate issued by the local land office was also cancelled. The revocation has not impacted Kumtor operations as the Company is relying upon assertions by the Prime Minister that the revocation would have no impact on or limit Kumtor’s activities or operations, its rights under the Restated Investment Agreement and the history of constructive dialogue between Centerra, Kumtor and the Government. Pursuant to the Restated Investment Agreement, Kumtor is guaranteed all necessary access to the Concession Area, including all necessary surface lands as is necessary or desirable for the operation of the Kumtor project.

Legal surveys are not required to establish the boundaries of the registered area, and accordingly, no surveys of such boundaries have been undertaken.

For geological work including drilling and block modelling, local grids are used that are aligned with the predominant structural direction in each area of interest. The long axes of the Central, Southwest and Northeast Deposit grids are oriented northeast-southwest (at 41°) and the Sarytor Deposit grid at 64.6°. Section lines are at nominal 40-metre intervals and are oriented perpendicular to the long grid axes.

The authors have been advised by KOC that all permits and licenses required for the conduct of mining operations at the Kumtor Project are currently in good standing, except for the cancellation of the surface rights as discussed in Section 2.1. The principal permits are described in Section 20.1, while the environmental aspects and liabilities are described in Section 20.2. There are no royalties, payments or other agreements or encumbrances related to the Kumtor Project other than the agreements noted above and various forms of royalties and local taxation as set forth in Section 22.2 of this report. For a discussion of the potential risks associated with the Kumtor Project, See Section 25 of this report.
5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

Access to the Kumtor Project (Figure 2) is by main road from Bishkek to Balykchy, located on the western shore of Lake Issyk-Kul at an elevation of 1 600 metres, a distance of 180 kilometres. This road is currently being upgraded to four lanes in many places. A secondary road for 150 kilometres along the south shore of the lake leads to the town of Barskaun. The final 100 kilometres into the Tien Shan Mountains to reach the mine site is on a winding road that climbs to an elevation of 3 700 metres through 32 switchbacks of the Sary-Moynuk Pass before proceeding eastward on a plateau through which the Kumtor River and other seasonal rivers flow. KOC has done considerable work to improve and maintain this vital access road and despite occasional avalanches and movements of gravel and till down steep slopes during heavy rains, there has not been any lengthy period during which the road has been out of service.

The processing plant is situated in alpine terrain at an elevation of 4 016 metres, while the highest waste and glacier mining occurs above 4 400 metres. The main camp, administration and maintenance facilities are at about 3 600 metres. Local valleys are occupied by active glaciers that extend down to elevations of 3 800 to 3 900 metres, and undisturbed permafrost in the area can reach a depth of 250 metres. The region is seismically active as a result of the continuing convergence between India and Eurasia, but the Concession Area has a relatively sparse history of seismic activity. All facilities at the Kumtor Project, including the processing plant and tailings storage dam, have been designed in accordance with recommended seismic standards for the area.

The climate is continental with a mean annual temperature of minus 8° C. Extreme recorded temperatures vary from plus 23° C to minus 49° C, with short summers that last from June to September. Precipitation is low at around 300 millimetres per annum, with the majority falling in the summer months, and snow accumulations of 600 millimetres. The Kumtor Project operates 365 days per year and there have been no significant interruptions to operations because of climatic conditions.

Reflecting the high elevation and the harsh climate, sparse low vegetation is restricted to the valley floors and lower mountain slopes, with a total absence of trees or shrubs (Figure 4).

Most employees of KOC are citizens of the Kyrgyz Republic. The remainder are skilled expatriates. In November 2012, KOC had 2 718 local and 108 expatriate staff, as more fully described in Section 20.4. Most employees work a two-week rotation, and are transported between the mine site from Bishkek and the Issyk-Kul region using a company-owned commuter bus service.

Supplies are transported by rail to the Kumtor marshalling yard in Balykchy at the west-end of Lake Issyk-Kul and then trucked 250 kilometres to the mine site. There are usually several daily truck convoys, and cyanide is transported to the site four times a year under a special set of precautions, with no other traffic allowed at those times. A helicopter pad is available at the mine site for emergency use.
Photos looking south down the south arm of the Davidov Glacier

March 2000

June 2012
The mine site is connected to the Kyrgyz Republic national power grid with a 110-kV overhead power line that was constructed for the project and that runs parallel to the access road. The mine maintains two standby generator stations in case of power outages. Fresh water for human and industrial use is taken from Petrov Lake, situated five kilometres northeast of the mill site (Figure 3). The minimum water inflow into this glacial lake is estimated to be in excess of 1,000 cubic metres per hour or approximately twice the average project demand.
6 HISTORY

6.1 Exploration and Project Development History

Intermittent exploration in the Kumtor Project area dates back to the 1920s. Debris from the Sarytor Deposit was discovered in 1978 by a geophysical expedition of the state Kyrgyz Geology department sampling float from the frontal moraine of the Sarytor Glacier (Figure 3). The sole outcrop of what is now called the Central Deposit was found during follow-up prospecting. From 1979 to 1989, a systematic evaluation of the Central Deposit, and to a lesser extent of the Southwest Deposit, was carried out consisting of several phases of surface trenching and geological mapping, diamond drilling and underground development on three levels culminating in a detailed sampling program of the central upper part of the Central Deposit. A report entitled “Results of Detailed Exploration of the Kumtor Gold Deposit” was issued in 1989, and an initial mineral reserve statement was issued by the USSR State Committee on Reserves in March 1990.

After the break-up of the Soviet Union and following the emergence of the Kyrgyz Republic as an independent country in 1991, Cameco became aware of the Kumtor Project, concluded an agreement with the Kyrgyz Republic in 1992 and retained Kilborn Western Inc. to undertake a feasibility study of the project (the Kilborn Feasibility Study). The feasibility work program included data verification (by re-sampling parts of the underground openings and re-assaying of original sample rejects), additional and definitive metallurgical testwork, and a re-estimation of mineral resources and reserves using geostatistical methods, a block model and pit optimization software. The Kilborn Feasibility Study was completed in 1993, with updates in April 1994 and in May 1995.

Final agreements were signed, and the Kilborn Feasibility Study was approved by the Kyrgyz authorities in 1994, financing arrangements were concluded in 1995 and project construction was completed late in 1996. After initial capital expenditures of $452 million, commercial production was achieved in the second quarter of 1997. Based on a historical mineral reserve estimate non-compliant with NI 43-101 of 53.5 million tonnes with an average gold grade of 3.9 g/t, the project was forecast to treat 4.8 million tonnes per year for eleven years, with a total gold production forecast of 5.4 million ounces.

As the Central Deposit was being mined, KOC undertook a substantial amount of additional diamond drilling on the deposit and on surrounding exploration targets beginning in 1998, to augment the limited deposit information below elevation 3 950 metres, and to identify additional mineral resources and reserves that would extend the life of the operation. The pertinent drilling data are summarized in Table 5.
Table 5 Summary of Additional Drilling Completed, 1998 to September 30, 2012

<table>
<thead>
<tr>
<th>Year</th>
<th>Central Deposit</th>
<th>Other Targets</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Number of Holes</td>
<td>Length (m)</td>
</tr>
<tr>
<td>1998</td>
<td>16</td>
<td>3 010</td>
</tr>
<tr>
<td>1999</td>
<td>48</td>
<td>12 708</td>
</tr>
<tr>
<td>2000</td>
<td>0</td>
<td>0</td>
</tr>
<tr>
<td>2001</td>
<td>43</td>
<td>12 735</td>
</tr>
<tr>
<td>2002</td>
<td>10</td>
<td>2 119</td>
</tr>
<tr>
<td>2003</td>
<td>50</td>
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<tr>
<td>2004</td>
<td>65</td>
<td>22 263</td>
</tr>
<tr>
<td>2005</td>
<td>146</td>
<td>44 863</td>
</tr>
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<td>2006</td>
<td>50</td>
<td>18 280</td>
</tr>
<tr>
<td>2007</td>
<td>28</td>
<td>15 362</td>
</tr>
<tr>
<td>2008</td>
<td>96</td>
<td>39 472</td>
</tr>
<tr>
<td>2009</td>
<td>68</td>
<td>30 088</td>
</tr>
<tr>
<td>2010</td>
<td>38</td>
<td>17 020</td>
</tr>
<tr>
<td>2011</td>
<td>88</td>
<td>29 301</td>
</tr>
<tr>
<td>2012</td>
<td>86</td>
<td>27 754</td>
</tr>
<tr>
<td>Total</td>
<td>832</td>
<td>289 325</td>
</tr>
</tbody>
</table>

The figures in Table 5 include completed drill holes only, but omit drill holes that had to be re-drilled. Holes drilled for geotechnical and condemnation purposes are also excluded.

6.2 Mineral Reserve History

The mineral resource and reserve estimates for the Central Deposit have evolved over time. The principal estimates from 1990-2012 are summarized in Table 6 which does not include the mineral reserve estimates for the Southwest or Sarytor Deposits in order to allow comparison with the original Soviet estimate, which was for the Central deposit only.

Historical mineral reserve estimates quoted in this report and in Table 6 that were prepared prior to February 2001 pre-date NI-43-101, are not classified in accordance with, and may not be comparable to, current CIM standards. These mineral reserve estimates are quoted for their historical interest and have been superseded by the current estimate described in Section 15. When comparing the results of the individual estimates in Table 6, it should be recognized that the cut-off grade has changed through the project history, making direct comparisons difficult. The initial Soviet polygonal estimate in 1990, given its character, over-estimated the grade and under-estimated the ore tonnage. It also used a cut-off grade that was below a reasonable economic level in an effort to mine as much of the Central Deposit as possible. The Soviet estimate is not in compliance with past or present reporting guidelines in Canada.
### Table 6 History of Mineral Reserve Estimates to September 30, 2012 (Central Deposit Only)

(Millions of tonnes ore and waste and millions of ounces of gold)

<table>
<thead>
<tr>
<th>Cut-Off Grade Au (g/t)</th>
<th>Block Model</th>
<th>Bottom Bench</th>
<th>Gold Price</th>
<th>Ore Tonnage</th>
<th>Bottom Bench Au (g/t)</th>
<th>Waste Tonnage</th>
<th>Waste S/R</th>
<th>Total In-situ Mineral Reserves plus Mined Production</th>
<th>Contained Gold (Ounces)</th>
</tr>
</thead>
<tbody>
<tr>
<td>USSR State Committee, 1990</td>
<td>1.0 Polygonal</td>
<td>3 700</td>
<td>273</td>
<td>5.1</td>
<td>No Production</td>
<td>66.2</td>
<td>4.3</td>
<td>No Data</td>
<td>9.2</td>
</tr>
<tr>
<td>Feasibility Study, April 1994</td>
<td>2.0 GSII</td>
<td>3 796</td>
<td>$350</td>
<td>53.5</td>
<td>3.9</td>
<td>No Production</td>
<td>53.5</td>
<td>3.9</td>
<td>273</td>
</tr>
<tr>
<td>KOC, October 1, 1995</td>
<td>1.7 GSII</td>
<td>3 722</td>
<td>$375</td>
<td>76.6</td>
<td>3.7</td>
<td>No Production</td>
<td>76.6</td>
<td>3.7</td>
<td>581</td>
</tr>
<tr>
<td>KOC, December 31, 1998</td>
<td>1.7 OK96</td>
<td>3 800</td>
<td>$325</td>
<td>31.4</td>
<td>4.6</td>
<td>1985</td>
<td>6.3</td>
<td>10.8</td>
<td>4.8</td>
</tr>
<tr>
<td>KOC, December 31, 1999</td>
<td>1.7 KS-1</td>
<td>3 800</td>
<td>$301</td>
<td>32.7</td>
<td>4.4</td>
<td>248</td>
<td>7.6</td>
<td>18.9</td>
<td>4.3</td>
</tr>
<tr>
<td>KOC, December 31, 2001</td>
<td>1.5 KS-3</td>
<td>3 770</td>
<td>$300</td>
<td>29.8</td>
<td>3.9</td>
<td>329</td>
<td>11.1</td>
<td>31.0</td>
<td>4.4</td>
</tr>
<tr>
<td>KOC, December 31, 2003</td>
<td>1.3 KS-4</td>
<td>3 754</td>
<td>$325</td>
<td>26.2</td>
<td>3.6</td>
<td>353</td>
<td>13.5</td>
<td>41.0</td>
<td>4.4</td>
</tr>
<tr>
<td>KOC, December 31, 2004</td>
<td>1.3 KS-5</td>
<td>3 754</td>
<td>$375</td>
<td>26.3</td>
<td>3.4</td>
<td>382</td>
<td>14.5</td>
<td>46.0</td>
<td>4.4</td>
</tr>
<tr>
<td>KOC, December 31, 2005</td>
<td>1.3 KS-6</td>
<td>3 620</td>
<td>$400</td>
<td>35.5</td>
<td>4.1</td>
<td>621</td>
<td>17.6</td>
<td>52.2</td>
<td>4.2</td>
</tr>
<tr>
<td>KOC, December 31, 2006</td>
<td>1.3 KS-7</td>
<td>3 650</td>
<td>$475</td>
<td>27.1</td>
<td>4.9</td>
<td>702</td>
<td>25.8</td>
<td>57.7</td>
<td>4.1</td>
</tr>
<tr>
<td>KOC, December 31, 2007</td>
<td>1.0 KS-8</td>
<td>3 650</td>
<td>$550</td>
<td>31.6</td>
<td>4.3</td>
<td>651</td>
<td>20.6</td>
<td>61.9</td>
<td>3.9</td>
</tr>
<tr>
<td>KOC, December 31, 2008</td>
<td>1.0 KS-9</td>
<td>3 642</td>
<td>$675</td>
<td>32.0</td>
<td>3.8</td>
<td>645</td>
<td>20.1</td>
<td>66.3</td>
<td>3.9</td>
</tr>
<tr>
<td>KOC, December 31, 2009</td>
<td>1.0 KS-10</td>
<td>3 642</td>
<td>$825</td>
<td>39.1</td>
<td>3.3</td>
<td>858</td>
<td>21.9</td>
<td>72.5</td>
<td>3.8</td>
</tr>
<tr>
<td>KOC, December 31, 2010</td>
<td>0.85 KS-11</td>
<td>3 618</td>
<td>$1 000</td>
<td>45.2</td>
<td>3.4</td>
<td>976</td>
<td>21.6</td>
<td>78.3</td>
<td>3.7</td>
</tr>
<tr>
<td>KOC, December 31, 2011</td>
<td>0.85 KS-12</td>
<td>3 618</td>
<td>$1 000</td>
<td>45.4</td>
<td>3.5</td>
<td>961</td>
<td>21.2</td>
<td>84.3</td>
<td>3.7</td>
</tr>
<tr>
<td>KOC, September 30, 2012</td>
<td>0.85 KS-13</td>
<td>3 500</td>
<td>$1 350</td>
<td>78.8</td>
<td>3.4</td>
<td>1 548</td>
<td>19.6</td>
<td>84.8</td>
<td>3.7</td>
</tr>
</tbody>
</table>

The tonnages mined before the mineral reserve estimate dates include the low-grade stockpiled ore not yet milled. The purpose of the table is to show the “growth” of the deposit over time; estimates for the years 1990 to 1999 are historical estimates not in compliance with NI-43-101.
Geostat Systems International Inc. (Geostat) used the Soviet information to develop a block model (GSII model) for the Kilborn Feasibility Study. The GSII model remained the official mineral reserve model until early 1999 and was in compliance with the reporting guidelines of National Policy 2A in effect at the time. It used the original “mineralized” envelope as defined by Soviet geologists, which was too broad. As a result, the grade interpolation of the GSII block model “smeared” gold grades away from higher-grade areas into lower-grade sections of the deposit, and thus over-estimated the tonnage but under-estimated the grade of the feasibility study mineral resources and reserves. Since 1999, additional block models have been created by KOC, Cameco Gold or Centerra, each an improvement over its predecessor, by incorporating the increasing geological knowledge about the deposit (Table 5) and about the grade distribution experienced during mining. This process has now culminated in the KS-13 model, which incorporates all information available as of September 30, 2012. All mineral resource and reserve estimates by KOC, Cameco and Centerra since 2002 have been undertaken in accordance with the CIM Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines as required by NI 43-101.

The mineral reserve estimates for the Central Deposit (excluding the satellite deposits), before mining, have varied over time, between 42 million tonnes grading 4.7 g/t gold with a strip ratio of 6.0 to 1, and most recently 164 million tonnes grading 3.6 g/t gold with a strip ratio of 16.8 to 1. Similarly, the gold estimated to be contained in the Central Deposit (production plus mineral reserves at any given time) has varied from a low of 6.4 million ounces to a high of 18.9 million ounces as of September 30, 2012, with the latter more than twice the original 1990 Soviet estimate of 9.2 million ounces of contained gold. In conjunction with the increase in the total economic inventory of the Central deposit, the total tonnage (ore plus waste) to be excavated has increased nine-fold from 327 million tonnes (feasibility study 1994) to the current 2.9 billion tonnes.

The variance in the mineral reserve estimates over the years is due to, improvements in the unit operating costs and more recently in the price of gold which allowed for an increased strip ratio, and the discovery of additional mineralization through the sustained and large amount of exploration drilling which culminated in the discovery of the SB Zone in 2005, a second, high-grade area of the deposit in addition to the Stockwork Zone.

### 6.3 Production History

The mill started processing ore in December of 1996 and achieved commercial production in late 1997. The ore mined from the Central Deposit was augmented by ore from the Southwest Deposit in the years 2006 to 2008. As of September 30, 2012, a total of 85.5 million tonnes of ore from both deposits has been milled with an average gold content of 3.9 g/t. Since start-up, 8.5 million ounces of gold have been recovered. Some 1.2 billion tonnes of waste have been mined for an overall strip ratio of 15.2 to 1. Annual production data compiled from the monthly operating reports issued by KOC are shown in Table 7. The mine data were determined from truck counts (for tonnages) and grade-control data (for the gold grade). The differences with the plant data are small (5% for each of the tonnage and the gold grade, with the contained ounces practically identical and reflecting the usual accounting differences between mine and mill.
Table 7 Production History to September 30, 2012

<table>
<thead>
<tr>
<th>Year</th>
<th>Ore &amp; Low-Grade Mined</th>
<th>Waste Mined</th>
<th>Ore Milled</th>
<th>Mill Recovery</th>
<th>Gold Produced</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes (’000)</td>
<td>Grade Gold (g/t)</td>
<td>Tonnes (’000)</td>
<td>Strip Ratio</td>
<td>Tonnes (’000)</td>
</tr>
<tr>
<td>1996</td>
<td>477 4.1</td>
<td>13 346 28.0</td>
<td>159 3.2</td>
<td>58.2</td>
<td>10</td>
</tr>
<tr>
<td>1997</td>
<td>5 017 5.2</td>
<td>17 946 3.6</td>
<td>4 023 5.3</td>
<td>73.3</td>
<td>502</td>
</tr>
<tr>
<td>1998</td>
<td>5 349 4.5</td>
<td>26 425 4.9</td>
<td>5 254 4.9</td>
<td>78.5</td>
<td>645</td>
</tr>
<tr>
<td>1999</td>
<td>8 054 3.5</td>
<td>33 105 4.1</td>
<td>5 298 4.5</td>
<td>79.4</td>
<td>611</td>
</tr>
<tr>
<td>2000</td>
<td>6 518 4.1</td>
<td>36 763 5.6</td>
<td>5 498 4.7</td>
<td>81.5</td>
<td>670</td>
</tr>
<tr>
<td>2001</td>
<td>5 606 5.2</td>
<td>46 863 8.4</td>
<td>5 470 5.2</td>
<td>83.1</td>
<td>753</td>
</tr>
<tr>
<td>2002</td>
<td>5 141 3.5</td>
<td>49 184 9.6</td>
<td>5 611 3.7</td>
<td>78.1</td>
<td>529</td>
</tr>
<tr>
<td>2003</td>
<td>4 828 5.0</td>
<td>72 881 15.1</td>
<td>5 631 4.5</td>
<td>82.6</td>
<td>678</td>
</tr>
<tr>
<td>2004</td>
<td>3 428 6.2</td>
<td>81 427 23.8</td>
<td>5 654 4.4</td>
<td>82.1</td>
<td>657</td>
</tr>
<tr>
<td>2005</td>
<td>6 135 3.1</td>
<td>74 903 12.2</td>
<td>5 649 3.4</td>
<td>81.2</td>
<td>499</td>
</tr>
<tr>
<td>2006</td>
<td>3 887 2.6</td>
<td>81 534 21.0</td>
<td>5 696 2.3</td>
<td>73.0</td>
<td>303</td>
</tr>
<tr>
<td>2007</td>
<td>5 132 2.5</td>
<td>109 649 21.4</td>
<td>5 545 2.4</td>
<td>72.7</td>
<td>301</td>
</tr>
<tr>
<td>2008</td>
<td>4 967 4.2</td>
<td>115 548 23.3</td>
<td>5 577 3.9</td>
<td>79.7</td>
<td>556</td>
</tr>
<tr>
<td>2009</td>
<td>4 464 4.7</td>
<td>111 079 24.9</td>
<td>5 780 3.7</td>
<td>76.7</td>
<td>525</td>
</tr>
<tr>
<td>2010</td>
<td>5 765 4.1</td>
<td>110 701 19.2</td>
<td>5 594 4.0</td>
<td>79.5</td>
<td>568</td>
</tr>
<tr>
<td>2011</td>
<td>6 020 3.5</td>
<td>145 854 24.2</td>
<td>5 815 3.8</td>
<td>80.8</td>
<td>572</td>
</tr>
<tr>
<td>2012 (1-9)</td>
<td>491 2.1</td>
<td>108 934 221.9</td>
<td>3 209 1.7</td>
<td>72.6</td>
<td>125</td>
</tr>
<tr>
<td>Total</td>
<td>81 279 4.1</td>
<td>1 236 142 15.2</td>
<td>85 463 3.9</td>
<td>78.9</td>
<td>8 504</td>
</tr>
</tbody>
</table>

Mining tonnages are reported above the cut-off grade used at the time. Because the low-grade material is currently being used as mill feed and will continue to be processed in accordance with the LOM plan, it is treated as ore when calculating the stripping ratio (S/R) in Table 7.
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

The gold deposit of the Kumtor Project occur in the Tien Shan Metallogenic Belt, a Hercynian fold and thrust belt that traverses Central Asia, from western Uzbekistan in the west through Tajikistan and the Kyrgyz Republic into north-western China, a distance of more than 2,500 kilometres (Figure 1). It “represents the central part of the Altaid Orogenic Collage of Central Eurasia” (Porter, 2006). Along this belt, described by Cole (1992) as “a major metallogenic province which contains many world-class mesothermal-type gold deposits” occur a number of important gold deposits including Muruntau (one of the largest gold deposits in the world), Zarmitan, Jilau and Kumtor. “The Tien Shan itself is an extremely complex fold and fault belt in which various components represent different orogenetic events that span the Phanerozoic and were later overprinted by Alpine-Himalayan deformation”. This belt is located at “…the margin of Paleozoic Asia (Baltica and Siberia) [to the north] and the Palaeo-Turkestan Ocean” (Cole, 1992).

The Tien Shan Belt is sub-divided into three main tectonic “elements” (Porter, 2006), the North, Middle and South Tien Shan. These elements, which may be interpreted as accretionary terrains, are up to 250 kilometres wide in Uzbekistan and Tajikistan but converge eastwards so that in the Kumtor Project area the Middle Tien is only a few tens of kilometres wide. The North and Middle Tien Shan terrains are separated by the Nikolaev Fault, while the Middle and the South Tien Shan are separated by the Atbashi-Inylchek Fault.

In the Kumtor area, the Nikolaev Fault occupies the Kumtor River valley (Figure 5), while the pronounced valley in the southern part is probably an expression of the Atbashi-Inylchek Fault.

7.2 Kumtor Geology and Structural Evolution

The known gold deposits of the Kumtor Project area are located in the northwestern section of the Middle Tien Shan (Figures 5 and 6). The Middle Tien Shan is cored by granitic and granodioritic intrusive rocks assigned to the Meso- or Neo-Proterozoic era and which contain remnants of even older gneisses. The intrusive rocks are thrust north-westwards over Proterozoic sedimentary and volcanic rocks which are assigned a Riphean age (also Meso to Neo-Proterozoic) and are in turn in fault and thrust contact with clastic sedimentary rocks of Vendian age (youngest Proterozoic or oldest Paleozoic). Further to the northwest, additional thrust and fault slices including the Kumtor Fault Zone (KFZ) complete a section of thrusting and faulting several kilometres wide, with the youngest rocks exposed in the footwall of the KFZ.
CONCESSION AREA

Legend

- Glacial Cover
- Major Alpine thrust
- Major Alpine steep faults
- Alpine tectonic melange
- Major Paleozoic thrust
- Other major Paleozoic faults

Middle Tien Shan

- Recent sediments (QIII-QIV)
- Red clay, conglomerate-breccia (Pg-N)
- Limestone (CI-2); siltstone, sandstone (C1)
- Siltstone (O1-2), black cherty (Cm-O1)
- Ore-bearing rock succession - tillite conglomerates, phyllites (Cm-V)
- Vendian rock succession - tillite conglomerates, phyllites
- Riphean rock succession - arkoses, tuffs, basals, rhyolites (R3-V)
- Early Proterozoic rock succession - gneiss, marble, schists (P1)

- Syenite (P1?)
- Granite, granodiorite (R2)
- Granite, diorite (PR?) with potassium alteration

Exploration Target

CLIENT
CENTERRA GOLD INC.

PROJECT
Kumtor 2012 Technical Report

TITLE
Surface Geology
Concession Area

APPROVAL
H.T.

DATE
December 2012

PROJECT No.
329-3

STRATHCONA MINERAL SERVICES LIMITED
TORONTO - CANADA

File: Fig6_2012_Targets.cdr
Figure 6
The South Tien Shan terrain is dominated by Early Paleozoic sediments with subordinate mafic volcanics and some felsic intrusive bodies but contains ophiolites which indicate further plate tectonic movements within this terrain.

As is apparent from the description of the various tectonic slices within the Middle Tien Shan terrain, the geology of the Kumtor area is dominated by structural features due to thrusting and faulting. Significant new knowledge has been gained in the past few years through the work of R. Seago (Seago 2006a-c, 2007a&b) and T. Starling of Telluris Consulting (Telluris 2006, 2007, 2009) whose work has shown that the structural geology of the Kumtor area has evolved through four main deformation events identified as D 1 to D 4 (oldest to youngest). D 1 and D 2 pre-date the Carboniferous. D 2 is of Caledonian age with D 1 being an even earlier burial metamorphism event. D 3 is of Hercynian age (late Carboniferous to early Permian) and extends over the mineralization episode with pre-, syn- and post-mineralization D 3 structures. Mao et al. (2004) report a late Carboniferous to early Permian age for the Kumtor mineralization itself. The observations at Kumtor correlate with the age of D 3 at Jilau (Cole et al. 2000) and Muruntau, where the age of the mineralization, however, is Triassic (Wilde et al. 2001). D 4 is of Alpine or Himalayan age, from Tertiary to the present.

The presence of a ubiquitous schistosity (S 1) in the meta-sediments of the area is a result of the D 1 deformation episode which peaked at low to mid-greenschist facies regional metamorphism. During the D 2 episode, the S 1 schistosity was folded into a series of open, asymmetric F 2 folds which trend NE-SW with an associated axial planar crenulation cleavage (S 2). While associated faults dip to the SE, these structures have been subjected to two further phases of deformation and their original orientation is therefore changed. Telluris (2007) reports of an early silicification event during D 2, but is silent on any gold mineralization that may have been introduced at this time.

The third deformation episode D 3 resulted in both S 1 and S 2 being deformed by an S-N compressional event resulting in the formation of E-W trending D 3 fore-thrusts (with dips to the south) and back-thrusts (with dips to the north), and a series of roughly N-S trending strike-slip faults, lateral ramps and small-scale kink bands (F 3). The most recent D4 event has re-activated many of the pre-existing structures, especially D 2, and has imparted a NE-SW striking structural fabric, with an overall SE dip, on the main faults and S 1 foliation. Many of the D 2 cohesive structures were re-activated during D 4 resulting in unconsolidated fault breccias and gouges. The D4 tectonic axis is SE-NW.

As a result of these multiple deformation events, the structural geology in the Kumtor area is dominated by several major thrust slices with an inverted age relationship (Figure 7). The dominant structural direction is northeast-southwest (D 4), with moderate dips to the southeast. Each thrust sheet contains older rocks than the sheet it structurally overlies. Four major structural slices, stacked upon one another and bounded by long-lived faults that were active repeatedly, have been identified, as follows:
• Slice 0 consists of Cambro-Ordovician limestone and phyllite, thrust over Tertiary sediments of possible continental derivation that in turn rest on Carboniferous clastic sediments with apparent profound unconformity;

• Slice 1 constitutes the Kumtor Fault Zone (KFZ), whose upper limit is the Upper Kumtor Fault. The KFZ is generally a dark-grey to black, graphitic gouge zone, The KFZ strikes northeasterly, dips to the southeast at moderate angles and has a width of up to several hundred metres. The adjacent rocks in its hanging wall are strongly affected by shearing and faulting for a distance of up to several hundred metres;

• Slice 2 includes the mineralization which is hosted by Vendian meta-sediments, grey carbonaceous quartz-sericite-chlorite schists or phyllites that are strongly folded and schistose, with a large proportion of faulted and sheared rocks. Slice 2 is delimited in the footwall by the Upper Kumtor Fault and in the hanging wall by the Lysii Fault. It appears that the mineralizing event, itself multi-phase as discussed in Section 7.3, has healed some of the earlier brittle features within Slice 2; and

• Slice 3 consists of phyllites, also of Vendian age, that show several phases of folding. The dip of the schistosity is shallow to steep to the northwest or shallow to the southeast. The subsequent brittle deformation is less strongly developed compared to Slices 1 and 2. Slice 3 is sub-divided into three units based on the orientation of the foliation. Slice 3 is important for the pit slope stability discussed in Section 15.3.1 because of the development of the Main Boundary Thrust (MBT) and other tectonic zones (TZ Faults).

It is important to note that most fault structures in the area are persistent, with thick gouge or tectonic breccia and are therefore potential failure surfaces. This is attributed to D 4 re-activation of pre-existing faults.

Figures 8 to 11 illustrate the geology in the third dimension for the three deposits, providing an illustration of the structural complexities of the Kumtor area. The location of the four sections is shown on Figure 12.
Geological Section
Line 122
Central Deposit
(section corridor +/- 20m)

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Figure 8
December 2012

CENTERRA GOLD INC.
Kumtor 2012 Technical Report

Geological Section
Central Deposit Line 122
(section corridor +/- 20m)

See Figure 12 for Section Line Location
Source: Map and data provided by SRK UK and KOC
Note: Drill holes were initially collared on the waste dump.

Source: Map and data provided by SRK UK and KOC

See Figure 12 for Section Line Location
The main structures of the Central Deposit are also present in the Southwest Deposit and at Sarytor. The main thrust faults at Sarytor strike E-W, and the faults and S1 schistosity along with the mineralized zones have a shallow southerly dip. The structures are truncated in the west, with Slice 0 juxtaposed against a steep NNE-SSW trending fault as shown in Figure 12 (Seago 2007a). This fault is strike-slip in nature indicating a D3 origin, and is likely the over-steepened continuation of the Lower Kumtor Fault, which separates Slices 0 and 1 in the Central Pit area. The E-W orientation of thrust faults with a southerly dip is a function of D3 deformation, with limited overprinting by the D4 structural event.

Structures start to swing to a NE-SW trend as they continue eastward into the Southwest Deposit area, where there is a strong D4 structural overprint. This D4 overprint can be traced northward into the Central Pit and thus the structural make-up of the Southwest Deposit is more like that of the Central Pit than Sarytor. These observations indicate that the D3 structures were rotated into a NE-SW trend by the D4 structural event.

7.3 General Description of the Kumtor Mineralization

Gold mineralization of economic importance occurs where the Vendian sediments have been hydrothermally altered and mineralized, an event that may have taken place in Permian time as discussed above, based on structural considerations (Ivanov et al., 2000) and as reported by Mao et al. (2004). Gold mineralization is developed over a strike distance of more than twelve kilometres. The Central Deposit is the most important accumulation identified to date and has considerable dimensions with a strike length of 2.4 kilometres, a vertical extent of one kilometre and a width of up to 300 metres. Other known occurrences along the mineralized trend are the Southwest Deposit, Sarytor Deposit, and the Northeast Deposit as shown on Figure 6. Figure 12 provides a composite of the main ore accumulations of the Central, Southwest, Sarytor and Northeast Deposits.

According to Ivanov et al., 2000, mineralization in the Kumtor area took place in four main pulses. An initial pulse resulted primarily in pervasive quartz-carbonate-albite-chlorite-sericite-pyrite alteration, with little gold of economic consequence being deposited. However, this early alteration may have “stiffened” the host rocks sufficiently to make them susceptible to the intensive veining, stockwork and hydrothermal breccia development during the next two pulses that deposited all of the economically significant gold. A few typical examples of mineralized drill core are in Figure 13.

The temperature of formation of the second stage veins was 310 ± 15°C, according to Ivanov & Ansdell, 2002. The mineralogy during the main phases includes early K-feldspar followed by later albite and variable amounts of carbonate (calcite, dolomite, ankerite and siderite), quartz, pyrite, sericite and chlorite, in addition to small amounts of chalcopyrite, haematite, barite, strontianite and accessory magnetite, scheelite, ferberite, rutile, cassiterite, sphalerite, galena, native gold and tetrahedrite, as well as a number of silver-gold, lead and nickel tellurides. The feldspars combine to comprise nearly 20% of the ore, the carbonates collectively 25 to 30%, pyrite 15 to 20%, quartz 5 to 10%, and the remainder are host rock inclusions.
Drill hole: D670 at 82.4m
Au: 1.2 ppm
Banded phyllite with quartz-carbonate veins and pyrite with micro-inclusions of pyrrhotite, chalcopyrite; pyrite+sphalerite clusters

Drill hole: D670 at 53.8m
Au: 0.5 ppm
Carbonaceous chlorite-sericite metasomatite with intense albite alteration

Drill hole: D670 at 177.6m
Au: 1.4 ppm
Albite carbonate metasomatite with stockwork of pyrite-quartz-feldspar-carbonate veins

Drill hole: D919 at 225.4m
Au: 15.9ppm
Pyrite - carbonate (dolomite) - albite metasomatite
The mineralization is most intense, and the gold grade is the highest, where the metasomatic activity was continuous through mineralization phases two and three. This is the case for the Stockwork and SB Zones, and explains their higher-than-average gold grades. The last pulse created planar carbonate-pyrite metasomatic rocks that are associated with zones of intense deformation of previously altered phyllites.

Native gold and the gold-silver tellurides are intimately associated with pyrite to the extent that gold grade and pyrite content are “positively correlated” (Ivanov et al., 2000). The gold and the gold-bearing minerals occur as very fine inclusions in the pyrite, with an average size of only 10 microns. This, together with the poor cyanide leach response of the gold tellurides, accounts for the partly refractory nature of the ore. The refractory characteristics are reflected in the relatively low historic and forecast gold recovery of around 80%, despite the very fine grind applied to the pyrite flotation concentrate from which most of the gold is recovered. However, the fine grain size of the gold also renders assaying of this mineralization relatively reliable, with only a small nugget effect.

Most of the mineralization takes the form of veins, veinlets, and breccia bodies in which the mineralization forms the matrix. In the more intensely mineralized areas, the surrounding host rock has also been altered. Post-ore faulting is generally parallel to, or at low angles with, the mineralized sequence. These faults often carry significant quantities of graphite, and other carbonaceous components which constitute the sources for the preg-robbing character of some of the mineralization.

7.4 The Central Deposit

Within the Central Deposit, several zones of gold mineralization have been delineated as shown in Figure 12. Two parallel zones of alteration and gold mineralization strike northeasterly and dip to the southeast at 45° to 60°, separated by 30 to 50 metres of barren or poorly mineralized rock. The South Zone, with a length of 700 to 1,000 metres and a horizontal width of 40 to 80 metres, is reasonably well mineralized throughout its entire length, with an average gold grade of 3 to 4 g/t. The North Zone, somewhat more extensive along strike but with a similar width, has lesser gold grade continuity and splits into a number of individual lenses that have average gold grades in the range of 2 to 3.5 g/t.

At their north-eastern end, the North and South Zones coalesce into the Stockwork Zone, which has been the heart of the deposit during the earlier years of mining, having the highest gold grades and the best grade continuity. Its dimensions in the upper part of the deposit are 400 to 500 metres long by 50 to 200 metres wide, with an average gold grade of 5 to 6 g/t. The Stockwork Zone plunges northeasterly at 40° to 50°, and diminishes in size below elevation 3,700 metres. The Stockwork Zone is located closest to the northeast pit high wall and thus has a large effect on the overall strip ratio of the pit. Drilling in recent years has further extended the Stockwork Zone down dip and outlined a higher grade core beneath the bottom of the planned open pit, parts of which are being considered for underground mining.
In the southwestern part of the Central Deposit, the SB Zone (structurally a part of the South Zone) tops out at elevation 3,900 metres, below which it widens significantly. It was the discovery of the SB Zone that gave rise to the large increase in the mineral reserves in 2005 of the Central Deposit (Table 6). Drilling since 2008 has extended the SB Zone along strike to the southwest increasing its current known strike extent to 1,000 metres, a vertical extent of 650 metres, and a width that ranges from 6 to 75 metres, with excellent grade in the range of 5 g/t gold.

7.5 The Southwest Deposit

The Southwest Deposit is located three kilometres to the southwest of the Central Deposit across the Davidov glacier, along the KFZ (Figures 5 and 6). Recent underground drilling has defined the southwestern limit of the SB Zone and the northeastern limit of the SW Deposit below the glacier, with a barren gap of approximately 600 metres. To the southwest, the Southwest Zone is covered by the Sarytor glacier, beyond which additional mineralization is known as the Sarytor Deposit.

The structural/lithological framework of the Southwest and Sarytor Deposits is identical to those of the Central Deposit, as described in Section 7.4 and as shown in Figures 10 and 11, with the structural dips generally at angles ranging from 20° to 50°, somewhat shallower than at the Central Deposit.

A number of individual zones of mineralization have been identified at the Southwest Deposit within an overall mineralized envelope that is around 100 metres thick and has been traced along strike for a distance in excess of one kilometre. Individual zones tend to be relatively narrow and of different intensities of mineralization, and their contacts are often marked by tectonic crush zones with black fault gouge. The footwall contacts are generally sharp and clearly defined, while the hanging wall contacts are more gradational. Gold enrichment along both contacts can be observed on many sections. Due to the flat orientation of the mineralized zones, their contacts have a sinuous feature in both plan and section.

7.6 The Sarytor Deposit

The Sarytor Deposit is located to the southwest of the Southwest Deposit. The two deposits are probably contiguous below the Sarytor glacier. The main geological structures are common for the Southwest and Sarytor Deposit. The drill results indicate that the mineralized section in the Sarytor Deposit strikes east-west and dips south at 20° to 30°. The thickness of the mineralized envelope is relatively consistent and varies from 80 to 120 metres, with the strike length of the known mineralization being approximately 800 metres.

Host rocks are structurally disturbed slates and phyllites with lenses of till-like conglomerates and dolomitic slates. Development of background alteration is weak and represented mainly by vein-type silicification. Unaltered host rocks do not carry any elevated gold values. The zone has been traced by drilling for 200 to 300 metres down dip.
The mineralized envelope hosts three mineralized zones separated by zones of strongly faulted, barren host rocks. Alteration intensity and zone thickness increase southward. Metasomatism is represented by banded albite-carbonate-quartz alteration with 3% to 5% pyrite. Barite and siderite are well developed in the southern part of Sarytor. As a rule, pyrite content is positively correlated with the gold grade.

### 7.7 The Northeast Deposit

The Northeast Deposit is an extension of the Kumtor mineral system exposed between the Lysii and Petrov glaciers approximately three kilometres northeast of the Central Pit. Soviet-era exploration included the excavation of a series of adits and surface drilling along 800 meters of anomalous surface exposures. KOC has completed several drilling campaigns over the last few years that allow the estimation of a near-surface inferred resource of currently 4.1 million tonnes at an average gold grade of 2.1 g/t. This resource covers 400 metres of strike length to a depth of approximately 200 metres.

The complex structural elements of the Central Deposit are well represented at the Northeast Deposit, including the Kumtor Fault Zone. Gold mineralization occurs in a series of stacked, relatively short and narrow lenses that strike northeast and dip to the southeast from 60° to 80°, considerably steeper than at the Central Deposit. Because of the limited size of individual mineralized zones, all of the mineral resources of the Northeast Deposit have been placed in the inferred class.

Exploration drilling in 2011 encountered higher gold grades to the east of the current mineral resource, and additional exploration work is planned.

### 7.8 Other Mineralized Zones

Several other mineralized prospects are known within the Concession Area including the Petrov, Muzdusuu, Bordoo and Akbel areas, shown on Figures 5 and 6. These exhibit many structural, alteration and mineralization features similar to the main zones described previously. Details on previous exploration results and future planned exploration activities of these areas are provided in Section 24.1.4.
8 DEPOSIT TYPE

In his summary paper, Porter (2006) states:

“Gold mineralization occurs in two principal settings within the Tien Shan Mineral Belt, namely as i) porphyry and epithermal systems developed within magmatic arcs, and ii) orogenic-type gold deposits that are structurally controlled, and temporally and spatially associated with late Palaeozoic, syntectonic to early postcollisional, highly evolved, I-type granodioritic to monzonitic intrusives in fore- and back-arc terranes.” (Porter 2006, page 1); and

“The orogenic gold deposits of the Tien Shan Mineral Belt,... include some of the largest economic gold accumulations in the world, and span the time scale from Lower to Late Palaeozoic. The greatest concentration of significant orogenic gold deposits however, is in the southwestern part of the belt, in the South and Middle Tien Shan of Uzbekistan and Kyrgyzstan. These deposits are associated with Permian magmatism emplaced during the final- to early post-collisional stages of orogenesis, within a sutured back-arc setting containing carbon-rich sedimentary sequences…” (Porter 2006, page 4).

The general characterization of the orogenic gold deposits by Porter (2006) above is borne out by the detailed observations described in the Sections 7 of this report. Given the location astride a major fault of regional importance and owing to the strong association of gold mineralization with a multi-phased metasomatic system at relatively high temperatures, the gold deposits of the Kumtor Project, are members of the class of structurally controlled meso-thermal gold replacement deposits.


9 EXPLORATION

There have been no geophysical or geochemical surveys or investigations undertaken since the Kumtor Project was brought into production in 1997. All the data used for the current resource estimate from which the current reserve estimate is derived, is based on diamond drilling, which is described in Section 10. Blast-hole data from open-pit mining are not used for grade estimation but are used qualitatively for variography. They provide the data for the daily grade control model which is the intermediary for the reconciliation of the resource block model with the mill production figures.
10 DRILLING

The principal exploration data acquisition method at the Kumtor Project is diamond drilling. There is a large historical drill-hole database (augmented by underground exploration results at the Central, Southwest and Sarytor Deposits) dating back to the Soviet era. To a large extent, this historical information is no longer relevant to the current mineral reserve estimate, since the upper parts of the Central Deposit, to which the majority of historical information pertained, has now been mined out. Models for the Southwest, Sarytor and Northeast Deposits were constructed with very little historical Soviet era data. There are only small tonnages in the current mineral resources that rely to a significant extent on Soviet data, and these old data are progressively being verified by in-fill or replacement drilling.

As a result of the lack of sufficiently detailed information at the Central Deposit below elevation 3,950 metres, about 28% of the 1994 Kilborn Feasibility Study open-pit mineral reserves, which contained one-quarter of the total gold to be mined, had been substantially less well documented than the upper part of the deposit. To fill this information gap, and to explore for extensions to the known mineralization, KOC has undertaken a large in-fill diamond drill program in the years 1998 to 2012 as described in Section 6.1 and as compiled in Table 5. Drilling continues to be undertaken from various pit benches and setups outside of the pit, including on the waste piles. This has now increased the density of the drill pattern in the lower part of the deposit to equal or better than was available at the time of the Kilborn Feasibility Study for the upper part.

In the Central, Southwest, Sarytor and Northeast Deposits, the drill holes are now generally spaced 30 to 40 metres along strike and 40 to 80 metres down-dip in geologically complex areas, and at 80 metres along strike and 60 to 80 metres down-dip in other areas. The Kumtor Project database as of September 30, 2012 consisted of more than 336,000 assays, with roughly 22% dating from the Soviet era as shown by deposit in Table 8. Of the remaining reserves and resources outlined in this report, only a small fraction, estimated less 10%, are reliant on Soviet era data and almost all cases some KOC-Centerra data is used to confirm these results.

Table 8 Assay Data by Source, Kumtor Project

<table>
<thead>
<tr>
<th>Deposit</th>
<th>Soviet Era</th>
<th>KOC-Centerra</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Central</td>
<td>42,681</td>
<td>166,361</td>
<td>209,042</td>
</tr>
<tr>
<td>Southwest</td>
<td>15,774</td>
<td>39,941</td>
<td>55,715</td>
</tr>
<tr>
<td>Sarytor</td>
<td>5,765</td>
<td>38,201</td>
<td>43,966</td>
</tr>
<tr>
<td>Northeast</td>
<td>4,465</td>
<td>13,037</td>
<td>17,502</td>
</tr>
<tr>
<td>Others</td>
<td>6,379</td>
<td>3,675</td>
<td>10,054</td>
</tr>
<tr>
<td>Totals</td>
<td>75,064</td>
<td>261,215</td>
<td>336,279</td>
</tr>
</tbody>
</table>
The KOC drill programs have been conducted with their own fleet of surface diamond drill rigs of which five are currently active, and four underground rigs of which three were active as of September 30th, 2012 are currently active. KOC drill crews are both national (Kyrgyz) and expatriate, under the supervision of expatriate drill foremen.

The majority of the KOC diamond drill holes are steeply inclined and recover HQ-size core, except when ground conditions necessitate a reduction in core size to NQ. For all of the holes, drill collars are surveyed and down-hole deviations are measured at intervals of 20 to 30 metres using a Reflex single shot camera. Limitations on set-ups dictate that a certain number of off-section holes are drilled. Drill cores are logged for geological and geotechnical information, and are photographed prior to sampling. Drill-collar coordinates, down-hole deviation surveys, assay results, and information on lithology, alteration and mineralization are recorded in the mine or exploration drilling databases. The drilling database and the assay database derived from it are used for mineral resource estimation as described in Section 14.

Drill core recovery typically varies from 80% to 100%, averaging greater than 95%. In certain cases where the core recovery from mineralized intervals is low, the hole is stopped and re-drilled to achieve better core recovery. There is no evidence that core recovery issues impact the reliability of the gold assay data used for mineral resource and reserve estimation. The angle of intersections between the drill holes and the mineralization is generally such that the true width of the mineralization is equivalent to 70% to 95% of the length of mineralized drill-hole intervals.


11 SAMPLE PREPARATION, ANALYSES AND SECURITY

With only a small amount of Soviet era data still relevant for resource estimation, historical methods of core sampling, sample preparation and analytical protocols and quality control methods will not be described in this report; they have been discussed in previous technical reports, e.g., Redmond et al., 2011 as is quoted below

“The sampling protocol employed in the years prior to 1993 is summarized below from the descriptions in the Kilborn Feasibility Study. As in many projects of the Soviet era, the entire core was removed for sampling, in intervals of an average length of 1.4 metres. Core recovery averaged only 75%. Trench samples were generally one metre long, presumably taken horizontally, but the sampling method is not described. Channel samples were collected from the extensive underground openings approximately one metre above the floor and varied from 0.5 to 2 metres long. The channels are reported to have measured 10 centimetres wide by 5 centimetres deep.”

Lynda Bloom of Analytical Solutions Limited visited the site in August 2012 and has made a number of recommendations (Bloom, 2012), which are addressed in the following sections, as appropriate. A recurring and general remark is that all procedures of logging, sampling and assaying would benefit from the introduction of software programs that streamline the processes and reduce the incidence of transcription errors.

11.1 Core Sampling Method and Approach

For the drilling completed by KOC, the drill core length is measured and checked against the depth blocks inserted by the drillers in the core boxes. The core is logged and photographed. Sample intervals are chosen to be representative of geological features such as veining, alteration and mineralization. Individual samples are normally one metre long, but the interval may be increased to 2.0 metres in unaltered rocks. With the exception of geotechnical holes, drill holes are sampled over their entire length.

Competent drill core selected for sampling is cut by a diamond saw into two halves. One half is placed into a numbered bag and sent to the laboratory for assaying. The other half is placed back in the core box and retained in permanent storage on site. Incompetent core intervals are sampled with a scoop that fits snugly into the individual rows, removing one-half of the material at the discretion of the sampling technician.

Blasthole cuttings are sampled with a pie-shaped wedge that is placed radially away from the collar of the hole. It collects about ten kilograms per bench. Duplicate samples are taken at a rate of 4%. The duplicate results were reviewed by Bloom (2012) who observed that for assays $\geq 1$ g/t, most of the duplicate results fell within an error of $\pm 50\%$, with no bias. The relative error below 1 g/t is larger. Given the relatively forgiving nature of the mineralization with respect to sampling, this is satisfactory, if not ideal.
11.2 Sample Preparation

All sample collection, preparation and assaying from the 1998-2012 drilling programs were performed by KOC personnel at the KOC-owned site laboratory, which is not certified but is subjected to periodic calibration and operations checks by the Kyrgyz National Accreditations agency. Sample collection protocols are monitored by KOC’s exploration manager and the QA/QC geologist. Preparation and assay protocols are supervised by KOC’s chief assayer at the Kumtor Project. Samples are delivered to and from the laboratory at the mine site by KOC personnel. Additional security of samples is not required in this mining environment.

Since 1998, drill core as well as blast hole, mill and tailings samples including solutions have been assayed at the mine laboratory using the following sample preparation and assaying procedures.

1. Samples are received by the sample preparation section with a corresponding manifest indicating the number of samples and the numerical sample identification;

2. Dry the samples at a temperature of 105° C;

3. Crush the entire sample in three sequential jaw crushers to 95% passing 1.7 millimetres (10 mesh);

4. The last of the three jaw crushers directly feeds a rotary splitter that is set to obtain a 150-gram sub-sample. The remaining reject material is returned to the original bag and, in the case of core samples, is delivered to the exploration department for storage; and

5. Pulverize the sub-sample to 100% passing 106 microns (150 mesh) using a ring-and-puck pulverizer.

The responsible author is of the opinion that the KOC sample collection and sample preparation protocols in place at the Kumtor Project are in accordance with normal industry operating practices. The laboratory currently handles about three times the volume of samples it was originally designed for. Bloom (2012) has commented on the outdated sample preparation equipment that was first installed in 1997 and recommends updating the equipment and available space to increase the efficiency, reliability and safety of the sample preparation procedure.

11.3 Analytical Methods

Since 1998, drill core as well as blast hole, mill and tailings samples have been assayed at the mine laboratory. A 30-gram aliquot of the pulp is fire assayed with a suitable flux and a gravimetric finish. The sample weight is decreased to 20 grams for samples with high sulphide content. The laboratory does have three atomic absorption (AA) spectrometers, but an AA finish of low-grade samples has not been implemented since no analytical-grade acetylene gas
was available in Kyrgyzstan. As a result, the laboratory has a relatively high detection limit for gold.

The responsible author is of the opinion that the KOC assaying protocol in place at the Kumtor Project is in accordance with normal industry operating practices.

### 11.4 Quality Control Procedures

Quality control procedures have changed over time. Those in effect before 2008 have been described in Redmond et al. (2011) and are not repeated here. Since 2008, the KOC laboratory has participated twice yearly in the Geostats international round-robin, and the lab has performed well on samples with gold grades higher than 0.5 g/t but poorly on samples with less than 0.10 g/t Au.

During an original round of assaying of drill core samples, only field blanks but no standard reference materials (SRMs) are inserted. The field blanks consist of sawed core that is known to contain less than 50 ppb gold, and is inserted within what is expected to be a mineralized zone at a rate of 5%. There is no duplicate or check assaying at this stage.

Once the initial assays have been received, new pulp splits are produced for 20% of the samples that returned ≥0.1 g/t gold for re-assay at the KOC laboratory. The new splits are re-numbered and certified SRMs from Geostats, are inserted at a rate of one for every 20 samples. Thirty such standards are in use, with relevant gold values and matrices. The field blanks remain part of this re-numbered set of pulps.

External check assaying at the Alex Stewart laboratory is also undertaken on 25% of the duplicates re-submitted to the KOC laboratory (5% of the assays >0.1 g/t gold). The original pulps are re-homogenised prior to making duplicates, to avoid any bias due to settling of sulphides in bags during transport/storage.

Samples that returned gold values ≥10 g/t initially are assayed in duplicate at each lab, using two thirty-gram aliquots, and the results are averaged. If either of the two laboratories produces results for the standards that are outside of the accepted limits, then the entire batch is re-assayed at the laboratory in question. If there is a conflict between the two laboratories despite satisfactory standard results, coarse rejects of the sample batches in question are re-submitted to both laboratories, including the field blanks, and including a new set of standards. If there are still unresolved issues after the rejects have been re-assayed, re-splitting of the half-core and a repeat of the entire sampling and assaying protocol is undertaken.

In her report, Bloom (2012) has made the following recommendations to simplify the current quality control protocol of the mine chemical laboratory:

a. There are currently up to 44 Geostats reference materials (RMs) being used. Only 5 to 6 RMs should be in use in a 6-month period with these RMs selected to cover the expected range of gold concentrations;
b. All drill core samples submitted to the Kumtor laboratory will have either one blank or one RM submitted with each batch of 20 samples. RMs should be alternated with blanks. This is consistent with the laboratory batch size of 22 samples plus its internal control samples;

c. Results for blank samples will be acceptable up to 0.2 g/t gold. Any results over 0.2 g/t will result in repeat assays for ten samples assayed before and after the QC failure. Note that if the laboratory successfully achieves a lower detection limit for exploration drill core in the future, then a value of 0.1 g/t gold could be used as the designation of QC failures;

d. Results for RMs will be verified upon receipt and any values outside acceptable limits (± 2 standard deviations or a minimum of ± 10%) will result in repeats for ten samples assayed before and after the QC failure being requested;

e. Requests for repeats do not require renumbering and re-submission of the samples;

f. It is not necessary to resubmit samples greater than 0.1 g/t gold for re-assay at the Kumtor laboratory;

g. The practice of sending samples quarterly to ALS Kara Balta should continue. Based on 2011 activity, the current rate of 350 samples quarterly represents 7% of all samples and this could be modified to include only 5% of samples with greater than 0.1 g/t gold. It is not necessary to re-number the sample bags. The same pulp should be submitted that was assayed at Kumtor and RMs should be inserted as previously. Two of the 30 gm bags of Geostats standards should be submitted to Kara Balta for each RM insertion;

h. It is not necessary to resolve differences of greater than 20% between Kara Balta and the Kumtor laboratory but further investigation should be done to define acceptable precision limits at different grade range.

It is anticipated that the KOC laboratory will implement these recommendations.

11.5 Quality Control Results

The results of the coarse reject and pulp check assay program undertaken by KOC from 2002 to 2012 are the most pertinent for the Kumtor Project mineral resource estimate, along with the results of the check assays performed at ASAEL. This information is compiled in Table 9 for assay pairs averaging greater than 0.1 g/t gold.

Detailed analysis of the comparison between the KOC laboratory and the Central Scientific Research Laboratory assays for samples from before 2007 shows that the detection limits of the two laboratories were different, with CSRL reporting higher values than KOC for values <0.1 g/t. In the range from 0.1 to 1.0 g/t, KOC was systematically higher, typically by a factor of 10% to 20%. Above 1 g/t, the two laboratories produced identical average results in most cases.

The comparison of the assays from KOC mine assay lab and from ASAEL shows good performance by the KOC lab for samples with gold grades higher than 0.5 g/t, excellent performance for samples with higher than 1.0 g/t Au, but poor performance for samples with less than 0.25 g/t Au. This is largely a reflection on the continued use of gravimetric finishes for all samples, including those with values less than 1.0 g/t Au.
Table 9 Check Assay Results (>0.1 g/t Gold)

<table>
<thead>
<tr>
<th>Period</th>
<th>Number of Pairs</th>
<th>Pairs Removed</th>
<th>Original KOC (g/t)</th>
<th>KOC Re-split (g/t)</th>
<th>Check Results (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2003 and earlier</td>
<td>1 279</td>
<td>8</td>
<td>2.6</td>
<td>2.6</td>
<td>2.6</td>
</tr>
<tr>
<td>2004</td>
<td>3 424</td>
<td>42</td>
<td>2.8</td>
<td>2.8</td>
<td>2.7</td>
</tr>
<tr>
<td>2005</td>
<td>4 990</td>
<td>89</td>
<td>4.5</td>
<td>4.5</td>
<td>4.4</td>
</tr>
<tr>
<td>2006</td>
<td>4 578</td>
<td>74</td>
<td>4.4</td>
<td>4.4</td>
<td>4.4</td>
</tr>
<tr>
<td>2007</td>
<td>768</td>
<td>18</td>
<td>2.5</td>
<td>2.5</td>
<td>2.4</td>
</tr>
<tr>
<td>Total</td>
<td>15 039</td>
<td>221</td>
<td>3.8</td>
<td>3.8</td>
<td>3.7</td>
</tr>
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</table>

Check Assays at Alex Stewart Assayers and Environmental Laboratory

<table>
<thead>
<tr>
<th>Period</th>
<th>Number of Pairs</th>
<th>Pairs Removed</th>
<th>Sample Type</th>
<th>Original KOC (g/t)</th>
<th>Check Results (g/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2002</td>
<td>489</td>
<td>4</td>
<td>Reject</td>
<td>2.3</td>
<td>2.4</td>
</tr>
<tr>
<td>2002</td>
<td>44</td>
<td>0</td>
<td>Pulp</td>
<td>2.9</td>
<td>2.7</td>
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<tr>
<td>2005</td>
<td>38</td>
<td>0</td>
<td>Pulp</td>
<td>1.4</td>
<td>1.4</td>
</tr>
<tr>
<td>2007</td>
<td>197</td>
<td>0</td>
<td>Pulp</td>
<td>0.9</td>
<td>0.8</td>
</tr>
<tr>
<td>2008</td>
<td>400</td>
<td>5</td>
<td>Pulp</td>
<td>12.6</td>
<td>11.8</td>
</tr>
<tr>
<td>2008</td>
<td>38</td>
<td>1</td>
<td>CRM</td>
<td>6.5</td>
<td>6.7</td>
</tr>
<tr>
<td>2009</td>
<td>29</td>
<td>0</td>
<td>Reject</td>
<td>81.9</td>
<td>82.7</td>
</tr>
<tr>
<td>2009</td>
<td>597</td>
<td>8</td>
<td>Pulp</td>
<td>13.1</td>
<td>13.3</td>
</tr>
<tr>
<td>2009</td>
<td>57</td>
<td>5</td>
<td>CRM</td>
<td>9.5</td>
<td>10.4</td>
</tr>
<tr>
<td>2010</td>
<td>0</td>
<td>0</td>
<td>Reject</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>2010</td>
<td>442</td>
<td>5</td>
<td>Pulp</td>
<td>10.6</td>
<td>10.3</td>
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<tr>
<td>2010</td>
<td>35</td>
<td>1</td>
<td>CRM</td>
<td>7.4</td>
<td>8.1</td>
</tr>
<tr>
<td>2011</td>
<td>973</td>
<td>30</td>
<td>Pulp</td>
<td>10.50</td>
<td>10.29</td>
</tr>
<tr>
<td>2011</td>
<td>78</td>
<td>0</td>
<td>CRM</td>
<td>7.17</td>
<td>7.35</td>
</tr>
<tr>
<td>2012</td>
<td>470</td>
<td>13</td>
<td>Pulp</td>
<td>11.11</td>
<td>11.22</td>
</tr>
<tr>
<td>2012</td>
<td>45</td>
<td>0</td>
<td>CRM</td>
<td>5.92</td>
<td>5.80</td>
</tr>
<tr>
<td>All</td>
<td>518</td>
<td>4</td>
<td>Rejects</td>
<td>6.8</td>
<td>6.9</td>
</tr>
<tr>
<td>3 161</td>
<td>61</td>
<td>Pulps</td>
<td>10.5</td>
<td>10.4</td>
<td></td>
</tr>
<tr>
<td>253</td>
<td>7</td>
<td>CRM</td>
<td>7.4</td>
<td>7.8</td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td>3 932</td>
<td>72</td>
<td></td>
<td>9.85</td>
<td>9.76</td>
</tr>
</tbody>
</table>

The “pairs removed” constitute a small proportion of the overall check assay population. They were excluded from the comparison in Table 9 because the pairs are so dissimilar as to most likely be caused by something other than an assay accuracy problem or the natural variability (sample error) of the material being assayed.
Bloom (2012) has commented on the recent performance of the mine laboratory with respect to the reliability of the assay data produced as follows:

“The various data sets that test the accuracy of the Kumtor laboratory generally confirm that the laboratory’s performance is acceptable. Some of the contradictory data, such as the Geostats RMs assayed at Kumtor being biased high for grades greater than 5 g/t, may be related to the quality of the RMs and not necessarily the laboratory’s abilities.

There is however consistent evidence that the assays less than 1 g/t are not reliable. It is most likely that this is because the Kumtor laboratory currently uses a gravimetric finish for all assays (with the exception of tailings). Gravimetric assays are not used at commercial laboratories for samples with less than 3 g/t Au. This is because the method requires the physical manipulation and weighing of the fire assay bead and for low grade samples the bead is near non-existent ....

In general, the Kumtor laboratory operates at industry standards but it is challenged by old equipment, especially in sample preparation, and is operating in a space that was designed for about 30% of the work load.

Access to analytical grade acetylene will allow the laboratory to provide fire assays with an AAS-finish. The method needs to be implemented after thorough method development and testing. The fire assay AAS-finish assays will improve the quality and reliability of assays, in particular in the range of less than 1 g/t gold.”

11.6 Conclusions

The author responsible for this section of the report is satisfied that the Kumtor assay data base is without a bias that would make the Kumtor resource estimate unreliable. This is based on the following observations:

- The sample preparation and assaying methods used by KOC meet industry standards. The results of the check assay program indicate that there are no major apparent issues with respect to assay accuracy as shown in Table 9. However, the QA/QC protocol used prior to 2008 was both incomplete (the lack of true blanks and standards that were blind to the KOC laboratory and to CSRL) and cumbersome, since much duplicate assaying was performed on low-grade to very low-grade samples; and
- The revised QA/QC protocol introduced in 2008 has resulted in a marked improvement of the reliability of the assays within mineralized zones while allowing a significant reduction in duplicate assaying of waste material;
- While the audit by Bloom (2012) has confirmed the high bias of the KOC laboratory for gold grads <1.0 g/t, the effect on the resource and reserve estimate is negligible in light of the good reconciliation between the KS-13 block model and actual production figures as described in Section 15.11.

Special sample security measures at Kumtor are unnecessary in an operational environment and are therefore not being taken.
12 DATA VERIFICATION

While data verification is the principal method of determining the reliability of the data used for mineral resource or reserve estimation in projects in the pre-production stage, the main verification in the case of a mine with an operating history of many years is the ability of the reserve estimate to reliably forecast past production tonnages and metal grades for the ore mined. This not only verifies the data underlying the mineral reserve and resource estimates, but also the various parameters and procedures applied to the data during the estimation process.

In the case of the Kumtor Project, a program of mineral reserve-production reconciliation is routinely undertaken with good results, and this is described in more detail in Section 15.11. Comments on the historical and current data verification procedures are as follows.

12.1 Historical Database

The historical database created during the Soviet era has not changed since the Kilborn Feasibility Study, by which it was verified. Details have been reported in previous technical reports, most recently in Redmond et al., 2011, who confirmed the Soviet data by mine production and KOC drilling. Problematic areas of the deposit have been mined out in the meantime. The substantial additional drilling undertaken by KOC since 1998 (Table 5) has generally confirmed the Soviet data, and losses in one part of the deposit were usually balanced by gains elsewhere.

12.2 KOC Database

Standard database checks are being performed regularly under the supervision of the KOC Exploration Manager, who is responsible for its upkeep and reliability, with the verification being performed by the Gemcom validation routine. Assay results, lithology, drill-hole locations and down-hole surveys are verified back to the Laboratory Data Sheets and original data by the mine QA/QC geologist for every drill hole.

The in-fill drilling program to cover the new, deeper KS 13 pit design is largely complete, and the database for the deposit is now considered reliable down to below the 3500 metre elevation, well below the bottom of the original Soviet pit design at 3700 metres. After the removal of previous spurious drill results based on recent drilling, the successful completion of the in-fill drill program and mine production statistics, the data relied upon for the estimation of the mineral resource model appear valid.

Dan Redmond has on several occasions undertaken verification of both the exploration and production data which included the validation of the assay database and the bench scale reconciliation between the exploration drilling data with those obtained from mine production. This provides confidence in the reliability of both the exploration and production data used in this report.
12.3 Bulk Density

In-situ volumes of both ore and waste are translated into tonnes by applying a bulk density factor of 2.85 tonnes per cubic metre. This figure has been used since the Kilborn Feasibility Study in 1995 and at that time was based on existing bulk-density data. As shown in Section 15.11, block-model, grade-control model and milled tonnages match within two percent, which shows that the bulk density used on average gives reliable tonnage data. Open-pit excavations do not readily lend themselves to a simple calculation that uses the volume excavated vs. the tonnage of ore plus waste to confirm at least a general bulk density value because the waste tonnage is determined by truck counts.

The responsible author concludes that the use of the general value of 2.85 tonnes per cubic metre is reasonable in light of the operating experience.

12.4 Check Sampling

Given the status of the Kumtor Project as an active mine, KOC has in the past not undertaken any independent sampling or check assaying for the purpose of verifying the assay database, since the successful years of operation, together with the good reconciliation between the mineral resource model estimates and actual production results as described in Section 15.11, had not indicated any requirement for independent data verification. However, Strathcona was asked to undertake a program of independent random spot re-sampling of some high-grade intersections of the SB Zone in conjunction with the preparation of the 2008 technical report. Strathcona collected the remaining half core from three high-grade drill-hole intervals. The samples were assayed in duplicate at the SGS Lakefield laboratory with the same assay protocol as is used at KOC. The results have been reported previously (most recently in Redmond et al., 2011) and were found to be in general agreement with the high-grade nature of the SB Zone. A slight low bias of the re-sampled assays was interpreted as being due to the small size of the re-sampling program.
13 MINERAL PROCESSING AND METALLURGICAL TESTING

This item is discussed in Section 17 of this report.
14 MINERAL RESOURCE ESTIMATES

14.1 General

As is shown in Table 6, mineral reserve/resource estimation at the Kumtor Project has been undertaken using a succession of block models which have followed procedures in accordance with the standards as required by NI 43-101 since 2001. Each new model was generally an improvement on its predecessor models, by incorporating new drilling information as it became available (Table 5), and by being able to compare the results of each model with actual mining performance.

Four models were used for the September 30, 2012 mineral resource estimate. The model identified as KS-13 was developed from its predecessor, KS-12 for the Central Deposit by adding a substantial amount of additional drill-hole information completed in the first 9 months of 2012. No changes were required for the existing SW-1 model developed for the Southwest Deposit in 2004 or for the SR-2 model that was developed for the Sarytor Deposit in 2006. In both cases, only a few drill holes have been completed since the models were created and this drilling did not impact the resource models. Model NE-2, which was an update to the first model of the Northeast Deposit, was developed in December 2011, and was used again for this September 30th, 2012 estimate and no additional drilling was completed at the Northeast Deposit during 2012. It is worth noting that each of the models provides an estimate for the entire deposit before any mining, but reports only the tonnages left below the pit outlines that exist as of the effective date of the estimate. In this way each new resource model version can be validated against historical operating production results for which significantly greater amounts of assay data are available as a result of grade control sampling.

These models were used to estimate mineral resources for the Kumtor Project as of September 30, 2012, applying the parameters and procedures discussed in Section 14.3. The block models in turn were used to estimate the mineral reserves for the Kumtor Project using the technical and economic pit design parameters stated in Sections 15.4 and 15.5. The estimation process was supervised by Dan Redmond, P. Geo., Director of Mining, of the Technical Services Department of Centerra.

14.2 Geological and Mineralized Zone Modeling

While fault contacts impart sharp boundaries between mineralized and un-mineralized rocks, gold-grade boundaries at the Kumtor Project gold deposits are often gradational over one to several metres. The main geological challenge in creating a viable geological model for mineral resource estimation has been the delineation of “mineralized zones”. For the Kilborn Feasibility Study in 1993, which addressed the Central Deposit only, the original Soviet model, an all-encompassing “mineralized envelope” around the main mineralized zones, was used. This proved too vague and did not provide sufficient constraint during grade interpolation. As a result, mineral reserve predictions that used this historical model tended to be correct for the contained gold, but were high for the ore tonnage and low for its gold grade.
In subsequent models, vein and alteration intensities together with gold-grade information were used to sub-divide the gold mineralization into a number of individual mineralized zones, taking into account the major faults described in Section 7.2. There were up to twenty such zones at the Central Deposit, twelve at the Southwest Deposit, eleven at Sarytor, and three at the Northeast Deposit. The interpretation and delineation of each mineralized zone was completed on 20 to 40 metre spaced sections, took into account the observations (geological mapping and blasthole data) on the mining benches and has benefited from the substantial additional drilling conducted since the project began commercial production. Wireframes were created for each zone in GEMCOM using natural grade shells of between 0.5 and 0.75 g/t gold.

First starting for the 2006 year-end mineral reserve and resource estimates and continuing for the KS-13 model, the SB and Stockwork Zones were further sub-divided into an outer, low-grade shell and an inner, high-grade shell, roughly at a 6 g/t gold cut-off grade, which effectively separates two different gold-grade populations in these two parts of the Central Deposit.

This method provided several advantages for the overall modeling process including better control of the high grade assay population during gold-grade interpolation, the ability to report mineral resources below the pit potentially amenable to underground mining above a realistic cut-off grade, and the ability to apply different top-cutting thresholds and interpolation parameters to the distinctly different gold assay populations.

14.3 Block Models

14.3.1 Central Deposit Model (KS-13)

The KS-13 model was developed in 2012 for the Central Deposit. It is based on the model of the mineralized zones described above and upon the most recent drilling information, including the results of all in-fill drilling completed from 1998 to September 30, 2012 (Table 5). The KS-13 model uses blocks measuring 10 by 10 by 10 metres, with the vertical dimension matching the mining bench height, which was changed from 8 metres in 2011 as a result of the introduction of larger mining equipment. Each whole or partial block is assigned either to the high-grade core, surrounding lower grade mineralization or is coded as waste and is then not available for gold grade interpolation. Waste blocks are further coded into bedrock waste, glacial till, waste dump or glacial ice based on lithological interpretation from the exploration and till depressurization drilling that has been completed in and around the Central Pit.

As with previous models created after 2005, the KS-13 model, utilizes three-dimensional solid modelling of the mineralized zones and “partial” or percentage blocks to more accurately estimate the tonnage of the narrower mining zones. This also allows the manipulation of the blocks to include an external dilution provision for each block as described in Section 15.2. The KS-13 model also incorporates estimates of metallurgical recovery into each block to improve the reliability of the mineral reserve estimate in general, and as an aid to shorter-term mine scheduling.
The classification of the different waste material types was important to mineral reserve modelling of the Central Pit as each waste material (rock, waste dumps, ice) has different in-situ densities, unit mining costs per tonne and geotechnical slope parameters as outlined in Sections 15.4. and 15.5.

The KS-13 model was developed in two steps. The first was to construct an open-pit diluted block model that could be utilized for open pit optimization and mineral reserve estimation. The second phase was to model high grade mineralization for potential underground mining and to define additional low grade mineral resources located outside of the ultimate pit design.

For the open-pit diluted block model, the following workflow was used:

1. All available assays results for a particular sample interval were averaged, and this value was used for the sample interval in question;

2. Each resulting averaged sample value was then top-cut at a series of different top cutting values ranging from 20 to 100 g/t in 10 g/t intervals. Each of the resulting nine top-cut values for each sample interval was stored in the sample database. In this way, a range of different top-cutting values can be evaluated before final grade interpolation and different top-cut factors could be used for different areas in the deposit;

3. The raw averaged assay (from step 1) and the nine corresponding top-cut values (from step 2) were then separately composited to a down-hole length of four meters. The composite raw assay sample intervals were used to define the limits of low-grade and high grade shells;

4. Composites were then coded to be either in the high-grade shell, low grade shell or located outside of the overall mineralized envelope;

5. All mineralized blocks within the limits of the overall low-grade envelope (including any blocks coded as high-grade blocks) were interpolated using ordinary kriging of the composite assays with a top cutting gold value of 50 g/t Au applied to each individual raw assay. The 50 g/t top-cutting value was chosen because it provided the best reconciliation results with actual production. The grade interpolation was done using three progressively larger search passes in the plane of the overall mineralization with ranges at 30, 60 and 90 metres respectively;

6. Gold grades were interpolated without a “hard boundary” between the low and high grade zones such that both high and low-grade composites could be used to interpolate both high and low-grade mineralized blocks. The goal was to create a smoother gold grade distribution within the entire mineralized envelope which lowers overall gold grade of the mineralized zone similar to what was observed from open-pit production results;

7. For a block to be interpolated required a minimum of four composites to be located within the search range, and at least one of these had to be from a second drill hole. A maximum of 12 composite data points is imposed for block grade interpolation;
8. The provision for external dilution as outlined in Section 15.2 was applied to all interpolated blocks;

9. A review of the resulting block model was undertaken relative to detailed ore control information for the years 2006 to 2012 as described in the preceding section. It was concluded that the KS-13 block model was a reasonable estimator of open pit mineral reserves. The model was then used for pit optimization (Section 15.6), open-pit design Sections 15.4 and 15.5) and reserve reporting (Section 15.9).

With step 1 complete, step 2 of the KS-13 block model development was to provide a realistic estimate of additional mineral resources in the Stockwork Zone and SB Zone that may be mined using selective and high-cost underground mining methods, and to estimate additional low grade mineral resources located outside of the ultimate pit design. To accomplish this, blocks or partial blocks outside of the ultimate pit were re-interpolated with the same three search passes as described above. There were, however, two differences compared to the open-pit estimate:

1. A “hard boundary” (the shell designed at gold grade of 6 g/t) was observed between the high and low grade shells and composite data. In this way only composites coded as high grade were used to interpolate blocks within the high grade shells and only composites coded as low grade were used to interpolate blocks within the low grade shells; and

2. To better reflect the denser information needed for the classification of the high-grade mineralization, each of the three interpolation runs required information from a minimum of three holes. For a block to be interpolated required a minimum of 7 composites located within the search range, with a maximum of three composites allowed per drill hole.

For phase 2, top-cutting factors of 70 g/t and 50 g/t were used for high grade and low grade for composites interpolation, respectively.

### 14.3.2 Southwest Deposit Model (SW-2)

The Southwest block model (SW-1) was constructed in 2004 and following production from the deposit from 2006 to early 2008, has proven to be a reliable measure of mineral reserves, as shown in Section 15.11. The mineralization is contained within twelve individual mineralized zones, with three of the zones containing the majority of the mineral resources and reserves. After capping of individual assays at levels from 10 to 30 g/t depending upon the zone, grade interpolation using two-metre composites within each of the shells is accomplished by ordinary kriging. Variography identifies primary ranges of 30 to 50 metres along strike, 20 to 55 metres down-dip and seven to ten metres across the dip. Secondary ranges are 40 to 100 metres along strike, 40 to 50 metres down-dip, and 15 to 20 metres across the dip.

Since 2008, no material amounts of exploration drilling or mining have been completed at the Southwest Deposit that would require changes to the SW-2 block model. As a result, the model originally created in late 2008 has been used to estimate current Southwest Deposit mineral resources and reserves.
14.3.3 Sarytor Deposit Model (SR-2)

Following a substantial amount of in-fill drilling in 2006, the Sarytor block model identified as SR-2 was created for the estimation of mineral resources and reserves at Sarytor. A new geological model was developed, identifying ten mineralized zones, with two of the zones containing the majority of the mineral resources and reserves. After capping at 30 g/t of individual assays, grade interpolation using two-metre composites within the two main shells was accomplished by ordinary kriging. The small zones were interpolated using anisotropic inverse distance squared methods because of the lower overall drilling density. Variography identified primary ranges of 20 to 30 metres along strike, 20 to 50 metres down-dip and 7 to 10 metres across the dip. Secondary ranges are 40 to 80 metres along strike, 40 to 50 metres down-dip, and 12 to 16 metres across the dip.

Since 2006, no material amounts of exploration drilling or mining have been completed at Sarytor that would require changes to the SR-2 block model. As a result, the model originally created in late 2006 has been used to estimate current Sarytor mineral resources and reserves.

14.3.4 Northeast Deposit Model (NE-2)

The original Northeast block model (NE-1) was constructed in 2010 and inferred mineral resources for this small Northeast Deposit were first published in a Centerra press release dated February 7, 2011. An updated model was created for the 2011 year-end resource estimate due to additional drilling. The mineralization is contained within three narrow individual mineralized zones. After capping of individual assays at 30 g/t, grade interpolation using two-metre composites within each of the shells is accomplished using a single pass inverse distance interpolation with primary ranges of 60 metres along strike and down-dip and 15 metres across the dip.

14.4 Resource Classification

The mineral resource classification for the Kumtor Project into measured, indicated and inferred categories for mineral resources considered for open-pit mining is in general based on the distance to the nearest composite. If the nearest composite in the Central and the Southwest Deposits is within 30 metres, then a block is placed in the measured category. If the nearest composite is at a distance larger than 30 metres but shorter than 60 metres, then the block is placed in the indicated category. All blocks having the nearest composite at a distance greater than 60 metres are placed in the inferred category.

The proof of continuity for the mineral resources considered for underground mining at the increased cut-off grade of 6 g/t requires reduced drill spacing compared to what currently exists. Fill-in drilling in the Stockwork Zone is now partly complete, and as a result, indicated resources are reported from high-grade mineralization in this zone for the first time. However,
more stringent requirements of an increased number of composites required for grade interpolation into a block potentially considered for underground mining have been applied, as described in Section 14.3.1 above.

The search distances used at Sarytor are smaller, from 20 to 50 metres for the indicated category (first-pass interpolation), depending on the size and grade continuity of the individual zones. The inferred category was assigned to those blocks at twice the distance of the first pass. There are no measured mineral resources at Sarytor, reflecting the lack of actual mining experience for this deposit.

With limited exploration data available on the Northeast Deposit and the small size of the mineralized zones, all the mineral resources in the Northeast Deposit were classified as inferred regardless of the distance of the block from the sample.

Given the generally good grade continuity at low cut-off grades of these medium-sized to large mineral deposits, and the satisfactory results of the mineral reserve-mine-mill reconciliation described in Section 15.11 the classification approach for the Kumtor Project is in accordance with the guidelines of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definitions Standards for Mineral Resource and Mineral Reserve Definitions as required by NI 43-101.

Except as stated in Section 25 the responsible author is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the mineral resources of the Kumtor Project.
15 MINERAL RESERVE ESTIMATES

15.1 General

Mineral reserves are that part of the mineral resource that can be legally, safely and profitably mined given a specific set of technical and economic parameters. These include the gold price, mine, mill and administrative operating costs, metallurgical recovery, geotechnical behaviour of the rocks in the future pit walls, and equipment size parameters. Computer software “optimizes” the pit shape by interrogating each block of the block model as to its ability to pay for its removal plus the incremental tonnage of waste that must be removed to mine the block. Detailed mine planning using commercial software then creates a number of intermittent pit designs that test the ability to access sufficient ore to provide adequate mill feed while postponing waste mining as long as possible. This process results in the creation of one or more “pit shells” which recover the economic part of the mineral resources and which are then engineered in detail by adding ramps for mining access and by smoothing of the pit walls.

For the Kumtor Project, updated pit designs were created in 2012 and were selected from a number of alternatives investigated, with particular emphasis on geotechnical considerations as described in Section 15.3. The economic studies undertaken by KOC (summarized in Section 22), and the LOM plan subsequently adopted by Centerra (described in Section 15.12) demonstrate that the Kumtor Project mineral reserves are the “economically mineable part of a Measured or Indicated Mineral Resource” as defined by the CIM mineral Resource and mineral Reserve Definitions as required by NI 43-101, that read in part as follows:

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 ‘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.

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.

Factors affecting the Kumtor Projects mineral resources and reserves relating to geotechnical issues are discussed in Section 15.3. Except as stated in Section 25 the responsible author is not aware of any environmental, permitting, legal, title, taxation, socio-economic,
marketing, political or other relevant factors that could materially affect the mineral reserves of the Kumtor Project. Risks associated with the non-technical issues are discussed in Section 25.2.

15.2 Dilution Provisions

All of the historical block models have included a provision for internal dilution since low-grade intervals were included in the composite grades used for grade interpolation. However, the early block models up to and including model KS-5 created in 2004, did not provide for external dilution or mining losses that naturally occurs during the mining process.

For the current block models for the Central and Sarytor pits, external dilution was provided for by adding to the tonnage of each block containing more than one rock type (i.e., ore and waste) one-half of the waste tonnage in such a block. Since the bulk densities for ore and waste are identical, this represents simply a shift of the waste/ore ratio inside a block. A comparison of the two undiluted and diluted KS-13 and SR-2 models within their respective pit designs of at a cut-off grade of 0.85 g/t gold in the Central Pit and 1.0 g/t gold in the Sarytor pit is compiled in Table 10.

Table 10 External Dilution Contained in the KS-13 and SR-2 Models
Cut-off grade 0.85 g/t gold for Central Pit and 1.0 g/t gold for Sarytor Pit

<table>
<thead>
<tr>
<th></th>
<th>Tonnes ('000)</th>
<th>Grade Gold (g/t)</th>
<th>Contained Gold ('000’s ounces)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Central Pit</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Undiluted</td>
<td>73 934</td>
<td>3.6</td>
<td>8 515</td>
</tr>
<tr>
<td>Diluted</td>
<td>78 581</td>
<td>3.4</td>
<td>8 572</td>
</tr>
<tr>
<td>Ratio (Diluted/Undiluted)</td>
<td>106%</td>
<td>96%</td>
<td>101%</td>
</tr>
<tr>
<td><strong>Sarytor Pit</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Undiluted</td>
<td>7 698</td>
<td>3.1</td>
<td>767</td>
</tr>
<tr>
<td>Diluted</td>
<td>9 057</td>
<td>2.6</td>
<td>742</td>
</tr>
<tr>
<td>Ratio (Diluted/Undiluted)</td>
<td>118%</td>
<td>84%</td>
<td>97%</td>
</tr>
</tbody>
</table>

For the Central Deposit, the net effect is an increase in the total tonnage by about 6%, a grade reduction of about 4%, and a small gain of contained gold. The performance of the diluted model against actual production is discussed in Section 15.11.

For the Sarytor Deposit, the net effect is more significant with an increase in the total tonnage of about 18%, a grade reduction of about 16%, and a 3% loss of contained gold due to the thinner and more irregular size and shape of the mineralized zones. There is as yet no production from the Sarytor Deposit to directly compare to.

The Southwest SW-1 block model utilized a simple dilution faction of 5% at zero gold grade and no mining losses.
15.3 Geotechnical Issues

Figure 14 is a satellite image of the Kumtor Project mining areas taken on August 1, 2012 showing the areas affected by geotechnical issues discussed in this section.

Operations at the Central Pit have been negatively affected by two substantial failures of the northeast highwall in 2002 and 2006 and by a significant and continuing creeping of the historical waste dump and glacial ice in the SB Zone section of the Central Pit.

15.3.1 Host-Rock Related Geotechnical Issues

It is useful to remember that the host rocks in which the Kumtor gold mineralization occurs are generally of poor to very poor competency due to the multiple phases of mostly brittle deformation described in Section 7.2. Additionally, faults, joints and foliation planes are often at poor intersection angles with the pit walls which has resulted in overall pit slopes being designed at angles that have decreased from the range of 38 to 40° in earlier years to the general 30° range currently, with a concomitant increase in the strip ratio.

Golder Associates (Vancouver office) have been the long-term geotechnical and pit-slope design consultants for KOC and Centerra and have produced a number of reports listed in Section 28. The latest relevant report (Golder Associates, 2012b) addresses the slope stability question of the KS-13 pit expansion in the southern part of the Central Pit. Also, since 2006, Rob Seago of SRK Consulting (UK) Ltd. has provided assistance with the structural and geotechnical mapping of the Kumtor open pits, and his reports are also listed in Section 28. Much of the work by Golder and Seago has been incorporated into the current understanding of the rock-mechanical issues at Kumtor and has been taken into account for the current design of the Central Pit slopes.
15.3.1.1. **The Northeast High Wall**

A detailed description of the high-wall issues can be found in Section 16.2 of the previous Kumtor technical report (Redmond et al., 2011). The following is a summary.

An initial high-wall failure on July 8, 2002 resulted in the loss of a life, and the temporary suspension of operations, and led to a shortfall in 2002 production because the high-grade Stockwork Zone was rendered temporarily inaccessible. A second failure of similar magnitude occurred on July 13, 2006, in an area above the Stockwork Zone that was planned to be mined in 2006 and 2007. Although the high wall has been stable since the 2006 slide, mining from the Stockwork Zone has been deferred.

Following the second ground wall movement, KOC and Centerra continued to assess the causes of the pit wall failure and have developed remedial and long-term pit slope design criteria that would reduce the possibility of a recurrence. This work has provided insight into why the high wall failures occurred. Commencing in 2009, the slope design was substantially revised from a slope angle of 36° to a slope angle of 30° to provide a safe and stable pit wall. This design has been incorporated into the current reserve estimate. No mining has been carried out in this part of the Central Pit since 2006, and the high wall has been stable.

15.3.1.2. **Southwest Central Pit (SB Zone)**

As Figure 19 shows, the major expansion of the Central Pit will take place in its southwestern part to accommodate extraction by open-pit mining of a much larger portion of the SB Zone than had previously been considered. To reach the planned pit bottom at elevation 3 500 metres, the SB pit will ultimately attain excavated slope heights of 560 to 640 metres on the southwest, south and east walls. The maximum slope of the SB Zone final pit will be as high as 980 metres when the natural mountain slopes above the pit rim are included. The western wall will have a more modest height of 440 metres. The footprint of currently 1 500 by 1 500 metres will expand to 2 100 by 2 000 metres.

The interpretation of the geological structures in the southwest part (SB Zone) of the Central Pit (Seago, 2010) has shown that the grain of the various structural features discussed in Section 7.2 tend to have an orientation that is parallel or sub-parallel to the pit walls for structural slices 0, 1, 2 and 3B. This is shown schematically in Figure 15.

The result is that pit wall and faults, foliation and other planar structures within the very poor-quality rocks of the Kumtor Fault (structural Slice 1) and Slice 2 are essentially parallel on the western side of the pit, where extensive ravelling has occurred but has been contained by catch berms. On the eastern walls, the planar structural features are variably more or less parallel to, or dip into the pit wall, depending on the location of the pit walls with respect to folds and faults in structural Slices 3A, 3B and 3C (Figure 15).
KS13 Pit Design

Surface

original topography

Lower Kumtor Fault

Upper Kumtor Fault

Lysii Fault

3A / 3B Boundary Fault

Upper 3B Boundary Fault

D3 Fault

D2 and D4 Faults

S1 Foliation

Footwall Rocks

Kumtor Fault Zone

Ore Zone

Phyllites with Fore-Thrusts and Back-Thrusts

Slice 0

Slice 1

Slice 2

Slice 3A

Slice 3B

Slice 3C

KS13 Pit Design Surface

To prevent wall failure, pit walls have to be flat enough so that undercutting of sub-parallel structures wall does not occur, or is at least minimized. Initial analysis by Golder had indicated that a slope angle of 20° is required in areas where the structures are oriented poorly with respect to the pit walls. However, Golder noted that the rock slope angle can be steepened substantially to about 30° if depressurization is undertaken (Golder Associates Ltd, 2007). Horizontal drainage wells have been installed in some of the pit walls since 2008, with little or no flows reported by KOC staff. However, the distribution of groundwater has not been fully characterized and drilling of additional wells including piezometer installations and pressure testing will be needed to confirm that KOC will be able to control water pressures, if required.

Golder Associates, 2012b have reaffirmed the steeper slope design angles which have been used for the design of the KS-13 Central Pit. However, since the new pit expands beyond the areas for which geotechnical information is currently available, Golder have recommended a program of 14 geotechnical holes to be drilled to expand the existing structural geology model as a precondition for the determination of final slope angles for the final KS-13 pit.

If the pit wall cannot be depressurized, if the current slope design angles need significant flattening or if the pit walls prove to be unstable at the slope angles assumed for the September 30, 2012 mineral reserve estimate, there would be a noticeable negative impact on the LOM plan and on the current open-pit mineral reserves. Since the pit rim in some segments of the southern part of the Central Pit cannot be pushed back, a one-degree change in the overall slope angle will result in a 15-metre vertical change of the final pit floor elevation, with a commensurate negative change in extractable reserves.

The north-west side of the pit is currently approximately 280 metres high and the wall slopes to the southeast away from the processing plant site, which is set back 350 metres from the pit crest. The ore body dips to the south-east, also away from the plant. However the final pit crest as planned for the extraction of the KS-13 mineral reserves will be located some 140 metres away from the mill. Although the overall slope angle of the wall design in this sector at 29° is shallower than the maximum recommended slope recommended (Table 12 in Section 15.4), the wall may still be subject to deformation if the assumptions made for the slope design turn out to be invalid. Any effect of ground stability issues on the grinding mills would have grave consequences for the Kumtor Project.

15.3.1.3. The Southwest and Sarytor Pits

The Southwest and Sarytor ultimate pit designs are significantly smaller and have far less vertical extent (up to 950 metres diameter and 365 metres depth) than the Central Pit.

During mining of the Southwest pit from 2006 to 2008, no serious geotechnical issues were encountered in the mining of a 65 metre thick section of the Sarytor glacier or of the bedrock high wall. Monitoring of the Southwest pit slopes continues and no material slope movements have been observed since the suspension of mining in 2008. The wall stability to date supports the choice for a slight increase of the overall slope angles for the expanded Southwest pit as detailed in Section 15.4.
The Sarytor Pit has not had any mining production experience to date, but slope parameters described in Section 15.4 are similar to those of the Southwest pit and have been based on structural geological mapping by Seago (2007a) combined with KOC geotechnical drill-hole data.

As Figure 14 shows, the Southwest and Sarytor pits will coalesce after mining has been completed. The high walls of the two pits will cut across the ice of the Sarytor Glacier, which is retreating according to observations over the last several years. Mining of the two pits involving the ice is scheduled to take place in winter. The seasonal melt waters from the glacier will be managed by diverting the flow initially into the existing Southwest pit until mining of the Sarytor deposit is complete, and into the new Sarytor pit afterwards, when the Southwest Deposit is being mined. Any overflow needs to be pumped and diverted around the future Sarytor Valley waste dumps.

15.3.2 Glacier-Related Issues

The KS 13 final pit intersects two glaciers, the Lysii glacier at its northeastern end and the Davidov Glacier at its eastern and southern ends, while the Sarytor pit intersects the terminal part of the Sarytor Glacier. The interaction between the current and future open pits and the glaciers, and the challenges arising from this interaction, are described below.

From previous experience there are issues associated with mining glacier ice. These include voids in the ice, decreased mining productivity compared to waste rock and water management of the surface flows especially during summer time. Mitigating measures to address these issues include the use of ultrasound radar for cavity detection, the allocation of adequate capacity for haulage and the adequate management of melt water.

15.3.2.1. The Davidov Glacier

Davidov Glacier consists of four arms identified as “East Arm 1”, “East Arm 2”, “SE Arm” and “South Arm”, from north to south (Figure 16). Before commencement of mining at Kumtor, there was only one East Arm. Pre-stripping of the Central Pit began in 1995 and from that time until the discovery of the SB Zone in 2005, the majority of the waste rock and sections of the Davidov Glacier were mined and deposited on and along the lateral margins of the Davidov Glacier in an effort to push the flow of the remaining ice away from the crest of what was then the ultimate pit design (upper photo Figure 4). The intent was to displace the ice and form a rock-fill buffer between the flowing ice and the active mining area. As a result, a substantial amount of waste rock had been dumped directly onto Davidov Glacier adjacent to the pit (Figures 4 and 16). This resulted in the gradual displacement of the glacier ice away from the pit such that in some areas, waste rock that was originally dumped on the glacier, came to rest on the basal moraine (till) of the glacier.
Interaction of Davidov Glacier with Central Pit

Pre-mining flow path of South East Arm of Davidov Glacier

Pre-mining flow path of South Arm of Davidov Glacier

2002 & 2006 Failures

High Movement Area

2012 flow paths of Davidov Glacier

Processing Plant

KS13 Pit Design

Lysii Glacier

East Arm 1
Davidov Glacier

East Arm 2
Davidov Glacier

South East Arm
Davidov Glacier

South Arm
Davidov Glacier

Geosat satellite photo August 1, 2012
Since 2009, the performance of the open-pit operation and subsequent gold production has been negatively affected by the creep movement of a section of the historical waste dump and the Davidov Glacier ice below. A section of the historical waste dump and original glacial ice approximately 1 100 metres long was found to be moving into the Central Pit. The rates of movements varied from a low of 5 mm/hr (4 m per month) to as high as 80 mm/h (60 m/month). A snapshot of the ice movements of the Davidov South arm area is presented in Figure 17.

The situation is a unique geotechnical problem that to the best knowledge of the authors has no parallel. The main effects have been the repeated delay of mining of some of the high-grade parts of the SB Zone and an increased overall stripping ratio of the pit. There has been no loss of mineral reserves.

Significant efforts have been made to manage the influx of waste dump and glacier ice material creeping into the open pit. This has included the continuing removal of the historical waste dump material and an intensive dewatering program intercepting the water flowing along the ice-till contact. Late in 2011, however, the movement accelerated again during a time of year when such movement in the past was typically retarded because of the cold weather, and has continued at a pace that has made removal of the remaining waste and ice in this part of the pit a pre-condition for safe mining below (Figure 16). As a consequence, the original 2012 mine plan had to be altered to allow complete removal of the loose waste and underlying and intermixed disturbed ice before resuming mining of the waste rock below that is necessary to expose the SB Zone mineralization for mining. These actions have interrupted ore mining from the Central Pit for a period of three months in 2012 and have postponed the mining of high-grade SB Zone ore into 2013. The processing plant was shut down from July 24 to September 18, 2012, after all stockpiles had been exhausted.

A background study on the Davidov Glacier has recently been completed by Golder Associates, 2012c who have, on the basis of available data, developed “base-case” estimates of ice-wall thickness and movement of the undisturbed ice as follows:

**Table 11 Base-Case Estimates of Ice-Wall Thickness and Ice Movement**

<table>
<thead>
<tr>
<th></th>
<th>Unit</th>
<th>East Arms 1 &amp; 2</th>
<th>Southeast Arm</th>
<th>South Arm</th>
</tr>
</thead>
<tbody>
<tr>
<td>Thickness</td>
<td>metres</td>
<td>100</td>
<td>100</td>
<td>120</td>
</tr>
<tr>
<td>Movement Rate</td>
<td>mm/hr</td>
<td>10</td>
<td>32</td>
<td>30</td>
</tr>
<tr>
<td></td>
<td>m/year</td>
<td>90</td>
<td>280</td>
<td>260</td>
</tr>
</tbody>
</table>

Table 11 shows modelled movement rates for the cross-sectional centre where ice is the thickest. As ice becomes shallower at the edges of the glacier, the modelled movement rates tend to decrease. Average modelled rates for each glacier arm as a whole have been predicted to comprise roughly 50% of the rates indicated in Table 11. It is these reduced rates that were used for the estimates of future ice movements.
A radar survey to measure ice thickness of the various arms of Davidov Glacier was carried out from September 30 to October 3, 2012 (Chernomoret s et al., 2012). This has confirmed the ice thickness assumptions by Golder which had been based on an earlier and less detailed radar survey undertaken during the historical Soviet-era exploration of the Kumtor Project.

To keep glacier ice and related melt water from entering the Central Pit, KOC plans to deal with the undisturbed Davidov Glacier as follows:

- The ice adjacent to the pit wall will be mined back to a distance of 150 to 200 metres above every active cut-back (CB) as indicated for the South Arm in Figure 18. This will amount to 19 million tonnes (Mt) of ice in the remainder of 2012 followed by 40 Mt, 14 Mt and 10 Mt in the years 2013 to 2015, respectively;
- Engineered and lined sumps will be constructed in three locations in front of the excavated glacier ice above the pit to collect melt water. The sumps will be connected to dewatering pipes that divert the collected melt water around the Central Pit.

The southern part of the Central Pit will be mined out at the beginning of 2018, and if the glacier continues to creep, the ice will then be allowed to fill the excavated areas above the pit and the southwest end of the Central pit itself. Should, Centerra decide to establish new underground access in the bottom of the SB zone pit to potentially develop the underground resources in the SB zone, additional open pit mining capacity maybe required to further control the creep of glacial ice and melt water into this section of the pit and the costs associated with this work will have to be borne by the underground project.

The knowledge to plan a successful campaign of mining ice at a rate that prevents it from entering the Central Pit until the pit is mined out in 2018 is currently inadequate. While the volume of the glacier ice near the Central Pit is reasonably well known, the rate at which the ice will be advancing toward the pit remains essentially unknown. The figures for ice mining that have been considered for the LOM plan described in Section 15.12 reflect the base case as laid out in Table 20 and include an allowance of 50% for additional ice moving into the pit. If the worst-case scenarios were to occur, ice mining capacities would have to be 2.5 to 3 times higher than currently planned.

Additional information will be gained from upgrading of the ice monitoring system on all four arms of the Davidov Glacier. This should provide early warning should the base-case assumptions about ice movement rates turn out to be low.

From the preceding, it is obvious that the rate at which glacier ice mining has to be accomplished during the operation of the southern part of the Central Pit is currently uncertain. The volumes incorporated into the KS 13 LOM plan, and the additional mining equipment required to accomplish this, are therefore subject to upward revision, possibly in a substantial way. Should ice mining not keep up with the forward ice movement, interruptions to the LOM plan with respect to mining of the high-grade SB Zone would occur, with well-known negative implications.
15.3.2.1. The Lysii Glacier

The Lysii glacier, which initially flowed directly over the open-pit northeastern highwall, was mined beyond the open-pit footprint in the early years of the operation. However, a tongue or “snout” of this glacier continues to be intersected by the current pit at its northernmost point. The situation of the Lysii glacier is advantageous since its main movement is away from the open pit due to the bedrock sloping away from the pit in this area.

Historically, mining of the Lysii Glacier has not posed any significant operational difficulties. The ice benches have proven to stand up well with no creep deformations. Safe mining was achieved by placing a 0.5-metre thick layer of waste material on top of each working area for the safe movement of mining equipment. The LOM plan includes the complete removal of the Lysii cap. This was also one of the critical requirements of Golder Associates, 2012b to prevent melt water from seeping into the pit slopes below.

The total volume of ice to be excavated on the Lysii Glacier comprises approximately 9 million tonnes. The majority of the ice is planned to be deposited into the NE valley separately from the waste dumps (NE of the Lysii Glacier, as shown on Figure 14).

15.3.2.2. The Sarytor Glacier

Only the terminal 200 metres of Sarytor glacier will be mined, and no substantial movement of the ice into the constructed pits is expected into the Southwest or Sarytor pits. A total of approximately 2 million tonnes of ice will be mined. Initial experience of mining the Sarytor glacier was gained with cut-back 1 of SW pit. It is expected that similar challenges will be faced during future mining of the Sarytor glacier. These include the inflow of melt water during the warm months and the necessity to manage it, low productivity while mining ice and till, and the working conditions of the ice benches. The ice mined will be deposited on a separate ice dump that will be located in the same Chon-Sarytor Valley.

15.3.3 Geotechnical Summary

The Central Pit is a very large man-made opening in a structurally, hydrologically and glaciologically complex setting. Two large failures in the rocks of the northeast highwall have in the past led to loss of life and significant production delays. In addition, high deformation rates in the southeast wall of the Davidov Glacier ice and accumulated waste piles on the ice have been experienced. All of these incidents have had economically significant and unforeseen negative effects due to the temporary unavailability for mining and processing of high-grade ore from the Stockwork and SB Zones. There has been no loss of mineral reserves.

The structural geology of the high wall has been mapped and interpreted by Rob Seago of SRK UK, an independent structural geologist. The structural failure mode of the two previous wall failures appears to be understood, with water seeping into the slope from the Lysii glacier above considered a contributing factor. To create a stable final pit wall, flattening of the high wall
is planned with the intent to mine out all of the known wedges, or at least to prevent them from daylighting. Pit-wall stability is strongly influenced by the level of water saturation of the rock, with dry conditions being more stable than wet, saturated conditions. To mitigate against glacier water entering the pit wall, the snout of Lysii glacier will be mined in 2014 and 2015 to minimize the amount of melt water seeping into the pit walls. In addition, horizontal drainage wells will be installed to depressurize the rocks in the wall as part of the waste mining in this area. Piezometer installations and pressure testing must be carried out to confirm that these mitigation measures are effective and adequate and that KOC will be able to depressurize the high wall, as necessary.

With respect to the southeast and south walls of the Central Pit removal of waste rock overlying the Davidov Glacier and of the ice in the high movement area will continue until 2015. Continued dewatering of the ice-till contact and depressurization of the bedrock below will be required until this part of the Central Pit is mined out in early 2018.

Experience since 2008 indicates that dewatering of the ice-till contact is possible but is not without challenges, because the high ice movement rates tend to cause the pumping wells installed through the glacier ice to become dysfunctional, shearing off after sometimes only a few days of operation. The solution has been to re-drill and install replacement wells where necessary as well as to divert surface water flows around the open pit by means of sumps, trenches and pipes.

Updated pit-slope angles for the KS 13 Ultimate Pit have been provided by Golder Associates, 2012 and have been incorporated into the pit design which in turn determines the amount of ore and waste that will form part of the new LOM plan. However, these slope angles presume essentially dry (depressurized) wall-rock conditions and are preliminary until additional geotechnical drilling can be completed and evaluated to extend the structural geology model into the expanded pit. Even a moderate flattening of the design slope angles will have a negative effect on the recoverable reserves by open-pit mining particularly of the SB Zone.

The expanded design of the KS13 Ultimate Pit will necessitate the prior and continuing removal of the advancing arms of Davidov Glacier. The current LOM plan is based on incomplete information with respect to ice advance rates with the risk that substantially larger efforts than currently planned for will be required to keep the glacier ice from moving into the active Central Pit and from disrupting the exploitation of the SB Zone. The Kumtor mining permits, subject to annual renewal as more fully described in Section 20.1, have always been issued in the past with the full knowledge of the Kyrgyz authorities that removal of glacier ice is part of the Kumtor mining operation, and it is expected, but not guaranteed, that future annual mining permits will be issued without a change of policy.

As part of undertaking the assignment of producing this report, the responsible author has reviewed the final pit designs for the Central, Sarytor and Southwest pits produced by KOC that contain the September 30, 2012 mineral reserves of the Kumtor Project. The author concludes, subject to the reservations expressed above, that the designs are reasonable and achievable, based on the current knowledge and understanding of all features and parameters affecting their future stability. However, the geotechnical risks affecting the Central Pit remain to be completely understood and mitigated. If, contrary to expectations, mitigation is not fully successful, then
further disruptions of the LOM mine plan and, as a worst case, losses of part of the Central Pit mineral reserves, may be experienced. Mining of the smaller Sarytor and Southwest pits are not expected to present serious geotechnical challenges, based on experienced gained to date from mining of the Southwest pit.

### 15.4 Physical Pit Design Parameters

The design of the Central Pit is subject to geotechnical considerations that have received a great deal of attention following the two highwall failures in 2002 and 2006, as described in Section 15.3, and the more recent geotechnical issues related to the mining of the historical waste dump and glacial ice above the SB Zone. As a result of extensive geotechnical studies by KOC and its consultants, the Central Pit has been sub divided into five major design sectors namely the footwall (Slice 1), Ore (Slice 2) hanging wall (Slice 3 with a number of smaller slices), Waste Dump and Glacial Till. The slope design parameters for the individual sectors are summarized in Tables 12, 13 and 14.

For the Central Pit, the slope design angles reflect recommendations in Golder Associates, 2012b. It should be noted that Golder calls the recommended slope angles “preliminary” until additional geotechnical drilling can be completed into the walls into which particularly the southwestern part of the Central Pit is going to expand. The resulting ultimate pit crests, part of the ultimate waste dump limits and the crests of the September 30, 2012 pits for the Central, Sarytor and Southwest Pits are shown in Figure 19 and Figure 20.
<table>
<thead>
<tr>
<th>Pit Sector</th>
<th>Wall Design Azimuth (Degrees From North)</th>
<th>Maximum Inter-Ramp Angle</th>
<th>Maximum Slope Angle</th>
<th>Maximum Berm Width (m)</th>
<th>Bench Height (m)</th>
<th>Berm Batter Angle</th>
</tr>
</thead>
<tbody>
<tr>
<td>Zone-1 Footwall</td>
<td>00 to 280</td>
<td>36</td>
<td>36</td>
<td>12.5</td>
<td>10.0</td>
<td>53</td>
</tr>
<tr>
<td></td>
<td>280 to 360*</td>
<td>32-34</td>
<td>31</td>
<td>14.5-17</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zone 2 Ore Zone</td>
<td>00 to 30</td>
<td>36</td>
<td>36</td>
<td>12.5</td>
<td>10.0</td>
<td>53</td>
</tr>
<tr>
<td></td>
<td>30 to 130</td>
<td>32</td>
<td>32</td>
<td>17-19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>130 to 280</td>
<td>32</td>
<td>32</td>
<td>17-19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>280 to 360*</td>
<td>32-34</td>
<td>31</td>
<td>14.5-17</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zones 3A, 3C Hanging Wall</td>
<td>00 to 80</td>
<td>30</td>
<td>30</td>
<td>19.5</td>
<td>10.0</td>
<td>53</td>
</tr>
<tr>
<td></td>
<td>80 to 360</td>
<td>32</td>
<td>32</td>
<td>17-19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zone 3B1 Hanging Wall</td>
<td>00 to 70</td>
<td>30</td>
<td>30</td>
<td>19.5</td>
<td>10.0</td>
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<td></td>
<td>70 to 100</td>
<td>34</td>
<td>32</td>
<td>14.5-17</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>100 to 160</td>
<td>32</td>
<td>30</td>
<td>17-19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>160 to 360</td>
<td>30</td>
<td>30</td>
<td>19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Zone 3B2 Hanging Wall</td>
<td>00 to 80</td>
<td>32</td>
<td>30</td>
<td>17-19.5</td>
<td>10.0</td>
<td>53</td>
</tr>
<tr>
<td></td>
<td>80 to 130*</td>
<td>32</td>
<td>30</td>
<td>17-19.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>130 to 195*</td>
<td>36-32 (top to bottom)</td>
<td>34</td>
<td>12.5-17</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>195 to 360</td>
<td>36</td>
<td>36</td>
<td>12.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Waste Dump</td>
<td>00 to 90</td>
<td>36</td>
<td>36</td>
<td>12.5</td>
<td>10.0</td>
<td>53</td>
</tr>
<tr>
<td></td>
<td>90 to 130 (HMA)</td>
<td>26</td>
<td>26</td>
<td>26.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>130 to 200</td>
<td>31</td>
<td>31</td>
<td>18.2</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>200 to 360</td>
<td>38</td>
<td>38</td>
<td>10.5</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Glacial Till</td>
<td>00 to 360</td>
<td>38</td>
<td>38</td>
<td>10.5</td>
<td>10.0</td>
<td>53</td>
</tr>
</tbody>
</table>

Note: HMA = high movement area. Berm-width is indicated for double-benching design.
* Single-benching used for the slope sectors as proposed by Golder Associates, 2012b

The Sarytor Pit will be relatively small compared to the Central Pit. Initial design parameters were developed by the KOC geotechnical staff with input by SRK and Golder and were further updated in 2009 by Centerra’s geotechnical engineer. The design angles used for the September 30, 2012 mineral reserve estimate for the Sarytor pit are summarized in Table 13.
Table 13 Sarytor Pit Physical Design Parameters

<table>
<thead>
<tr>
<th>Wall Design Azimuth (Degrees From North)</th>
<th>Maximum Inter-Ramp Angle</th>
<th>Maximum Slope Angle</th>
<th>Maximum Berm Width (m)</th>
<th>Bench Height (m)</th>
<th>Berm Batter Angle</th>
</tr>
</thead>
<tbody>
<tr>
<td>00 to 20</td>
<td>30</td>
<td>30</td>
<td>19.6</td>
<td></td>
<td></td>
</tr>
<tr>
<td>20 to 165</td>
<td>43</td>
<td>43</td>
<td>9.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>165 to 210</td>
<td>40-43</td>
<td>40-43</td>
<td>9.0-10.9</td>
<td>4.0</td>
<td>63.0</td>
</tr>
<tr>
<td>210 to 255</td>
<td>38</td>
<td>38</td>
<td>12.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>255 to 295</td>
<td>43</td>
<td>43</td>
<td>9.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>295 to 360</td>
<td>30</td>
<td>30</td>
<td>19.6</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Till</td>
<td>36</td>
<td>36</td>
<td>13.9</td>
<td>4.0</td>
<td>63.0</td>
</tr>
</tbody>
</table>

The Southwest Pit will also be relatively small compared to the Central Pit. Initial design parameters were developed by the KOC geotechnical staff with input by SRK and Golder and were further updated in 2009 by Centerra’s geotechnical engineer. The design angles used for the September 30, 2012 mineral reserve estimate for the Southwest pit are summarized in Table 14.

Table 14 Southwest Pit Physical Design Parameters

<table>
<thead>
<tr>
<th>Wall Design Azimuth (Degrees From North)</th>
<th>Maximum Inter-Ramp Angle</th>
<th>Maximum Slope Angle</th>
<th>Maximum Berm Width (m)</th>
<th>Bench Height (m)</th>
<th>Berm Batter Angle</th>
</tr>
</thead>
<tbody>
<tr>
<td>10-330</td>
<td>38</td>
<td>38</td>
<td>12.3</td>
<td>4.0</td>
<td>63.0</td>
</tr>
<tr>
<td>330-10</td>
<td>40</td>
<td>40</td>
<td>10.9</td>
<td>4.0</td>
<td>63.0</td>
</tr>
</tbody>
</table>

15.5 Economic Pit Parameters

The Kumtor Project mineral reserves available for mining at September 30, 2012 were estimated by Dan Redmond, and John Baker, Mine Manager, KOC on the basis of the KS-13 block model for the Central Deposit and the SR-2 and SW-1 block models for the Sarytor and Southwest satellite deposits. There were no mineral reserves estimated for the Northeast Deposit, since all of the resources are in the inferred class. The pit design parameters described in Sections 15.4 and 15.5 were used, and the main economic parameters are summarized in Table 15.
Table 15 Economic Design Parameters, Central and Sarytor and Southwest Pits

<table>
<thead>
<tr>
<th></th>
<th>2012 Actual</th>
<th>Central Pit</th>
<th>Sarytor and Southwest Pits</th>
</tr>
</thead>
<tbody>
<tr>
<td>Gold Price $/ounce</td>
<td>$1,350</td>
<td>$1,350</td>
<td>$1,350</td>
</tr>
<tr>
<td><strong>Mining</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average cost per tonne of ore mined</td>
<td>$1.47</td>
<td>$1.55</td>
<td>$2.35</td>
</tr>
<tr>
<td>Average cost per tonne of waste mined</td>
<td>$1.47</td>
<td>$1.55</td>
<td>$1.50</td>
</tr>
<tr>
<td><strong>Milling</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average cost per tonne milled</td>
<td>$12.92</td>
<td>$11.75</td>
<td>$11.75</td>
</tr>
<tr>
<td><strong>Administration</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Average cost per tonne milled</td>
<td>$15.47</td>
<td>$9.89</td>
<td>$9.89</td>
</tr>
<tr>
<td><strong>Metallurgical Recoveries</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Grade Range (g/t)</td>
<td>Recovery (%)</td>
<td>Recovery (%)</td>
<td></td>
</tr>
<tr>
<td>&gt; 5.0 g/t</td>
<td>75 to 88 %</td>
<td>74 to 80 %</td>
<td></td>
</tr>
<tr>
<td>3.0 to 5.0 g/t</td>
<td>65 to 83 %</td>
<td>60 to 74 %</td>
<td></td>
</tr>
<tr>
<td>0.85 to 3.0 g/t</td>
<td>47 to 76 %</td>
<td>40 to 73 %</td>
<td></td>
</tr>
</tbody>
</table>

(*) Unit mining costs for 2012 equal operating mining costs plus capitalized pre-stripping mining costs

The average projected unit mining costs are higher than the actual 2012 year to date costs as a result of increased assumptions on fuel costs for 2013 and the increasing length of the average haulage distance over the LOM. The average projected unit milling and administrative costs (which are both expressed as a costs per tonne milled) are both lower than the actual 2012
year to date costs due the negative impact of the mill shutdown for 6 weeks during 2012 which resulted in an abnormal increase in units costs for 2012.

Metallurgical recoveries are determined from the results of the bench composite assaying and testwork described in Section 16.2, individually for each mineralized zone. The recovery values are assigned to specific grade ranges and mineralized zones within the block model. The recoveries include the full impact of the ISA mill described in Section 17.1, which has led to a small incremental improvement of the gold recovery since its installation in 2005. The actual performance of the mill since 1999, when initial recovery problems had been overcome, is shown in the inset in Table 15.

15.6 Pit Optimization and Pit Design

To define the mineral reserves for the Kumtor Project Centerra undergoes a three step process as follows;

1. The Central, Southwest and Sarytor Deposits underwent a process of “pit optimization” where Whittle computer software optimizes the potential future financial return for a number of intermittent pit shells and defines the ultimate pit size and shape for each of the three deposits. To complete this optimization, geotechnical limitations that define the maximum slope angles that can be achieved in each pit, and economic parameters including a range of gold prices, applicable royalty/revenue based taxes, mine, mill, administrative operating costs and metallurgical recoveries are included with the measured and indicated resource blocks of the models. The software then interrogates each block of the block model as to its ability to pay for its removal and required processing/administrative costs and net revenue taking into account the incremental tonnage and associated costs of waste that must be removed to mine this block. The pit shell offering the best economic results was chosen and defines in a general way the size and shape of the ultimate pit which achieves the maximum financial return based on the defined parameters while maintaining the geotechnical limitations;

2. With the ultimate pit limits defined, practical design parameters such as haulage ramps, safety berms and practical interim cutback limits based on the overall size of the pit and the equipment to be used are completed within the GEMS software package. This process results in a series of minable cutbacks that together form the ultimate pit design for each of the deposits;

3. With each cut-back designed, as series of potential production schedules are produced based on the practical sequencing of each cut-back, the mining equipment available or available with additional capital expenditures, the practical limitations of the amount of mining equipment that can be operated on each cut-back, the production rates of equipment in different material types (ice, waste dump and bedrock), the haulage distance to waste dumps, ore stockpiles or to the mill crusher and the limitations of the throughput capacity of the mill itself.
From this process, which in most cases is iterative, a practical LOM production schedule is developed that tries to maximize gold production and minimize costs and defines the annual mining, milling and gold production schedules.

15.7 Mineral Reserve Classification

The mineral reserve classification will normally reflect the original mineral resource classification, with measured mineral resources becoming proven mineral reserves and indicated mineral resources becoming probable mineral reserves. However, as discussed in Section 15.3, both the high wall and the creep movement of a section of the historical waste dump adjacent to the Central Pit have remaining geotechnical uncertainties that constitute a certain risk for the eventual recovery of part of the mineral reserves. For this reason Centerra has chosen a more conservative approach to mineral reserve classification whereby only mineral reserves currently in a stockpile are classified as proven and the remaining mineral reserves that are in-situ are classified as probable. Given the history of geotechnical issues negatively impacting gold production from 2002 to 2012, the responsible author of this section believes this reclassification to be prudent and reasonable.

15.8 Cut-Off Grades

The mineral reserves and resources of the Kumtor Project are reported at cut-off grades of 0.85 g/t gold for the Central Deposit and for the current ore stockpiles. Given the extended ore haulage distances and lower expected mill recoveries, a cut-off grade of 1.0 g/t gold for the Sarytor Southwest and Northeast Deposits is used. The cut-off grades are calculated assuming the economic parameters outlined in Table 15 and include a provision for the payment of the Gross Proceeds Tax to the Government of Kyrgyz Republic described in Section 22.2.

15.9 September 30, 2012 Mineral Reserve Estimate

As the data in Table 6 have shown, the estimate of the total mineral reserve tonnage for the Central Deposit has changed significantly over time in response to variations in the economic parameters. The current estimate for the Central, Sarytor and Southwest Deposits at a gold price of $1 350 per ounce is summarized in Table 16. The in-pit mineral reserves are those quoted by the LOM plan developed in September 2012, and reflect the mineral reserve status as of September 30, 2012.

Figures 21 to 24 show the block model and mineral reserve and resource information for the four geology sections presented as Figures 8 to 11.
### Table 16 Kumtor Project Mineral Reserves as of September 30, 2012
Thousands of tonnes of ore and waste, thousands of ounces

<table>
<thead>
<tr>
<th></th>
<th>Tonnes</th>
<th>Gold Grade (g/t)</th>
<th>Contained Gold (Ounces)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>By Category</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Stockpiles (&gt; 0.85 g/t)</td>
<td>233</td>
<td>1.8</td>
<td>13</td>
</tr>
<tr>
<td>Total Proven Mineral Reserves</td>
<td>233</td>
<td>1.8</td>
<td>13</td>
</tr>
<tr>
<td>Probable</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Central Pit in-situ (&gt; 0.85 g/t)</td>
<td>78 581</td>
<td>3.4</td>
<td>8 572</td>
</tr>
<tr>
<td>Sarytor Pit in-situ (&gt; 1.0 g/t)</td>
<td>9 057</td>
<td>2.6</td>
<td>742</td>
</tr>
<tr>
<td>Southwest Pit in-situ (&gt; 1.0 g/t)</td>
<td>5 210</td>
<td>2.4</td>
<td>403</td>
</tr>
<tr>
<td>Total Probable Mineral Reserves</td>
<td>92 848</td>
<td>3.3</td>
<td>9 717</td>
</tr>
<tr>
<td>Total Mineral Reserves</td>
<td>93 081</td>
<td>3.3</td>
<td>9 730</td>
</tr>
<tr>
<td><strong>By Deposit</strong></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Central Pit</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven Stockpiles</td>
<td>233</td>
<td>1.8</td>
<td>13</td>
</tr>
<tr>
<td>Probable (in-situ)</td>
<td>78 581</td>
<td>3.4</td>
<td>8 572</td>
</tr>
<tr>
<td>Waste</td>
<td>1 548 397</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>19.7</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sarytor Pit</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Probable (in-situ)</td>
<td>9 057</td>
<td>2.6</td>
<td>742</td>
</tr>
<tr>
<td>Waste</td>
<td>156 404</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>17.3</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Southwest Pit</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Probable (in-situ)</td>
<td>5 210</td>
<td>2.4</td>
<td>403</td>
</tr>
<tr>
<td>Waste</td>
<td>81 688</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>15.7</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Project Total</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Proven Stockpiles</td>
<td>233</td>
<td>1.8</td>
<td>13</td>
</tr>
<tr>
<td>Probable (in-situ)</td>
<td>92 848</td>
<td>3.3</td>
<td>9 717</td>
</tr>
<tr>
<td>Total Mineral Reserves</td>
<td>93 081</td>
<td>3.3</td>
<td>9 730</td>
</tr>
<tr>
<td>Waste</td>
<td>1 786 489</td>
<td></td>
<td></td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>19.2</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Notes: Figures may not add due to rounding.
The strip ratio (S/R) is calculated on in-situ materials.
Current Pit (September 30, 2012)
SOUTHWEST PIT
(September 30, 2012)

Original Topography

Lower Kumtor Fault
Upper Kumtor Fault
Lysii Fault
3A / 3B Boundary Fault
Upper 3B Boundary Fault
D3 Fault
D2 and D4 Faults
S1 Foliation
Direction of Thrusting
Drillhole Trace

See Figure 12 for Section Line Location
Source: Map and data provided by SRK UK and KOC
15.10 Reconciliation with 2011 Year-End Mineral Reserve Estimate

The Kumtor Project proven and probable mineral reserves at the end of 2011 were summarily reported in a Centerra press release dated February 9, 2012 and stood at 59.7 million tonnes with an average gold grade of 3.3 g/t. Table 17 provides a comparison between the December 31, 2011 and the September 30, 2012 estimates.

Table 17 Comparison of December 31, 2011 and September 30, 2012 Mineral Reserves and Additional Mineral Resources

<table>
<thead>
<tr>
<th></th>
<th>December 31, 2011</th>
<th>September 30, 2012</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>Tonnes ('000)</td>
<td>Grade Au (g/t)</td>
</tr>
<tr>
<td>Proven Stockpiles</td>
<td>3 023</td>
<td>1.6</td>
</tr>
<tr>
<td>Probable in situ</td>
<td>56 671</td>
<td>3.4</td>
</tr>
<tr>
<td>Total Mineral Reserves</td>
<td><strong>59 694</strong></td>
<td><strong>3.3</strong></td>
</tr>
<tr>
<td>Waste</td>
<td>1 199 476</td>
<td></td>
</tr>
<tr>
<td>Strip Ratio</td>
<td>20.1</td>
<td></td>
</tr>
<tr>
<td>Additional Measured and Indicated Mineral Resources (Open Pit)</td>
<td></td>
<td></td>
</tr>
<tr>
<td></td>
<td>65 949</td>
<td>2.3</td>
</tr>
<tr>
<td></td>
<td>34 486</td>
<td>2.4</td>
</tr>
</tbody>
</table>

The current increase in the open pit reserves is contained entirely within the Central Pit. The successful exploration drilling of the SB Zone over the last three years has doubled the strike length of the SB Zone and extended the SB Zone resource down dip, resulting in an expansion of the resources. The resource expansion, in conjunction with the decision made in March 2012 to unload the ice above the Southeast highwall of the Central Pit, has created the opportunity to expand the Central Pit with the resulting significant increase in reserves and the extension of the mine life. Table 18 summarizes the principal additions and deletions that explain the difference between the December 31, 2011 and September 30, 2012 mineral reserve estimates.
Table 18 Reconciliation between December 31, 2011 and September 30, 2012 Reserve Estimates

<table>
<thead>
<tr>
<th></th>
<th>Tonnes ('000)</th>
<th>Au (g/t)</th>
<th>Ounces ('000)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2011 year-end reserves</td>
<td>59 694</td>
<td>3.3</td>
<td>6 278</td>
</tr>
<tr>
<td>Milled 2012 (1-9)</td>
<td>3 209</td>
<td>1.7</td>
<td>172</td>
</tr>
<tr>
<td>Depleted 2011 reserves</td>
<td>56 485</td>
<td>3.4</td>
<td>6 106</td>
</tr>
<tr>
<td>September 30, 2012 reserves</td>
<td>93 081</td>
<td>3.3</td>
<td>9 730</td>
</tr>
<tr>
<td>Increase in reserves</td>
<td>36 596</td>
<td>3.1</td>
<td>3 624</td>
</tr>
<tr>
<td>SB underground resources now in KS-13 pit</td>
<td>2 669</td>
<td>14.2</td>
<td>1 220</td>
</tr>
<tr>
<td>SW Zone underground resources now in KS-13 pit</td>
<td>313</td>
<td>10.5</td>
<td>105</td>
</tr>
<tr>
<td>Conversion of additional open-pit resources</td>
<td>28 906</td>
<td>2.2</td>
<td>2 063</td>
</tr>
<tr>
<td>Additions (new drilling)</td>
<td>4 708</td>
<td>1.6</td>
<td>235</td>
</tr>
<tr>
<td>Total additions</td>
<td>36 596</td>
<td>3.1</td>
<td>3 624</td>
</tr>
</tbody>
</table>

15.11 Accuracy of the Mineral Reserve Models

The open-pit production history from the Central Pit and to a lesser extent from the Southwest pit together with the virtual exhaustion of all stockpiles as of September 30, 2012 provides an opportunity to review the performance of the mineral reserve estimation process with actual production data. Table 19 outlines the reconciliation where the predictions of the two block models (KS-13 and SW-2) used for the September 30, 2012 mineral reserve estimate are compared to the grade control models (used to define dig limits during mining operations) and actual mill feed for the years 2004 to 2012.

It should be borne in mind that a direct comparison between block model predictions and mill feed actuals are not possible even on an annual basis due to the delay in milling the low-grade stockpiles that have been accumulating over the years and were only treated when a shortage of pit ore occurred. For the reconciliation of annual production figures with the reserve block models it is therefore necessary to use the grade-control model data as a proxy for the mill, since they are in close agreement with the mill figures.

The comparison between the prediction by the block models and the actual production figures as determined by the grade control models of annual gold grades, ore tonnes and contained ounces of gold for the years 2004 to 2012 are shown in Figure 25. The individual variances between the block models and the grade control models range from 1% to 10% for both tonnage and grade with no obvious bias. The exception are the data for 2004 when the block model under-estimated both tonnage and gold grade by a substantial margin. The reason for this particular discrepancy is unknown. For the total milled, the variances are very small, with the block model a little lower than the mill.
Table 19 Comparison of Grade Control Model and Ore Milled,  
2004 to September 30th, 2012
Thousands of Tonnes and Ounces of Gold

<table>
<thead>
<tr>
<th>Year</th>
<th>CoG (Au – g/t)</th>
<th>KS-13 Block Model</th>
<th>Grade Control Model</th>
<th>SW-2 Block Model</th>
<th>Grade Control Model</th>
<th>Reserve Block Models</th>
<th>Grade Control Models</th>
<th>Mill Production</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Tonnes</td>
<td>Gold Grade (g/t)</td>
<td>Gold Ounces</td>
<td>Tonnes</td>
<td>Gold Grade (g/t)</td>
<td>Gold Ounces</td>
<td></td>
</tr>
<tr>
<td>2004</td>
<td>1.3</td>
<td>2 974</td>
<td>5.3</td>
<td>503</td>
<td>3 428</td>
<td>6.2</td>
<td>687</td>
<td></td>
</tr>
<tr>
<td>2005</td>
<td>1.3</td>
<td>6 636</td>
<td>3.2</td>
<td>653</td>
<td>6 134</td>
<td>3.3</td>
<td>645</td>
<td></td>
</tr>
<tr>
<td>2006</td>
<td>1.3</td>
<td>3 337</td>
<td>2.7</td>
<td>293</td>
<td>2 903</td>
<td>2.5</td>
<td>236</td>
<td></td>
</tr>
<tr>
<td>2007</td>
<td>1.0</td>
<td>4 334</td>
<td>2.2</td>
<td>302</td>
<td>3 727</td>
<td>2.3</td>
<td>278</td>
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</tr>
<tr>
<td>2008</td>
<td>1.0</td>
<td>4 791</td>
<td>4.4</td>
<td>670</td>
<td>4 967</td>
<td>4.2</td>
<td>676</td>
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</tr>
<tr>
<td>2009</td>
<td>1.0</td>
<td>4 567</td>
<td>4.9</td>
<td>714</td>
<td>4 464</td>
<td>4.7</td>
<td>671</td>
<td></td>
</tr>
<tr>
<td>2010</td>
<td>0.85</td>
<td>6 225</td>
<td>3.9</td>
<td>785</td>
<td>5 765</td>
<td>4.1</td>
<td>766</td>
<td></td>
</tr>
<tr>
<td>2011</td>
<td>0.85</td>
<td>5 683</td>
<td>3.6</td>
<td>665</td>
<td>6 020</td>
<td>3.5</td>
<td>675</td>
<td></td>
</tr>
<tr>
<td>2012</td>
<td>0.85</td>
<td>547</td>
<td>2.0</td>
<td>35</td>
<td>491</td>
<td>2.1</td>
<td>33</td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>39 094</td>
<td>3.7</td>
<td>4 620</td>
<td>31 899</td>
<td>3.8</td>
<td>4 667</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Year</th>
<th>CoG (Au – g/t)</th>
<th>Reserve Block Models</th>
<th>Grade Control Models</th>
<th>Mill Production</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Tonnes</td>
<td>Gold Grade (g/t)</td>
<td>Gold Ounces</td>
</tr>
<tr>
<td>2004</td>
<td>1.3</td>
<td>2 974</td>
<td>5.3</td>
<td>503</td>
</tr>
<tr>
<td>2005</td>
<td>1.3</td>
<td>6 636</td>
<td>3.1</td>
<td>653</td>
</tr>
<tr>
<td>2006</td>
<td>1.3/1.0</td>
<td>4 438</td>
<td>2.6</td>
<td>375</td>
</tr>
<tr>
<td>2007</td>
<td>1.0</td>
<td>5 842</td>
<td>2.5</td>
<td>473</td>
</tr>
<tr>
<td>2008</td>
<td>1.0</td>
<td>5 154</td>
<td>4.2</td>
<td>701</td>
</tr>
<tr>
<td>2009</td>
<td>1.0</td>
<td>4 567</td>
<td>4.9</td>
<td>714</td>
</tr>
<tr>
<td>2010</td>
<td>0.85</td>
<td>6 225</td>
<td>3.9</td>
<td>785</td>
</tr>
<tr>
<td>2011</td>
<td>0.85</td>
<td>5 683</td>
<td>3.6</td>
<td>665</td>
</tr>
<tr>
<td>2012</td>
<td>0.85</td>
<td>547</td>
<td>2.0</td>
<td>35</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td>42 066</td>
<td>3.6</td>
<td>4 904</td>
</tr>
</tbody>
</table>

Stockpiles not included but milled

<table>
<thead>
<tr>
<th>Year</th>
<th>CoG (Au – g/t)</th>
<th>Reserve Block Models</th>
<th>Grade Control Models</th>
<th>Mill Production</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Tonnes</td>
<td>Gold Grade (g/t)</td>
<td>Gold Ounces</td>
</tr>
<tr>
<td>Stockpiles</td>
<td>6 266</td>
<td>1.5</td>
<td>295</td>
<td>6 266</td>
</tr>
<tr>
<td>Total Stockpiles</td>
<td>48 332</td>
<td>3.3</td>
<td>5 199</td>
<td>47 237</td>
</tr>
<tr>
<td>Block Models vs. Mill</td>
<td>-0.4%</td>
<td>-0.7%</td>
<td>-1.1%</td>
<td></td>
</tr>
</tbody>
</table>

CoG = Cut-off grade.

Low-grade stockpiles: at the end of 2003 - 5 369 000 tonnes at 1.6 g/t Au and at the end of 2009 - 897 000 tonnes at 1.0 g/t Au
Figure 25  Block Models vs. Grade Control Models

Gold Grade (Au - g/t)

Block Models vs. Grade Control Model

Tonnes ('000)

Block Models vs. Grade Control Model
On the basis of this comparison it can be concluded that the total ore to be mined according to the new LOM plan has a high degree of reliability, while the annual figures have a variance of perhaps ±5% to ±10%, but no bias. Studies presented in earlier reports (e.g., Redmond et al. 2011) have shown that the prediction variance for tonnages smaller than one represented by the production of one year is larger. This can only be overcome with a fairly robust in-fill drilling program that would allow smaller tonnages to be estimated with a higher degree of certainty. The responsible author is of the opinion that such a program is not warranted and would in fact be a waste of financial resources.

15.12 Life of Mine Plan

Based on the estimate of mineral reserves as of September 30, 2012 (Table 16) Dan Redmond and John Baker developed an updated (LOM) plan for the Central, Sarytor and Southwest Pits that is summarized in Table 20. Figures 19 and 20 show the resultant ultimate outlines of the Central pit and the Sarytor/Southwest pits together with the September 30, 2012 pit configurations and adjacent existing and future waste dumps. Figure 26 shows the major mining phases in the Central Pit for the duration of the LOM plan.
### Table 20 Kumtor Life-of-Mine Plan, Mine and Mill Production Forecast

Thousands of tonnes of ore and waste, cubic metres of ice and ounces of gold.

<table>
<thead>
<tr>
<th>Year</th>
<th>Ore Tonnes</th>
<th>Grade Au (g/t)</th>
<th>Contained Gold Ounces</th>
<th>Waste Rock Tonnes</th>
<th>Fill and Till Tonnes</th>
<th>Ice Tonnes</th>
<th>Total Waste Tonnes</th>
<th>Strip Ratio</th>
</tr>
</thead>
<tbody>
<tr>
<td>2012</td>
<td>5,230</td>
<td>3.1</td>
<td>525</td>
<td>10,419</td>
<td>5,896</td>
<td>15,874</td>
<td>32,190</td>
<td>6.2</td>
</tr>
<tr>
<td>2013</td>
<td>7,785</td>
<td>3.9</td>
<td>976</td>
<td>96,667</td>
<td>40,616</td>
<td>43,173</td>
<td>180,457</td>
<td>23.2</td>
</tr>
<tr>
<td>2014</td>
<td>3,338</td>
<td>5.2</td>
<td>559</td>
<td>133,685</td>
<td>40,051</td>
<td>19,126</td>
<td>192,861</td>
<td>57.8</td>
</tr>
<tr>
<td>2015</td>
<td>8,686</td>
<td>3.0</td>
<td>827</td>
<td>133,967</td>
<td>44,375</td>
<td>6,392</td>
<td>184,734</td>
<td>21.3</td>
</tr>
<tr>
<td>2016</td>
<td>8,725</td>
<td>3.3</td>
<td>928</td>
<td>145,411</td>
<td>23,290</td>
<td>5,501</td>
<td>174,202</td>
<td>16.7</td>
</tr>
<tr>
<td>2017</td>
<td>6,398</td>
<td>3.7</td>
<td>765</td>
<td>156,524</td>
<td>17,697</td>
<td>1,861</td>
<td>176,083</td>
<td>15.9</td>
</tr>
<tr>
<td>2018</td>
<td>8,192</td>
<td>2.4</td>
<td>624</td>
<td>118,971</td>
<td>7,0143</td>
<td>73,268</td>
<td>171,575</td>
<td>21.5</td>
</tr>
<tr>
<td>2019</td>
<td>10,824</td>
<td>2.9</td>
<td>993</td>
<td>68,798</td>
<td>101,430</td>
<td>3,989</td>
<td>142,066</td>
<td>3.7</td>
</tr>
<tr>
<td>2020</td>
<td>4,265</td>
<td>5.0</td>
<td>682</td>
<td>118,079</td>
<td>70,145</td>
<td>121</td>
<td>122,068</td>
<td>19.7</td>
</tr>
<tr>
<td>Totals</td>
<td>52,581</td>
<td>3.4</td>
<td></td>
<td></td>
<td></td>
<td>1,087,891</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

### Mining Central Pit

- **Ore Tonnes:** 5,230
- **Grade Au (g/t):** 3.1
- **Contained Gold Ounces:** 525
- **Waste Rock Tonnes:** 10,419
- **Fill and Till Tonnes:** 5,896
- **Ice Tonnes:** 15,874
- **Total Waste Tonnes:** 32,190
- **Strip Ratio:** 6.2

### Mining Sarytor and Southwest Pits

- **Ore Tonnes:** 2,374
- **Grade Au (g/t):** 3.5
- **Contained Gold Ounces:** 270
- **Waste Rock Tonnes:** 30,572
- **Fill and Till Tonnes:** 4,377
- **Ice Tonnes:** 800
- **Total Waste Tonnes:** 34,950
- **Strip Ratio:** 14.7

### Total Mining

- **Ore Tonnes:** 5,230
- **Grade Au (g/t):** 3.1
- **Contained Gold Ounces:** 525
- **Waste Rock Tonnes:** 10,419
- **Fill and Till Tonnes:** 5,896
- **Ice Tonnes:** 15,874
- **Total Waste Tonnes:** 32,190
- **Strip Ratio:** 14.7

### Stockpile Closing

- **Ore Tonnes:** 5,222
- **Grade Au (g/t):** 2.1
- **Contained Gold Ounces:** 261
- **Waste Rock Tonnes:** 3,828
- **Fill and Till Tonnes:** 2,169
- **Ice Tonnes:** 184,734
- **Total Waste Tonnes:** 34,950
- **Strip Ratio:** 23.2

### Milling

- **Ore Processed Tonnes:** 1,635
- **Feed Grade Au (g/t):** 5.3
- **Feed Contained Gold Ounces:** 277
- **Plant Recovery %:** 82.3%
- **Recovered Gold Au Ounces:** 228

October 1 to December 31, 2012
15.13 Additional Mineral Resources

15.13.1 Additional Mineral Resources Mineable by Open Pit

The mineral reserve estimation process described in Section 15 estimates those portions of the Central, Sarytor and Southwest Deposit resources that have been converted to mineral reserves mineable by open-pit methods, and these have been summarized in Table 16.

Additional mineral resources exist outside of the ultimate pit designs in the three deposits for which mining is planned, and in the Northeast Deposit, as set forth in Table 21. The additional mineral resources are reported at a cut-off grade of 1.0 g/t gold for the Sarytor, Southwest and Northeast deposits and at 0.85 g/t gold for the additional mineral resources potentially mineable by open pit in the Central Deposit.

<table>
<thead>
<tr>
<th>Category</th>
<th>Cut-Off Grade (Au – g/t)</th>
<th>Tonnes (‘000’)</th>
<th>Gold Grade (g/t)</th>
<th>Contained Gold (‘000 Ounces)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Measured</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Central Deposit</td>
<td>0.85</td>
<td>21 709</td>
<td>2.3</td>
<td>1 600</td>
</tr>
<tr>
<td>Southwest Deposit</td>
<td>1.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Sarytor Deposit</td>
<td>1.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td><strong>Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Central Deposit</td>
<td>0.85</td>
<td>7 599</td>
<td>2.5</td>
<td>604</td>
</tr>
<tr>
<td>Southwest Deposit</td>
<td>1.0</td>
<td>3 329</td>
<td>2.0</td>
<td>217</td>
</tr>
<tr>
<td>Sarytor Deposit</td>
<td>1.0</td>
<td>1 498</td>
<td>2.3</td>
<td>108</td>
</tr>
<tr>
<td><strong>Total Measured &amp; Indicated</strong></td>
<td></td>
<td><strong>34 135</strong></td>
<td><strong>2.3</strong></td>
<td><strong>2 529</strong></td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Central Deposit</td>
<td>0.85</td>
<td>1 917</td>
<td>2.7</td>
<td>165</td>
</tr>
<tr>
<td>Southwest Deposit</td>
<td>1.0</td>
<td>2 568</td>
<td>2.6</td>
<td>214</td>
</tr>
<tr>
<td>Sarytor Deposit</td>
<td>1.0</td>
<td>1 144</td>
<td>2.4</td>
<td>89</td>
</tr>
<tr>
<td>Northeast Deposit</td>
<td>1.0</td>
<td>4 068</td>
<td>2.1</td>
<td>278</td>
</tr>
<tr>
<td><strong>Total Inferred</strong></td>
<td></td>
<td><strong>9 697</strong></td>
<td><strong>2.4</strong></td>
<td><strong>746</strong></td>
</tr>
</tbody>
</table>

Mineral resources that are not mineral reserves have no demonstrated economic viability. Additionally, inferred mineral resources have a large degree of uncertainty as to their existence and it cannot be assumed that all or any part of the inferred mineral resources can be upgraded to a higher mineral resource category.
Excluded from the data in Table 21 are additional mineral resources potentially mineable by underground methods and calculated at a higher cut-off grade as explained in Section 15.13.1. These additional mineral resources amount to 289,000 tonnes at an average gold grade of 10.8 g/t in the indicated class and 1,380,000 tonnes at an average gold grade of 10.4 g/t in the inferred class. However, these additional resources are included in Table 22 below.

The additional mineral resources potentially mineable by open pit occur in the space between the current ultimate pit designs that are based on a gold price of $1,350 per ounce, and larger pit shells (mineral resource shell) that are considered to be uneconomic at this time. The conversion of the additional measured and indicated mineral resources into mineral reserves at the three open pits is a function of economics. These are governed principally by the increased incremental stripping ratios related to this material, and increased capital costs. The author believes that the assumptions used to define the larger pit shells satisfy the CIM guidelines that require mineral resources to “offer reasonable prospects of economic extraction”. There are no additional mineral resources in the southwestern part of the KS-13 final pit since the surrounding mountains preclude any further deepening of the pit in this area. The additional mineral resources potentially mineable by open pit are located in the central and northeastern parts of the Central Deposit starting at the Saddle Zone. The trace of the mineral resource shell is shown in Figure 30.

15.13.2 Additional Mineral Resources Mineable by Underground Methods

Similar to Central Pit block models created since 2006, the KS-13 model includes an estimation of SB and Stockwork mineral resources considered potentially amenable to underground mining. These additional mineral resources are located within the footprint of the KS-13 final pit and extend below the bottom of the KS-13 design pit to a maximum vertical distance of 200 metres below the KS-13 Ultimate pit. The underground mineral resource estimates do not include a provision for dilution or for a crown pillar at the base of the open pit and are summarized in Table 22.

Table 22 Additional Mineral Resources Potentially Mineable by Underground Methods
As of September 30, 2012

<table>
<thead>
<tr>
<th>Category</th>
<th>Cut-Off Grade (Au – g/t)</th>
<th>Tonnes (‘000’)</th>
<th>Gold Grade (g/t)</th>
<th>Contained Gold (‘000 Ounces)</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Indicated</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>SB Zone</td>
<td>6.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Stockwork Zone</td>
<td>6.0</td>
<td>351</td>
<td>10.7</td>
<td>121</td>
</tr>
<tr>
<td><strong>Total Measured &amp; Indicated</strong></td>
<td></td>
<td>351</td>
<td>10.7</td>
<td>121</td>
</tr>
<tr>
<td><strong>Inferred</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>SB Zone</td>
<td>6.0</td>
<td>3,193</td>
<td>11.2</td>
<td>1,150</td>
</tr>
<tr>
<td>Stockwork Zone</td>
<td>6.0</td>
<td>2,002</td>
<td>11.0</td>
<td>705</td>
</tr>
<tr>
<td><strong>Total Inferred</strong></td>
<td></td>
<td>5,195</td>
<td>11.1</td>
<td>1,855</td>
</tr>
</tbody>
</table>
Mineral resources that are not mineral reserves have no demonstrated economic viability. Additionally, inferred mineral resources have a large degree of uncertainty as to their existence and as to whether all or any part of the inferred mineral resources can be upgraded to a higher mineral resource category.

The mineralization constituting the additional mineral resources amenable to underground mining in both the SB and Stockwork Zone shows reasonable to good grade continuity at the elevated cut-off grade and with the current drill-hole spacing. Following recent programs of in-fill drilling, a small portion of these additional mineral resources have for the first time been placed in the indicated category, as described in Section 15.3.2 This classification recognizes that the higher cut-off grade applied to the underground mineral resources compared to the pit mineral resources requires a tighter drill pattern than is in place for much of this mineralization. The classification as inferred mineral resources of most of the additional resources which assumes an underground method reflects this lack of detailed drilling.

With the historical experience of underground development during the Soviet era, geotechnical knowledge gained over 15 years of open-pit mining, extensive geotechnical drilling and most recently four years of underground development, Centerra has chosen to report the underground mineral resources at what would normally be considered a high cut-off grade of 6.0 g/t to reflect the high expected mining costs for a mining method that deals with the poor ground conditions, and to honour the natural continuity character of the high-grade mineralization of the Stockwork and SB Zones.
16 MINING METHODS

The Central Pit is a large excavation elongated in a south-west to north-east direction, and both the present and the final pit design have a shape resembling an hourglass, with the two wider areas reflecting the locations of the high-grade Stockwork and SB Zones, respectively, as described in Section 7.4. The elongated northern part of the Central Pit, referred to as the northeast or high wall, has a vertical height of between 360 and 630 metres, a maximum width of 1 200 metres and a length of 1 300 metres (at the pit rim). The high wall governs mining access to the ore of the Stockwork Zone.

At the neck of the hourglass, the current pit is some 1 000 metres wide. In the southeast, the SB-Zone part of the pit has reached a vertical height that ranges from 320 to 440 metres. At the pit rim, the opening above the SB Zone currently measures 1 500 by 1 500 metres.

16.1 Mining Operations

Mining operations at Kumtor use conventional open-pit mining methods. The Central Deposit is mined in a large open pit where total material excavated in the first nine months of 2012 was 109 million tonnes or 400 000 tonnes per day. During the past several years, ore stockpiles have from time to time provided mill feed when the open pits were unable to supply the daily mill feed of approximately 15 500 tonnes per day due to the high overall ore to waste stripping ratio and due to the geotechnical issues described in Section 15.3. As part of the latest such issue in 2012, all existing stockpiles had been depleted in July of 2012, and the mill subsequently closed from July 24 to September 18, 2012. Stockpile accumulation since then has been minimal.

Mining in the Central Pit has recently converted from eight-metre to ten-metre benches to allow more efficient use of the larger mining equipment purchased in recent years. Ore at the smaller Sarytor and Southwest pits will be mined on nominal four-metre benches for better mining selectivity of the smaller ore zones.

Blast holes are drilled using 6 diesel-powered Sandvik DR-460 rig and two Drilltech D45KSH rotary-percussion drill rigs, with a hole diameter of 300 millimetres (mm). Charging the holes is undertaken by special bulk explosives trucks delivering either ammonium nitrate with fuel oil (ANFO), or emulsion explosives for wet holes. The explosives consumption is about 0.26 kg per tonne of ore or waste.

The main loading fleet operating at the end of September, 2012 consisted of two Hitachi 3600 shovels that were commissioned in Augusts/September, 2012, nine Liebherr 9350 hydraulic shovels that have been commissioned between 2006 and 2010 as part of the original SB zone expansion, five CAT 5130 B hydraulic shovels which were part of the original mining fleet, and three CAT 992C front-end loaders. Typically, the shovels are used for production and the loaders for ore blending, cleanup and support during shovel maintenance.
The haulage fleet operating at the end of September, 2012 was 59 CAT 789 haul trucks that came into service in 2010 to 2012, 32 CAT 785 haul trucks commissioned between 2006 and 2008 as part of the original SB Zone expansion, and 5 CAT 777 haul trucks, many of which were part of the original mining fleet. In 2009-2010 KOC updated its fleet production tracking system to Modular Mining to further improve fleet utilization and overall mining efficiency.

With the continued expansion of the Central pit, additional mining equipment, including 1 additional Hitachi 3600 hydraulic shovel and 27 CAT 789 haul trucks will be purchased between 2012 and 2014 to accommodate the mine production schedule of the new LOM plan. The scheduled major equipment increases and later retirements of the mining fleet over the new mine life is summarized in Table 23.

The Central Pit has had the benefit of a favourable topographical situation. The top mining elevation in the current ultimate pit design is at 4,474 metres, and the very deepest part of the final pit excavation will be at 3,500 metres in the southwest part of the deposit. The crushing plant to which ore is delivered is at about 4,050 metres and ore transport was thus downhill for the upper portion of the ore body, and will have a maximum uphill vertical haul of 550 metres for the lower portion of the ore body. The haulage distance from the Sarytor and Southwest Deposits, scheduled to be mined starting in 2018, will be approximately 4.5 kilometres using the relocated haul road after the mining out of a section of the Davidov glacier as part of the KS13 Central Pit design.

The initial stripping of the Central Pit in 1995 had the unusual challenge of mining a portion of the Lysii glacier that covered the northeastern area of the planned open pit, and lesser quantities of ice have been removed in subsequent years as the northeast high wall of the open pit was pushed back. Additional mining of the Lysii glacier is planned as part of the next high wall push-back starting in 2014. Mining of ice and superimposed waste dumps has continued since 2007, as more fully described in Section 15.3.2.

The waste does not have acid generation potential because of its high carbonate content. The strategy for the disposal of the significant amounts of waste is described in Section 18.2.
### Table 23 Additions and Retirement of Major Mining Equipment, July 2012 to 2023

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Hitachi 3600 Shovels</td>
<td>2</td>
<td>+2</td>
<td>+1</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-2</td>
</tr>
<tr>
<td>Liebherr 9350 Shovels</td>
<td>9</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-2</td>
<td>-2</td>
<td>-2</td>
<td>-4</td>
</tr>
<tr>
<td>CAT 5130 Shovels</td>
<td>5</td>
<td>-2</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-3</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CAT 789 Trucks</td>
<td>59</td>
<td>+2</td>
<td>+10</td>
<td>+15</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-12</td>
<td>-54</td>
<td></td>
</tr>
<tr>
<td>CAT 777 B Trucks</td>
<td>5</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-5</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>CAT 785 Trucks</td>
<td>32</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>-25</td>
</tr>
</tbody>
</table>
Hydrological conditions in the open pits are controlled by the presence of originally up to 250 metres of permafrost that has become more discontinuous in the areas exposed by mining and the seepage of seasonal surface waters and ground waters into the open pit and its walls.

Groundwater volumes are relatively minor while the inflow of seasonal melt waters can be as high as 2,000 L/sec. The Davidov Glacier is the dominant source of melt water entering the Central Pit in the summer months.

Where possible, surface waters are diverted away from the pit using diversion ditches, sumps and gravity pipelines, while water within the pit is channelled to sumps along dewatering ditches and is then pumped outside of the pit limits using electrical pumps purchased in 2011 and 2012. Recent experience indicates that up to two cubic metres of water per second can enter the bottom of the open pit in the summer. KOC maintains a significant inventory of spare pumps, parts and piping in the event of a pump failure or required expansion of the system during peak flow periods.

16.2 Grade Control Procedures

Ahead of the actual mining activities, bench composites of existing diamond drill core are tested in the mill laboratory for their metallurgical character, and refractory and carbonaceous ore types are delineated on this basis. This data are also included in the block model used for mineral resource and reserve estimation and determines in part the value of a block. In general, the northern part of the Central Deposit has the poorest recoveries, but higher grades are matched by higher recoveries. The Southwest and Sarytor Deposits show recoveries that are lower than those experienced in the Central Deposit. The metallurgical information is included in the data used for pit optimization (Table 15).

Grade control in the pit is based on the sampling of blasthole cuttings whose grade and metallurgical character are determined at the mill laboratories. This information is entered into the grade control module of the “GEMS” mining software. Based on the GEMS output, the various ore blocks are staked in the field for digging, taking into account experience with ground movement during blasting. The ore is then delivered to the crusher or the appropriate stockpile depending on the daily blending requirements. KOC has an active and dynamic blending program in close contact with the mill that adjusts the ore blend as required to maximize the gold recovery. The grade control personnel work seven days per week.

The blasthole assay information, combined into the ore control model, is also used to estimate the monthly pit production and for short and medium-term planning, as monthly forecasts of tonnes and grade by the mineral resource block model have a variance that is too high for short-term planning. In addition, logging of the blasthole chips allows the intensity of the alteration to be mapped, an important input parameter into the definition of the structural ore zones that in turn play an important role in the mineral resource estimation process.

In October 2009, a review of grade control procedures at the operation was completed by AMEC Americas Limited in an effort to further improve procedures (Brettschneider, 2009).
Several recommendations were outlined in the study and many were implemented into standard operational procedures during 2010. A review of monthly production reports since 2010 indicate a good correlation between ore identified in the in-pit grade control with the ore received and sampled by the mill.

### 16.3 Mining Equipment Maintenance and Services

The maintenance department is currently responsible for over 162 major pieces of mine equipment, the process plant, the effluent treatment plant and the electrical distribution system. The department is also responsible for approximately 370 pieces of other transportation and mechanical equipment such as the fleet for hauling supplies to and from the mine site from the marshalling yard in Balykchy.

KOC has utilized a computerized maintenance system since start-up for mobile and plant maintenance requirements. Initially schedules were set in accordance with the manufacturers’ specifications but as the component history developed, the preventative maintenance schedules were adjusted where required. Work orders are used to control and track all maintenance employee and materials costs.

For the last several years, mechanical availability for the process plant was approximately 95%, and the mining fleet mechanical availability for the first nine months of 2012 is outlined in Table 24. The two new Hitachi 3600 shovels outlined in Table 23 have not been included in Table 24 as they had just completed commissioning during the month of September 2012.

<table>
<thead>
<tr>
<th>Major Mining Equipment Type</th>
<th>Mechanical Availability (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Liebherr 9350 Shovels</td>
<td>87.8%</td>
</tr>
<tr>
<td>CAT 5130 Excavators</td>
<td>83.5%</td>
</tr>
<tr>
<td>CAT 789 Trucks</td>
<td>93.7%</td>
</tr>
<tr>
<td>CAT 777 B Trucks</td>
<td>87.3%</td>
</tr>
<tr>
<td>CAT 785 Trucks</td>
<td>90.9%</td>
</tr>
<tr>
<td>Production Drills</td>
<td>83.8%</td>
</tr>
</tbody>
</table>

A comprehensive training program that focuses on the transfer of mechanical, electrical, diagnostics and planning skills from the expatriates to the national workforce continues to be successful. Additional planners were added in 2008 and 2009 to continue the process of improving proactive maintenance.
Power is provided from the Kyrgyz national grid under the Priority Power Supply Agreement. Power generation in the Kyrgyz Republic is from hydro and thermal plants. A new power line from Barskaun was constructed in 1995 to serve the Kumtor Project.
17 RECOVERY METHODS

17.1 Process Description

Extensive metallurgical testing was completed by Kyrgyz Geology from 1981 until 1989. During the Kilborn Feasibility Study, Kilborn completed additional test work. The current plant flowsheet (Figure 27) reflects the fine-grained nature of the gold and its intimate association with pyrite, and consists of crushing, grinding, pyrite flotation, and two-stage re-grinding of the flotation concentrate. Two separate carbon-in-leach (CIL) circuits extract the gold from the reground concentrate and from the flotation tails, with final gold recovery accomplished by electro-winning and refining. The mill was originally designed with a capacity to process 4.8 million tonnes of ore per year. The mill throughput is currently 5.9 million tonnes per year or a nominal capacity of 15 900 tonnes per day.

The ore to be milled is managed through a number of stockpiles that receive ore of different metallurgical character and of different grade ranges as determined by grade-control data and thus allow blending of the mill feed for optimum gold recovery. A gyratory crusher reduces run-of-mine to minus 200 millimetres. The ore is then fed to a coarse ore stockpile from which it is reclaimed for grinding. The stockpile has a nominal capacity of 100 000 tonnes and a live capacity of approximately 15 000 tonnes. Primary grinding is completed in a 9.14 m diameter by 4.27 m long 6 340 kW semi-autogenous (SAG) mill operating in closed circuit with a pebble crusher ball mill feed sizing screen. Secondary grinding is completed in a 5.49 m diameter by 7.92 m long 4 100 kW ball mill in closed circuit with 660 mm diameter hydrocyclones.

Hydrocyclone overflow at 80% passing 140 microns gravitates to the flotation circuit, comprised of two parallel banks of nine 42 m³ naturally aspirated flotation cells. A bulk sulphide flotation concentrate representing 7 to 11% of the original mill feed is produced with a grade of 30 to 50 g/t gold, about ten times the mill head grade, and a gold recovery of 87 to 92% into the concentrate. The flotation concentrate is thickened in the 15.2 m diameter concentrate wash thickener.

Ultra-fine grinding of flotation concentrate is completed in two stages. Flotation concentrate from the wash thickener underflow is first re-ground to 90% passing 20 microns in a 5.49 m diameter by 7.92 m long 4 100 kW ball mill in closed circuit with 150 mm diameter hydrocyclones. After thickening in the 15.2 m diameter CIL feed thickener to 50% solids, it is further ground to 95% to 98% passing 20 microns in a 2 600 kW IsaMill that was commissioned in October 2005. The IsaMmill provides additional liberation of the fine gold (2-5 microns) enclosed in pyrite.

The concentrate is diluted to 45% solids, pre-aerated for 40 hours and leached for 80 hours in the CIL circuit consisting of six 14.6 m diameter by 14.6 m tall agitated tanks in series. Cyanide solution, slaked quicklime and activated carbon to maintain a concentration of 14 grams per litre (g/L) carbon are added to the CIL circuit.
The flotation tailings are thickened to 50% solids in the 25.9 m diameter flotation tailings thickener and leached in the tailings CIL circuit, which consists of three 14.6 m diameter by 14.6 metre tall agitated tanks in series. Cyanide additions are lower in the tailings CIL circuit compared to the concentrate CIL concentrate and carbon concentration is 8 g/L. Overflow from all four thickeners is recycled through the process.

The carbon in both CIL circuits is moved counter-current to the slurry flow, and the loaded carbon from the first flotation tailings CIL tank is pumped to the third concentrate CIL tank to continue loading. Loaded carbon from the first concentrate CIL tank is pumped to the gold recovery plant. Total carbon inventory is approximately 300 tonnes.

The loaded carbon is stripped in two parallel 7-tonne carbon stripping circuits. Gold is subsequently recovered by electro-winning. Gold flake is washed from the cathodes, dried and smelted in an induction furnace and cast into doré bars. Carbon is reactivated in a 2 400 kW electrically heated horizontal kiln for reuse in the CIL circuits.

Tailings from both CIL circuits are combined in the 30.5 metre diameter tailings thickener and discharged by gravity to the tailings disposal area through a slurry pipeline (Figure 3). The tailings pipeline includes four choke stations for velocity control. The tailings line is twinned and is placed in a lined trench for containment in the instance of a leak. The tailings management facility is described Section 18.3 of this report.

Process control is provided by a Foxboro distributed control system, which allows the monitoring and control of the entire process. Six automatic samplers recover samples from all circuits. An automatic reagent addition system optimizes the performance of the flotation circuit. A particle-size monitor for the re-ground concentrate adjusts the grinding process in real time and thus reduces gold losses related to poor grinding. An automatic analyzer in the CIL circuit helps to maintain the optimum levels of sodium cyanide and the pH.

### 17.2 Gold Recovery

Gold recovery in the CIL circuits is 30% for the flotation tailings and 90% for the sulphide concentrate. Overall, 90% of the recoverable gold is from the concentrate CIL circuit, with the remainder from the tailings CIL circuit.

Gold recovery during the early phase of the operation was affected by the preg-robbing character of some of the ore due to active graphite. This negative effect has been moderated by adding diesel fuel as a masking agent to the SAG and the re-grind mills. Historically, the overall metallurgical recovery of gold in the Kumtor processing plant has averaged 79.4%. Based on the experience to date, future annual recoveries can be expected to range from 68% to 86%, depending on the head grade, ore source and taking into account the expected somewhat inferior performance of the Sarytor mineralization.
17.3 Mill Expansion

With the development of the new LOM plan, there is an opportunity to increase mill throughput due to the large projected ore stockpiles (Table 20). An earlier (2006) internal study was completed on expanding the mill to 6.6 Mt/a (18 000 t/d nominally). The preliminary capital cost identified at that time was approximately $ 24 million. This study was used as the starting point to evaluate an increase mill throughput for KS-13.

The throughput selected for the 2006 study was based on operating experience as to what could be achieved with the existing grinding mills. In 2011, SRK completed a review of the Kumtor grinding circuits and concluded that mill throughput could be increased without significant equipment changes (SRK, 2011). Ore hardness data was lacking to quantify how much the throughput could be increased. Centerra completed JKSimMet simulations of the Kumtor grinding circuit with typical ore hardness similar to Kumtor showed that throughputs from 6.6 Mt/a to 7.3 Mt/a were possible.

Ore hardness testing is required to provide a basis to accurately model the expanded throughput. This is expected to be completed by the end of 2012. For mine scheduling purposes, the base case for expanded mill throughput was selected at 6.7 Mt/a.

Flowsheet changes required to process 6.7 Mt/a were identified. The following list summarizes the major changes required.

- Upgrade SAG mill feed conveyor to 900 tonnes per hour.
- Convert the SAG mill drive to variable speed operation.
- Install larger SAG mill discharge pumps.
- Repurpose the current regrind ball mill as an additional secondary ball mill. Install ball mill feed splitter, hydrocyclones and hydrocyclone feed pumps.
- Install additional flotation capacity upstream of existing flotation cells.
- Install two vertically stirred ball mills and associated pumps and ancillary equipment for regrinding flotation concentrate using the existing regrind cyclones.
- Install a second IsaMill.
- Upgrade all thickeners for increased throughput.
- Install an expert system (as part of the Foxboro control system) for control of the grinding circuits.

The scope of work from the earlier 2006 study was updated. Costs were also updated using the US Project MCS Index from Costmine. The estimated capital cost is $ 33.5 million, including:
• Direct costs of $22.0 million.
• Indirect costs of $5.9 million.
• Contingency of $5.7 million, based on 20% of direct plus indirect costs.
• Currently sufficient electrical power and water are available to the expansion.

This estimate is of a scoping nature with an accuracy of ±30%.

Engineering studies are required to confirm the scope of work required for the expanded throughput. Using the above scope of work as the starting point, a detailed study of all changes required to increase the mill throughput is required. The study should identify the scope, downtime required, long lead items, capital cost. This study should commence in early 2013. Construction is estimated to require 18 to 24 months.

Commissioning and ramp up is estimated at 3 to 6 months, which roughly coincides with the LOM requirements for expanded throughput in 2016.

The authors understand that Centerra has agreed in principle to the mill expansion plan, and the associated capital costs are included in the project capital costs presented in Section 21.2.
18 PROJECT INFRASTRUCTURE

18.1 Infrastructure and Logistics

As part of the KS-13 pit expansion and the continued growth and movement of the waste dump located within the Davidov valley, Kumtor will be need to relocate and/or reconstruct several significant surface infrastructure facilities between 2013 and 2015 including:

- The main administration building;
- The lower heavy equipment and light vehicle shop;
- The main site electrical sub-station;
- The Mack truck shop;
- The upper shovel maintenance shop;
- The upper fuel farm;
- The upper emulsion plant; and
- The mobile concrete batch plant.

In 2012 Centerra retained a Western EPCM construction group to provide preliminary cost estimates and project scheduling and a budget of $56 million has been included in the LOM capital budget to complete this work from 2013 to 2015.

18.2 Waste Dumps Design and Capacity

To the end of year 2007, the majority of the Central Pit waste had been deposited on the Davidov glacier, as explained in Section 15.3.2, with a smaller dump on the Lysii valley slope also being used, which is located to the northeast of the processing facility (the Lysii Slope Dump). The expanding pit (due to the incorporation of the SB Zone) resulted in a curtailment of this practice, with waste being deposited in the Davidov valley immediately below the pit rim starting in 2009. Since then, almost all the waste from the Central Pit has been placed into the Davidov valley above the administration building (Figure 28). Waste from the Southwest pit has been placed in the Sarytor valley.

With open-pit mining now scheduled to continue until 2023, a significant volume of additional waste rock material will be generated (1.8 billion tonnes from the Central Pit and 0.2 million tonnes from the Sarytor and Southwest pits, Table 20). This material will be stored in roughly equal amounts in three depositories - the Davidov valley; the Sarytor valley downstream of the Southwest and Sarytor open pits which will receive all of the waste from both of these satellite pits as well as part of the waste from the Central Pit; and on the Lysii Slope Dump. The planned ultimate limits of all three waste dumps are outlined on Figure 28.

Valley dumps will be constructed using a rock drain as recommended by BGC (BGC Engineering 2010a) to ensure that melt water from above the dump can pass through the bottom of the dump with minimal impediment; otherwise, water will need to be pumped around the waste dumps in a controllable fashion. For the Davidov valley dump, this engineering principle comes late in its history, with most of its length having been created without a rock drain.
However, the water flow through the dump appears to be relatively unimpeded, and a relatively high hydraulic permeability of $10^{-4}$ metres per second was measured in one test hole.

A trench located on the NE side above the Davidov Valley dump currently diverts water around the dump with the objective to capture as much water as possible and divert it without allowing it to go through the dump. To reduce the inflow of glacial melt water into the Davidov valley and the waste dump, an upgraded water diversion system will be constructed adjacent to the southern corner of the Central Pit to intercept glacier melt water and divert it into the existing trench around the Davidov valley dump.

The Davidov valley dump is expected to contain 31% of the future waste. It will ultimately be 230 metres high, two kilometres long, and up to 1500 metres wide at its planned crest. There was significant dumping of waste material into the Davidov valley during 2011 and 2012, which used both bottom-up and top-down strategies in an effort to ensure that there is a substantial buttress at the toe of the dump that provides support for the material above.

The slopes in the area are underlain by permafrost soils which exhibit periglacial landforms called solifluction lobes that are indicative of natural slope instabilities on top of the permafrost. Such conditions provide poor foundations for side-hill dumps, and the Lysii Slope Dump has already failed, as is evident from Figure 14. The much larger final accumulation of waste on the Lysii Slope Dump will likewise fail and in the process occupy a larger footprint than originally laid out.

All of the waste dumps will be constructed in a series of lifts. Individual lifts will generally be 90 metres constructed at the angle of repose of approximately 36°. The overall slope angle of the dumps including berms will vary from 20° to 28° depending on the waste dump or dump sector.

Experience shows that none of the waste dumps will truly be stable. In 2008, the Sarytor dump failed and flowed 150 metres across the Sarytor valley floor, blocking the water flow of the Chon-Sarytor River. The waste dumps slid down-hill until its overall slope angle flattened and stabilized. The dump failure is believed to be attributed to the relatively high (up to 120 metres thick) waste dumps stacked on sloping, ice-rich permafrost soils.

In June 2012, the Davidov valley dump started to exhibit signs of deep-seated deformation and is currently moving down-valley at a velocity of 50 mm/hr. Surface cracking has developed with extensive deformations of the waste dump as well as the moraine surface in its front. An extensive network of inclinometers was installed that indicated consistent shearing at depth starting from as deep as 45 metres below the original surface. Inclinometers that were located farther away from the waste dump toe showed smaller movement. It appears that the substratum itself is shearing and moving, taking the waste and remaining ice from the displaced Davidov Glacier with it. A complete explanation of this mechanism is outstanding since the nature of the thick substratum is essentially unknown, because of difficulties experienced with drilling and recovery of undisturbed soil samples for testing. The current plans with respect to the two valley dumps are to let them fail, monitoring to make the deposition of the waste safe. For the Davidov valley, this means that the existing infrastructure (particularly the administration
building) below the dump needs to be moved out of the way, and this has been described in the previous section.

18.3 Tailings Management Facility

18.3.1 General

The tailings management facility is located in the Kumtor River valley (Figure 3) and consists of twin tailings pipelines (each approximately six kilometres in length), a tailings dam, an effluent treatment plant and two diversion ditches around the area to prevent runoff from natural watercourses from entering the tailings basin. These facilities received original approval in 1999, with a limit of 3,670.5 metres being set to which the dam could ultimately rise. Additional lifts are regularly added to the dam, and KOC is required to apply and obtain permits for the Government from time to time to address the interim raising and construction activities, as described in Section 20.1.

Tailings are deposited from the dam using conventional spigoting methods to push the transport tailings water pond to the back of the impoundment against natural ground. Beaches of 300 to 600 metres are maintained between the dam crest and the pond surface. During summer operations (May through October), some five million cubic metres of effluent from the tailings pond are treated and subsequently discharged into the environment, thereby lowering the water level in the pond. By October, the pumps are turned off for the winter and the water levels slowly rise again.

The tailings dam was designed and constructed to address the permafrost conditions at the mine site and to standards for seismic activity in the region. The tailings dam consists of a compacted fill dam approximately three kilometres long. The dam crest is ten metres wide and the side slopes are approximately 3 horizontal to 1 vertical (3H:1V). The dam is currently 36 metres high at its central part. The dam fill consists of alluvial sands and gravels borrowed from a pit located approximately five kilometres from the dam. A geomembrane liner has been placed on the upstream face and extends one hundred metres upstream of the dam toe on natural ground into the impoundment.

Construction activities to raise the dam crest have taken place almost every year since 1997. As of September 30, 2012, the dam had reached its current elevation of 3,664 metres, with the 2012 annual construction activities continuing. As at September 30, 2012, the tailings facility contained approximately 58 million cubic metres of tailings representing 85 million tonnes of ore processed. The currently approved ultimate dam and stabilizing toe berm have been designed to store up to 88 million cubic metres of tailings; or some 32 million cubic metres above current stored volume with the ultimate crest elevation of 3,670.5 metres.
The dams and appurtenances are regularly inspected by KOC personnel during routine work at the facility and visually inspected on an annual basis by Golder, with the most recent having been carried out in October 2011 (Golder, 2012a). Golder reported the dams and appurtenances to be in good condition and functioning as required.

### 18.3.2 Dam Deformation

Since its construction, the dam foundation has experienced horizontal deformations, with the Kyrgyz regulatory authority (Kyrgyz Republic Institute of Rock Mechanics - KRIM) initially raising concerns in 1999. The tailings dam is founded on permafrost which includes zones of frozen silt containing excess ground ice. The deformations have been caused by creep deformation of the ice-rich silt induced by the load from the dam embankment.

Monitoring of the deformations indicated that the rate of creep was constant, but the deformations have been of concern to the regulatory authorities. The horizontal deformations, which were in the order of 100 to 200 mm until 2006, were well within the limits of deformations previously recorded on several large water and tailings dam structures elsewhere as reported in the literature (Golder Associates Ltd, 2006).

To satisfy the regulatory concerns, a shear key and toe berm were constructed in 2003 to reduce the rate of movement. However, the deformations continued undiminished at the earlier rate, and an additional engineering assessment was undertaken by BGC in 2005 (BGC Engineering Inc 2005). The additional assessment indicated that the initial shear key did not penetrate the soils sufficiently deep to completely inhibit the creep deformations of the ice-rich foundation silt. KOC commissioned additional design for a shear key, and since 2006, sections of the initial shear key have been deepened and expanded and new shear keys have been added along other portions of the dam that previously did not have any. Design changes developed in close cooperation with KRIM and implemented since 2007 have assumed that the tailings dam will be raised to its ultimate permitted elevation of 3 670.5 metres. The new shear key has been excavated to an average depth of nine metres, and silt and clay including ice lenses have been removed to expose the underlying dense granular fill that contains little ground ice (Figure 29). Test pits one to two metres deep were excavated to confirm that sound foundations had been reached. The shear key was designed to withstand a seismic event of Richter magnitude 7.3 with an acceleration of 0.4 g (BGC Engineering, 2012b).

KOC has provided BGC with instrumentation monitoring data to develop a numerical model that predicted the observed deformations and provided guidance towards the stability of the dam if constructed to its ultimate dam crest elevation. The numerical model was initially developed based on observations up to 2006 (BGC, Engineering Inc 2007) and showed that the proposed shear key and buttress would effectively slow down deformations to about 3 mm per year by 2025, even with downstream raising of the dam to the ultimate crest elevation of 3 670.5 metres. In 2010, the numerical model was re-evaluated based on deformation data collected up to 2010 (BGC Engineering Inc, 2010). The observations indicated that the dam was deforming at a rate slightly higher than initially estimated by the 2007 model. However, the analysis confirmed
earlier predictions that the proposed shear key and buttress would effectively slow down deformations given the same dam-raising schedule.

KOC has been regularly providing Golder and KRIM with information regarding tailings dam construction activities, the status of items of specific interest for geotechnical monitoring, and updated instrumentation monitoring data. In early 2012, Golder Associates (2012a) reported on their site inspection and stated that the structures were in good condition and were functioning as required at the time of the site visit.

Data on deformation, groundwater levels within the dam and temperature readings have been regularly evaluated by KRIM. The most recent reports (Chukin 2012a and 2012b) state that the inclinometers installed throughout the dam indicate the consistent reduction of displacement rates, which are reduced to almost zero values at the front of the shear key.
18.3.3 Petrov Lake Outburst Flood Potential

Petrov Lake is contained by a natural moraine dam which has been assessed as to its likelihood of bursting by BGC Engineering (2012a). Petrov Lake is increasing in size and volume due to the melt and recession of Petrov Glacier which feeds the lake. Continuing global warming will lead to melting of the ice/permafrost within the dam and a breaching of the dam due to rapid erosion of a channel following an initial over-topping event. Alternatively, taliks (unfrozen ground below small lakes on the morainal dam) may coalesce through continued melting and cause a large-scale failure of the dam. Any of these events would result in “extreme” scouring and sediment transport and aggradation throughout the affected area along the Kumtor River which drains Petrov Lake (Figure 31). Such an event could result in erosion of the tailings dam shear key toe next to the Kumtor River. “Erosion of the shear key significantly reduces the factor of safety” (of the tailings dam) (BGC Engineering 2012a, page B-5). Other impacts include the partial or complete destruction of three bridges (one of which carries the tailings line), the gravel crushing plant, and some of the power line pylons, with the risk of multiple fatalities depending on the severity and the timing of the event.

In their report, BGC Engineering (2012a) concludes that a catastrophic event is unlikely to occur during the mine life, but is very likely to occur sometime after closure. This is mainly based on observations from four drill holes and the lack of seepage through the moraine dam that “...imply a relatively high level of stability of the Petrov Lake moraine dam that can be attributed to its largely frozen state.” (ibid, page 11).

Proposed preventive measures include the lowering of the Petrov Lake level by siphoning, by deepening of the existing outflow channel, or by creating an additional outflow channel. “Lowering the lake level would have two principal benefits: First it would increase the freeboard of the moraine dam to an extent that dam subsidence by permafrost degradation and ground ice thaw would be offset, and second it would decrease the total volume of water available to discharge into the floodplain.” (ibid, page 58). The placement of buried riprap along the toe and shear key of the tailings dam is proposed to provide protection against erosion during a moraine dam outbreak flood.

While the likelihood of a catastrophic event in the next 10 years is considered low by BGC Engineering (2012a), monitoring of the inside moraine temperature with thermistors, monitoring of any changes in the dam topography indicating subsidence as a result of thawing, systematic inspections for seepage through the dam, and deep drilling to investigate the foundations of the moraine are recommended by BGC.

Kumtor has now started a study of the engineering options relating to a possible Petrov Lake outburst flood. An early warning system will be installed before the end of 2012; remedial measures to armour and protect the tailings facility are under evaluation, and a spillway will be designed in 2013 and installed with the appropriate approvals in the winter 2013/14 with the aim of lowering the lake level by three metres.
18.3.4 Options to Increase the Ultimate Tailings Capacity

The currently approved ultimate limit of the tailing management facility has insufficient capacity to store all of the 93 million tonnes of ore to be processed in the current LOM plan (Table 20). The existing facility will reach its permitted capacity (1.5 m freeboard at a dam elevation of 3,670.5 metres) in 2020. The capacity shortfall amounts to approximately 50 million tonnes of ore or 33 million cubic metres of tailings. Three options have been investigated to provide the additional tailings capacity in a preliminary fashion by BGC Engineering (BGC, 2012). These are shown in Figure 31 and are described below:

**Option 1:** Raising the existing tailings dam. The additional tailings can be stored in the existing tailings facility by raising its dam by seven metres to a crest elevation of 3,677.5 m. The final dam height would likewise rise from 42.5 m to 49.5 m. This estimate is based on an existing volume versus elevation curve and assumes a freeboard of 1.5 m. The relocation of the existing water treatment plant to the north end of the impoundment is required and is currently underway. The proposed final dam height will be marginally below 50 m, at which height it would require a Class I designation from the existing Class II designation as defined by the 2007 Kyrgyz International Building Standards – Hydraulic Engineering Constructions. If the height reaches or exceeds 50 m and is placed into Category 1, more stringent design, monitoring and reporting criteria would be applied to operate the structure. Higher static and seismic-dynamic Factors of Safety would be required as well as a seismic investigation program that would have to be incorporated as part of the studies for Category I dams. The shear key and buttress configurations for the new ultimate dam have been evaluated by BGC using the 2007 ultimate dam design parameters. No fatal flaws are indicated that would prevent the raising of the dam by the seven metres beyond its currently approved final elevation. However, additional sub-soil investigations for the installation of the expanded shear key will be required, including drilling and test pit excavations. Additional creep deformation analyses will also be required.

**Option 2:** New Tailings Facility at Site 3. This was one of the potential sites identified by preliminary work in 2007 and would consist of two cells. This impoundments site is located away from any significant creeks and lie above wetlands “associated with the outlet from Jukuchak Lake and any risk arising from a potential for glacial lake outflow from the upper Jukuchak Lake basin... “BGC 2012, page 11. The site is, however outside of Kumtor Concession.

“This favourable location should minimize the risks associated with design, construction and operation of the dams and runoff diversion works. Nevertheless, a significant amount of geological and geotechnical investigation, monitoring, and design work will be needed to confirm the feasibility of this Option for tailings storage.” (Ibid). Such investigations would include geophysical surveys, test-hole drilling and trench or pit excavations.

**Option 3:** New Tailings Facility at Site 8 which is located “above ... wetlands associated with the Kumtor River, and set back from the banks of the Chong-Sarytor River.” (Ibid,
page 13). As is the case for Option 2, geological and geotechnical investigations and design work are required to prove the feasibility of this site.

While the geotechnical data for Option 1 (adding to the existing facility) are well known requiring only local additional site investigations, there is no reliable information of comparable quality available for the other two options, and their definite suitability to serve as tailings facilities is currently unproven. In addition, BGC (2012) have provided a table comparing the three options in their report, and this is reproduced below.

**Table 25 Comparison of Additional Tailings Storage Options**

<table>
<thead>
<tr>
<th></th>
<th>Option 1</th>
<th>Option 2</th>
<th>Option 3</th>
</tr>
</thead>
<tbody>
<tr>
<td>Fill Quantities</td>
<td>4.7 million m$^3$</td>
<td>11.1 million m$^3$</td>
<td>38.8 million m$^3$</td>
</tr>
<tr>
<td>Excavation Quantities</td>
<td>1.0 million m$^3$</td>
<td>4.6 million m$^3$</td>
<td>9.1 million m$^3$</td>
</tr>
<tr>
<td>Liner</td>
<td>90 000 m$^2$</td>
<td>746 000 m$^2$</td>
<td>854 000 m$^2$</td>
</tr>
<tr>
<td>New Tailings Line</td>
<td>Raise existing line</td>
<td>6 km</td>
<td>3 km</td>
</tr>
<tr>
<td>Water Treatment Plant</td>
<td>Relocate existing</td>
<td>New</td>
<td>New</td>
</tr>
<tr>
<td>Diversion Channel</td>
<td>Existing</td>
<td>5.0 km</td>
<td>2.1 km</td>
</tr>
<tr>
<td>Additional Ecological Impact</td>
<td>Very Low</td>
<td>Low</td>
<td>Low</td>
</tr>
<tr>
<td>Location</td>
<td>Inside Concession</td>
<td>Outside Concession</td>
<td>Inside Concession</td>
</tr>
</tbody>
</table>

Option 1 is the preferred solution. The capital budget in Table 29 in Section 21.2 includes provision for the annual raising of the dam to its currently approved top elevation. A provision for the capital required for the additional capacity for 33 million cubic metres has also been included, anticipating that the dam of the existing facility will be permitted to be increased by 7 metres (Option 1). If either of Options 2 or 3 turn out to be the required solution, additional capital expenditures would be required to make one of the two options operational by 2020.
18.3.5 Conclusions

The levels of deformation encountered in the tailings dam foundation to date are not excessive and fall well within the range of movements experienced by other such dams around the world. The Kumtor Project tailings dam material is strain tolerant and shows little effect of the minor deformations. The deformation data collected by KOC have been reviewed and interpreted by geotechnical consultant, (Golder 2012a) and by the Kyrgyz Republic Institute of Rock Mechanics (KRIM) (Chukin 2012a, 2012b) with the conclusion that the remedial works undertaken to date are effective. A numerical deformation model that was initially developed by BGC in 2007 in support of the shear key design showed reasonable agreement between observed and predicted dam displacements until 2010. In 2010, the model was re-calibrated against the observed data and the updated analysis confirmed that the dam could be safely raised another 6.5 metres above its ultimate approved design elevation at the proposed rate of dam construction.

The responsible author is of the opinion that the shear key and buttress are performing as designed and that the tailings dam is under no threat of failing. Provided the deformation data continue to show a stable dam with limited ongoing deformations, the minimum tailings beach width is maintained between the dam and the tailings pond, the tailings pond levels are maintained at the minimum height possible, and the dam is constructed at the proposed rate through 2022, there is no apparent technical reason why the dam cannot be raised another 6.5 metres from its currently-permitted ultimate design elevation of 3,670.5 metres.

The long-term threat posed by a possible glacial lake outburst flood from Petrov Lake to the tailings management facility has been recognized by Kumtor. A plan has been formulated to lower the water level of Petrov Lake in the winter of 2013/14 by building a spillway. Other flood protection measures are under evaluation.

To accommodate the 33 million cubic metres of future additional tailings that will be produced as a result of the enlarged Central Pit but that cannot be stored in the existing facility as currently permitted, three additional storage options have been evaluated. Raising of the existing tailings dam by seven metres is the preferred solution that is safe to do and has substantial ecological and economic advantages and is not expected to present any permitting issues on technical grounds. The second option is outside of the Kumtor Concession and therefore not practical. The third option, while located inside the concession, requires a large amount of earthworks.
19 MARKET STUDIES AND CONTRACTS

All gold doré produced by the Kumtor Project is purchased at the mine site by Kyrgyzaltyn under the Restated Gold and Silver Sale Agreement for processing at its refinery in the Kyrgyz Republic. Under the Restated Gold and Silver Sale Agreement, Kyrgyzaltyn is required to pay for all gold delivered to it based on the afternoon fixing of the price of gold on the London Bullion Market by the 12th calendar day following the date on which a shipment of gold doré is collected by it from the Kumtor mine. The obligations of Kyrgyzaltyn are partially secured by a pledge of a portion of the Centerra shares owned by Kyrgyzaltyn. All gold doré produced by the mine to date has been purchased by Kyrgyzaltyn pursuant to these arrangements without incident.
20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Permits and Licenses

All mining and related activities are carried out in accordance with licenses and permits issued by Kyrgyz government agencies based on the laws of the Kyrgyz Republic. The Restated Investment Agreement provides that KGC is entitled to all licences, consents, permits and approvals of the Government of the Kyrgyz Republic necessary or convenient for the operation of the Kumtor Project.

The Kumtor Health, Safety and Environment (HSE) Policy & Compliance Departments spend considerable time and resources ensuring that all permits and licenses are received and remain current.

Annual mine plans are submitted for approval to the National Mining Technical Oversight Agency (GGTN) for approval, with the mine plan for 2013 currently being prepared for presentation. The mineral reserves of the Kumtor Project are carried on the State balance which process is managed by the State Agency for Geology. The latest reserve estimate filed was based on the KS-10 block model in 2010, which is reflected in the current State balance. Annual depletion (“5GR”) reports are produced by Kumtor in January or February of each year for the previous year, and the depleted metal is removed from the balance. The new reserve estimate based on the KS-13 block model presented in this report will be filed with the State Agency for Geology to update the State balance sheet.

The Law on Protection of Atmospheric Air dated June 12, 1999 requires that each Kyrgyz enterprise with activities that have a potential negative impact on the environment must develop and maintain an ecological passport (Ecological Passport) providing for the basic levels of impact on the environment, including the level of Maximum Allowable Emission (MAE) and Maximum Allowable Discharge (MAD) and the volumes of waste disposal or utilization. The Ecological Passport is developed by an enterprise every five years and must be approved by the Kyrgyz State Agency of Environmental Protection (KR SAEP). The current Ecological Passport for the mine site developed by Kumtor was approved by the KR SAEP on December 3, 2009. The passport is valid until December 3, 2014. In 2009, KOC also developed and obtained approval by the KR SAEP for an Ecological Passport for the Balykchy Marshalling Yard, and this passport is valid until December 2, 2014. However, due to the relocation of the Balykchy Marshalling Yard in March 2011, a temporary Environmental Permit to operate was issued by the Issyk-Kul Regional Environmental Department. The development of the Ecological Passport for the new Balykchy Marshalling Yard (BMY) is in progress and will be submitted to governmental authority for approval. The Ecological Passport is developed and approved after commissioning of all BMY facilities. As of September 30th, 2012, BMY is at the reconstruction stage and has not yet been fully commissioned.

The Ecological Passport identifies some of the permits and approvals required by KOC for its operations, with annual permits required for MAE and MAD. The MAE permit regulates
the release of emissions into the air. There are two MAD permits regulating the discharge of effluents into surface water bodies, one to operate the tailings area treatment plant and the other to operate the sewage treatment plant. The MAE and MAD permits must be renewed annually within the first quarter of each year, and are designed to ensure that the water quality standards for communal-use streams are met at the mixing zone in the Kumtor River, downstream of site inputs and within the Concession Area.

KOC received the latest MAE and MAD permits on January 12, 2012 which are valid until December 31st, 2012.

Since May 2002, KGC has paid an environmental protection tax, for which the rate and method of calculation are approved by Order of the President and by the Government. The tax rate that Kumtor pays since 2009 is approved by the Kyrgyz Government and Parliament. The tax is comprised of payments for the discharge of effluents and treated sewage, air emissions and waste deposited until 2009 and is forwarded to the Kyrgyz treasury. According to the Restated Investment Agreement, annual payments of the environmental protection tax are now set at $310,000.

On May 14 2011, Kumtor received the general license for disposal of tailings and the general license for disposal of toxic wastes to tailings areas. Both licences are valid until May 30, 2014. Due to the issues with the deformations of the tailings dam described in Section 18.3.2, meetings were conducted regarding stabilization of the dam. Kumtor received approvals from the Kyrgyz authorities for dam stabilization designs, authorizing the continued use of the tailings facility at the time of this report. The authors have been advised that, based on the findings of a working commission consisting of the Mines Inspectorate, the Kyrgyz Agency of Environmental Protection, the Institute of Rock Mechanics and Kumtor representatives, that the remedial measures taken on the tailings dam and shear key are suitable. Geotechnical reviews by Golder and BGC resulted in similar conclusions. A more detailed discussion of this issue is in Section 18.3.2.

The transportation route for dangerous goods such as chemicals and blasting materials must be approved every six months. The approval includes permits for the vehicles transporting the specific materials. Blasting materials are imported from Kazakhstan and China and require import licenses issued by the Kyrgyz Ministry of Internal Affairs upon agreement with the State Agency for Geology. Sodium cyanide is imported from China and requires an import license issued by the Kyrgyz Ministry of Economic Development and Trade upon agreement with a number of other ministries and government agencies. Such licenses are issued for one year. KOC has obtained licenses for Kazakh blasting materials and sodium cyanide to be imported in 2012 with an expiry date of December 31, 2012. KOC also has an import license for Chinese blasting materials with an expiry date of December 31, 2012.

In addition, an annual permit for transit of sodium cyanide through the territory of the Kazakh Republic is required. The permit is issued by the Kazakh Ministry of Industry and New Technology upon agreement with a number of other Kazakh ministries. A transit permit has been issued on January 6, 2012 and is valid for one year.
The water use permit was renewed on March 1, 2012 and remains valid until March 1, 2013. This water permit covers the mine site and allows Kumtor to draw 6.75 million cubic metres of water per year from Petrov Lake, which provides the fresh water requirements for milling and camp operations.

### 20.2 Environmental Compliance

The authors have briefly reviewed, but have not independently verified, statements by Kumtor with respect to the environmental compliance of the Kumtor Project. As per Kumtor’s annual environmental reports posted on their webpage, and consistent with independent environmental due diligence review results by Prizma LLC (2012) and Environmental Resource Management (2012), Kumtor is in substantial compliance with Kyrgyz legislation and good international industry practice. However, particularly the ERM review has included a number of recommendations which are being addressed by an “ERM Audit Action Plan”. As detailed in its publicly available environmental report, Kumtor described several instances of exceeding its discharge permit limits and several of its in-river limits at the end of the mixing zone in 2011, all of which were temporary in nature and did not lead to environmental impact.

Hitchcock and Giovannetti (2009) have reported on a six-day mission to Kyrgyzstan on behalf of EBRD with respect to the loan facility described in Section 2.1 and concluded in 2010. Their report concludes that “Generally the mission raised no major environmental or social concerns and concluded, amongst others, that KOC runs a comprehensive environmental monitoring programme, that KOC human resources policy puts much focus on local recruitment with excellent results, that the compensation issues surrounding the 1998 cyanide spill appear to be resolved with payments made to 4,921 families, and that KOC occupational health and safety performance is as good as or better than world averages in the mining industry. The mission found no significant evidence to substantiate the claims of bad practice made by local NGOs. However, the mission formulated recommendations in the following areas: solid waste management, waste rock management and community development funding and planning.”

### 20.2.1 Environmental Management Action Plan

As part of its obligations to the original lending institutions in connection with the Kumtor Project financing, Kumtor implemented an Environmental Management Action Plan (EMAP) in 1995. The EMAP outlines Kumtor’s environmental and safety commitments, including the regulations applicable to the Kumtor Project. The EMAP was updated in 1999, in 2002, 2008, 2009 and again in 2010 to reflect the maturing operations. New monitoring stations were added to the EMAP monitoring program to cover the new mining activities in the Southwest Deposit area, underground development and dewatering discharge points.

The Restated Investment Agreement provides that Kumtor will continue to be obligated to operate in accordance with mine and operating plans that seek to limit the environmental impact of the project and protect human health and safety in accordance with good international mining
practices. Specifically, Kumtor continues to be obligated to operate in material compliance with the standards applicable under the EMAP.

The standards applicable include the most stringent of:

1. The environmental laws of the Kyrgyz Republic and the current KGC Occupational Health and Safety guidelines; or


20.2.2 Environmental Management System

In 2000, Kumtor developed a formal Environmental Management System (EMS) following ISO-14001 standards for determining and managing environmental aspects associated with its activities. The EMS addresses all impacts of the operation on the environment and monitors compliance with the various permits issued by the Kyrgyz authorities. The system provides scheduled monitoring, engineering controls and reporting on the following areas:

- Effluent treatment plant
- Tailings management facility
- Mill site and mine waste dumps runoff effluents
- Acid generation potential testing and recommendations
- Dust control
- Spill incidents on site and off site
- Hazardous materials handling
- Environment impact monitoring
- Planning for site decommissioning and rehabilitation
- Potable water treatment system
- Sewage operation
- Landfill operation and inventory

In addition to internal monitoring, several external audits have been undertaken since 2005:

1. An assessment of the tailings management system was undertaken by BGC in 2005 using Mining Association of Canada (MAC) guidelines (BGC Engineering Inc. 2005). The results of the audit showed that Kumtor conformed to the MAC guidelines and that the tailings management facility is being managed comprehensively and effectively, but the audit identified a few items where improvements are possible. Recommendations provided in the 2005 BGC Audit Report were taken into consideration within and/or were implemented in full except for the risk assessment which was completed the following year by Golder, 2006. The question of the stability of the tailings dam is discussed in Section 18.3;
2. In November 2006, Kumtor underwent a systems assessment by independent auditors from Blue Heron – Solutions for Environmental Management Inc. and WESA that covered environmental as well as health and safety issues. The assessment found that the general condition of the mine and health, safety and environmental awareness of the site personnel were excellent, and that the site and buildings were neat, with materials and wastes well organized. No evidence of spills or environmental damage was observed during the assessment. The assessment outlined areas of particular strength as well as opportunities for improvement. The issues needing improvement have been acknowledged by site management;

3. In September 2008 WESA Inc. audited the industrial hygiene program at Kumtor. The aim of the audit was to provide an independent assessment of the current status of the industrial hygiene exposure monitoring program and evaluation of industrial hygiene monitoring practices. Follow-up training was arranged for industrial hygiene staff as a recommendation from the audit.

The assessment reviewed the following key areas as outlined in the EMAP:

i. Hazardous materials usage, labelling, storage, transport and emergency response;

ii. Environmental protection including protection of wildlife, site drainage, site emissions (air and water), waste rock disposal, etc;

iii. Closure, decommissioning and reclamation;

iv. Spill containment, control and clean-up; and

v. Site policies, programs, training, regulations and reporting procedures.

Kumtor is in substantive compliance with the requirements of the EMAP; and

4. In October 2008 Citrus Partners LLP (Citrus) audited KOC’s compliance against the Environmental Management Action Plan (EMAP) Revision 3 (dated November 2008). The Report reviewed each provision of the EMAP and provides a text-based summary of compliance and other relevant issues. An assessment of the processes in place for reclamation and closure planning was carried out in August, 2010 by WESA Inc. The objectives of the assessment were to:

- review past reclamation activities
- review current reclamation activities
- review future yearly plans
- review Conceptual Closure Plans (CCP’s discussed in Section 20.3)
20.3 Closure Provisions

Under EMAP, Kumtor is required to update its Conceptual Closure Plan (CCP) every three years. This approach allows for the development and adaptation of the CCP, provides a period for testing and monitoring of several years to evaluate the various options contemplated by the CCP, and is followed by the development of a Final Closure Plan (FCP) two years prior to the end of mine life. The FCP will consider the results of the testing and monitoring as well as any changes to the environmental, regulatory and social environment that may have occurred over the life of the mine.

Under the Restated Investment Agreement, all immovable infrastructure items will become the property of the Government of the Kyrgyz Republic at the end of the mine life. This includes roads, buildings including the mill building, accommodations and any other related facilities but not the operating machinery.

A decommissioning plan was developed in 2008 as required by the Kumtor EMAP and by the Agency Lenders, in accordance with generally accepted environmental practices and applicable regulatory requirements, including World Bank guidelines and the laws and regulations of the Kyrgyz Republic, Canada and Saskatchewan. The decommissioning plan covers all aspects of the mining project, including the open pit (which will become a lake), mill complex and surrounding area, tailings basin, stockpiles and other surface facilities. Equipment, building and other structures will be salvaged to the extent possible.

The 1999 version of the CCP was described in the prospectus issued on occasion of the Centerra IPO, with the future decommissioning and reclamation costs estimated at $20.4 million. In 2004, a second CCP was developed by Lorax Environmental Ltd. for review by Centerra, and translated and submitted to the Kyrgyz authorities in 2005 for their information. The Lorax plan is more detailed and is technically different from the previous version. It uses a 1.5-metre thick, hydraulically-placed waste rock cover for the tailings to prevent evaporation and oxidation, deals in detail with future pit chemistry and water management, including shortcomings in the current data base, and abandons the idea of high-altitude re-vegetation in favour of contouring with glacier till material. The Lorax report is, based on the scientific knowledge available at the end of 2003.

In 2007 Golder Associates completed a third CCP which reinforced the Lorax 2004 CCP. The data presented indicated that the acid rock drainage (ARD) potential of both waste dumps and tailings is very low, but that sulphate released from the waste dumps may present a long-term concern. The report makes recommendations for further data collection and monitoring of the various aspects important for the closure plan such as ice movement under the load of the waste dumps, water flow and water quality into the open pits, further geochemical characterisation of the tailings and re-engineering of the waste dumps to limit their interchange with meteoric water in an effort to minimize sulphate discharge particularly in the Davidov drainage as a result of sulphide oxidation.
The Golder plan provided a total closure cost estimate of $24.2 million, slightly higher than the original 1999 closure plan. The major cost items are the tailings cover and spillway for the tailings dam. Since both the Lorax and Golder’s plans recognize that the waste rock dumps will provide neutral drainage, the additional operating years added to the mine life as a result of the KS-13 LOM plan will not result in a significant increase in the closure cost.

The original 1999 Closure Plan anticipated that the salvage value from the sale of plant machinery and equipment and other moveable assets would be applied against final reclamation costs. As part of the Golder 2007 CCP, an international salvage house estimated the salvage value to be $8.4 million. A reclamation trust fund of $16 million was established for the future costs of reclamation, net of estimated salvage value. KOC has commissioned an update of the CCP from Lorax in 2010 (Lorax 2011). With the new LOM plan, the increased mining equipment fleet and process equipment purchases, the plan provided a timely update for the trust fund. The updated CCP arrived at total funding requirement of $29.5 million without allowance for any salvage value. Funding is provided by KOC contributions over the mine life based on ounces of gold sold. As at March 31, 2012 the balance in the fund was $11.3 million. The balance of the estimated future costs will be funded over the remaining LOM plan, prorated based on annual gold production.

In 2012, Lorax was further commissioned to complete a scoping review of the KS13 expanded LOM plan and its impact on closure and related planning. Their review and assessment concludes:

While the KS-13 LOM plan will result in significantly greater quantities of waste rock removal and storage, primarily on the Davidov waste rock dump, the increased tonnages associated with the mine expansion are not anticipated to result in significant environmental changes when compared to those associated with the existing mine plan.

Lorax’s scoping review will form the basis for the 2013 CCP update, scheduled consistent with the EMAP requirement for CCP revision every three years.
20.4 Human Resources

In November 2012 the operation employed a total of 2,826 permanent employees, distributed by department and by citizenship as follows:

<table>
<thead>
<tr>
<th>Department</th>
<th>Kyrgyz Citizens</th>
<th>Expatriates</th>
<th>Total Employees</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining</td>
<td>1,000</td>
<td>5</td>
<td>1,005</td>
</tr>
<tr>
<td>Milling</td>
<td>132</td>
<td>1</td>
<td>133</td>
</tr>
<tr>
<td>Site Administration</td>
<td>673</td>
<td>16</td>
<td>689</td>
</tr>
<tr>
<td>Maintenance</td>
<td>552</td>
<td>53</td>
<td>605</td>
</tr>
<tr>
<td>Bishkek Administration</td>
<td>210</td>
<td>9</td>
<td>219</td>
</tr>
<tr>
<td><strong>Sub-Total</strong></td>
<td><strong>2,567</strong></td>
<td><strong>84</strong></td>
<td><strong>2,651</strong></td>
</tr>
<tr>
<td>Exploration</td>
<td>77</td>
<td>14</td>
<td>91</td>
</tr>
<tr>
<td>Underground Project</td>
<td>74</td>
<td>10</td>
<td>84</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>2,718</strong></td>
<td><strong>108</strong></td>
<td><strong>2,826</strong></td>
</tr>
</tbody>
</table>

The proportion of Kyrgyz citizens in the permanent workforce is 96%, having increased from 82% at the beginning of the operation as a result of the training programs that Kumtor has conducted, and reflects a policy of involving citizens of the Kyrgyz Republic at all levels in the workforce as soon as the necessary skills and experience have been acquired. Under the Restated Investment Agreement, Kumtor must use commercially reasonable efforts to increase the percentage of citizens of the Kyrgyz Republic in its workforce.

Not included in Table 26 are 399 temporary and permanent Kyrgyz contractors that perform a variety of work.

On February 6, 2012, unionized employees of the Kumtor Project commenced a ten-day illegal work stoppage during which time production was suspended. The current collective agreement is valid to December 31, 2012.

With the continued expansion of open pit mining operations, additional national and expatriate staff will be required mainly in the mining and maintenance departments. All costs related to the expansion of the workforce required over the next several years have been included in the updated LOM operating and capital costs.

20.5 Emergency Response

In May 1998, a truck operated by KOC en route to the mine accidentally overturned and spilled approximately 1.8 tonnes of sodium cyanide into the Barskaun River, which in turn drains into Lake Issyk-Kul (Figure 2). This spill incident resulted in extensive review of the mine’s...
Emergency Response Plan (ERP) and its hazardous material transportation procedures by local authorities, lending agencies and KOC. A revised ERP took effect December 1999. Since then, KOC has conducted quarterly mock exercises to test different aspects of the ERP including response time, effective communications and the skills of the emergency response team. The ERP has most recently been updated and approved in May 2008 to ensure notification protocols remain valid and improvements from the mock exercises are incorporated in the plan. The authors have been advised by KOC that this revision remains valid and meets all Kyrgyz legal requirements and follows international standards.

In October 2006, Water and Earth Science Associates Ltd. of Ottawa (WESA) was retained by Centerra to audit the transportation of cyanide from the warehouse facility in Urumqui, China to the mine site. The Cyanide Transportation Verification Protocol issued by the International Cyanide Management Institute (ICMI) in September 2006 was used to conduct the audit. In their report, WESA commended KGC for their adherence to international standards for transport of solid sodium cyanide. KGC was found to be “in full compliance” with all aspects of the transportation code with respect to the transportation cycle from the warehouse in China to the mine site in Kyrgyzstan.

In 2007, Citrus Partners was appointed as Environmental Consultant to a bank in connection with a proposed credit facility for Centerra. In their report, Citrus (2007) concluded that “Given the difficult nature of the access roads to the Kumtor Project, there remains a significant risk of further accidents. Appropriate driver training, truck maintenance, road maintenance and materials storage and packaging will be required on a continual basis to mitigate this risk.” (Citrus Partners 2007, page 3).

Kumtor has since implemented its vehicle accident reduction program (VARP) and continues to improve it to manage mobile equipment and vehicle risk. Subsequent to the above audits and improvements, the International Cyanide Management Institute certified Kumtor under its Cyanide Code early in 2012.

20.6 Health and Safety

The authors have briefly reviewed, but have not independently verified, statements by Kumtor with respect to occupational health and safety issues at the Kumtor Project. Centerra conducts an annual evaluation of the health, safety and environmental (HSE) management system’s implementation as part of management’s annual incentive.

20.6.1 Health and Safety Management Systems

KOC has developed and implemented a Health and Safety Management System (HSMS) that is based on the Occupational Health and Safety Assessment Series (OHSAS) 18001 standards developed by the British Standards Institute.
Annual targets and objectives in the HSMS are set for both the entire operation and for individual departments. Annual reviews are conducted by senior management as well as with employees. Tracking of action items, targets, objectives and deficiencies is done with the use of a Corrective and Preventative Action Ledger (CPAL System). The CPAL tracking system was placed on the KOC site intra-net to allow both the Safety and Environmental departments to enter corrective action items and to allow other departments to address and respond to any deficiencies or non-conformances.

The results of an external audits by Blue Heron and WESA in November of 2006 and SENES in October 2009, that covered environmental in addition to health and safety issues, has been described in the preceding section.

### 20.6.2 Workers Occupational Health and Safety Program

As outlined and defined in the KOC Health and Safety Management System described above, safety elements are incorporated in the design and operational procedures of the mine. The open-pit operation is carried out under safe blasting procedures. Pit slopes are designed to prevent toppling and outright failure, and their stability is constantly monitored for safety. The pit design has incorporated rock fall berms, and the haul road is constructed 35 metres wide to allow two haul trucks to pass safely with proper safety berms and drainage ditches. Waste is placed in the central valley basin using mostly a top to bottom dumping approach. The waste pile height is currently restricted to 90 metres to avoid slope instability. Dumping berms and procedures are in place to avoid incidents with equipment. A monitoring program is in place to ensure that waste pile deformations due to shifting ground or weather conditions are detected and addressed. Pit operators are trained in the safe handling of heavy equipment.

Process and effluent treatment facilities were designed to address issues of dust control, noise, toxic chemicals, moving pieces of stationary equipment, potential electrical and fire hazards.

The camp complex, providing accommodation, kitchen, dining and recreation facilities, is equipped with heat and smoke detectors, an integrated sprinkler system and hand-held fire hoses and extinguishers.

The transportation of materials and personnel, both on- and off-site, is undertaken under specific accident prevention and safety procedures that include speed limitations and control signs as required. All vehicles and personnel buses are equipped with two-way radios for emergencies. All transport equipment units have a preventive maintenance program. The mine site is under security with authorized entry policy enforced by specialized personnel.

At the mine site, medical staff, including two doctors, provide first aid, routine medical services and operate a fully-equipped first-aid clinic centre. An industrial hygiene monitoring program is conducted with analysis of samples contracted to an independent laboratory. Two ambulances, each equipped to accommodate a stretcher and containing appropriate medical
supplies, are on standby at the mill building. Emergency medical evacuation from the mine site is available if necessary.

All KOC and contractor employees are trained in the use of the Five Point Safety System and the Work Place Hazardous Information System before commencing work at the site. First aid, mine rescue and fire-fighting training is provided at the site on a regular schedule which accounts for approximately 70,000 man-hours of new employee and refresher training per year. Full mine rescue and fire-fighting teams are always available on site with current qualifications and training to address any emergency. The site is equipped with a fire truck. Hydrants were installed strategically throughout the major facility areas. Fire-fighting equipment is stored at convenient locations, ready for use.

20.6.3 Health and Safety Performance

Lost-time injuries have occurred at a rate of 2 to 8 in each of the years 1997 to 2011 with the last fatality occurring in 2009 when a haul truck collided with a KAMAZ bus carrying workers to the Central Pit. The deceased was not wearing a seatbelt and was thrown from the passenger door of the bus that opened upon collision.

Notwithstanding the fatalities, the lost-time accident frequency rate has declined from the range of 0.42 per 200,000 man-hours in 1997 to a level of less than 0.03 in 2012. From a statistical point of view, this is a good record which compares favourably with lost-time frequency rates assembled by such organizations as the Ontario Mining Association, which has reported frequency rates in the range of 0.6 to 1.4 for the period 2002 to 2011.

20.6.4 Socio-Environmental Action Plan

On November 16, 2010, Centerra entered into a three-year $150 million revolving credit facility with the EBRD (Credit Agreement). Among various covenants stipulated in the Credit Agreement which Centerra and its subsidiaries must comply with, is a covenant to undertake the implementation of and continued adherence with an Environmental and Social Action Plan (ESAP). The ESAP was developed with input from KOC and is intended to support ongoing compliance with agreed standards defined as Canadian Federal, Province of Saskatchewan, and World Bank environmental and social standards.

20.7 Economic and Budgetary Impact on the Kyrgyz Republic

The importance of the Kumtor mining operation to the Kyrgyz Republic is apparent from the following statistics (Kyrgyz National Statistics Committee preliminary reports):

- Kumtor’s share in GDP in 2011 was 11.7%;
- Kumtor’s share in the total industrial output is 26%; and
- Gold contributed 51% of the national exports in 2011.
In 2011, direct contributions and expenditures within the Kyrgyz Republic (including taxes, wages, and payments to local suppliers, community contributions and development projects) totaled more than $383 million. Since operations began in 1994, Kumtor expenditures and payments within the Kyrgyz Republic has been $1.85 billion to the end of 2011.
21  CAPITAL AND OPERATING COSTS

21.1 Historical Operating Cost Performance

The Kumtor Project has had a history of good operating costs until about 2005, but costs have increased in the past several years, an experience shared by many other mining operations as the result of increases in labour, energy and material costs. Table 27 presents a summary.

In Table 27, “Others” includes VAT and excise taxes, and customs duties. Starting in 2004, operating costs are net of by-product revenues and include refining fees, but exclude management fees paid to KOC when KOC became a subsidiary of the newly created Centerra. The capitalized pre-strip mining costs in 2006, 2007, 2010, 2011 and the first nine months of 2012 were included for the calculation of mine unit operating costs, but were excluded from the cash costs per ounce of gold. Cash costs per ounce of gold produced have been negatively affected in 2006 and 2007 by the overall low gold head grades and increased waste mining rates. The line item Administration excludes head office costs in Toronto but includes all costs incurred in the Republic of Kyrgyzstan.
Table 27 Historical Operating and Capital Costs, 1997 to September 30, 2012

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<tbody>
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<td><strong>Production</strong></td>
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<tr>
<td>Mining (Total) (T x 1000)</td>
<td>96 000</td>
<td>43 300</td>
<td>52 500</td>
<td>54 300</td>
<td>77 700</td>
<td>84 855</td>
<td>81 038</td>
<td>85 421</td>
<td>114 781</td>
<td>120 515</td>
<td>115 544</td>
<td>116 466</td>
<td>150 604</td>
<td>109 425</td>
<td>1 302 449</td>
</tr>
<tr>
<td>Milling (T x 1000)</td>
<td>14 575</td>
<td>5 498</td>
<td>5 470</td>
<td>5 611</td>
<td>5 631</td>
<td>5 654</td>
<td>5 649</td>
<td>5 696</td>
<td>5 545</td>
<td>5 577</td>
<td>5 780</td>
<td>5 594</td>
<td>6 020</td>
<td>3 209</td>
<td>85 509</td>
</tr>
<tr>
<td>Gold Production (Oz x 1000)</td>
<td>1 758</td>
<td>670</td>
<td>753</td>
<td>529</td>
<td>678</td>
<td>657</td>
<td>501</td>
<td>304</td>
<td>301</td>
<td>556</td>
<td>525</td>
<td>568</td>
<td>583</td>
<td>126</td>
<td>8 506</td>
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<tr>
<td><strong>Costs</strong></td>
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</tr>
<tr>
<td>Mining ($ x 1000)</td>
<td>74 800</td>
<td>26 000</td>
<td>33 600</td>
<td>37 500</td>
<td>40 508</td>
<td>47 804</td>
<td>62 116</td>
<td>76 678</td>
<td>143 364</td>
<td>113 175</td>
<td>123 324</td>
<td>157 838</td>
<td>46 461</td>
<td>1 031 065</td>
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<tr>
<td>Milling ($ x 1000)</td>
<td>88 900</td>
<td>29 300</td>
<td>30 900</td>
<td>29 000</td>
<td>28 900</td>
<td>30 585</td>
<td>32 346</td>
<td>37 038</td>
<td>39 412</td>
<td>50 242</td>
<td>54 663</td>
<td>56 103</td>
<td>63 515</td>
<td>41 452</td>
<td>612 356</td>
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<td>Administration ($ x 1000)</td>
<td>122 900</td>
<td>35 600</td>
<td>33 300</td>
<td>31 300</td>
<td>34 500</td>
<td>35 743</td>
<td>35 611</td>
<td>42 235</td>
<td>43 535</td>
<td>57 359</td>
<td>54 495</td>
<td>53 568</td>
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<td>49 647</td>
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<td>148</td>
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<td>231</td>
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(1) January 1, to September 30th, 2012

(2) Unit mining costs for 2006, 2007, 2010, 2011 and 2012 (1-9) equal operating mining costs plus capitalized pre-stripping mining costs divided by the total tonnes mined.
21.2 Capital and Operating Cost Estimates

The following material assumptions have been used in the life-of-mine plans, estimates of operating and capital costs and mineral reserve and resource estimates:

- A gold price of $1,350 per ounce,
- Exchange rates:
  - $1USD:$1.00 CAD
  - $1USD:47 Kyrgyz Som
- Diesel fuel price assumption: $0.80/litre delivered to the Balykchy yard.

Based on the operating cost experience to date, and anticipating the additional haulage costs associated with the expansion of the Central Pit and with the more distant Sarytor and Southwest pits, the LOM plan projects operating costs that are summarized in Table 28. Note that the net revenue taxes in Table 28 are based on a gold price of $1,350 per ounce as assumed for the mineral resource and reserve estimation process. The direct operating cost per ounce of gold produced which does not include capital pre-stripping, direct capital costs or the new revenue based tax varies each year depending on the mill head grade in addition to operating conditions and averages $422 per ounce of gold produced for the period of September 30, 2012 to the end of the life of mine in 2026. Total costs including the direct operating costs described above as well as capital pre-stripping, direct capital costs or the new revenue based tax averages $917 per ounce of gold produced for the period of September 30, 2012 to the end of the life of mine in 2026.

For the purpose of calculating cash costs per ounce, regular mining costs of ore and waste are included, but capitalized pre-stripping costs are not. Cash cost per ounce is a non GAAP measure and does not include revenue-based taxes. For a complete description refer to the Centerra’s most current Management’s Discussion and Analysis. Net Revenue Based Taxes as defined by the Restated Investment Agreement have been calculated using a gold price of 1,350 per ounce.

The capital cost forecast (excluding capital pre-stripping costs) shown in the lower part of Table 28 for the LOM plan is further broken down in Table 29. The total life-of-mine capital expenditures required to exploit the mineral reserves in the LOM plan is estimated at $726 million, which includes total sustaining capital amounts of nearly $557 million and growth capital of $169. Exploration expenditures are not included in Table 28.
### Table 28 Projected Operating and Capital Costs, 2012 to 2026

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<td></td>
<td>9.89</td>
</tr>
</tbody>
</table>

(1) October 1 to December 31, 2012
### Table 29 Detailed Projected Open Pit Capital Costs, 2012 - 2028

<table>
<thead>
<tr>
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</thead>
<tbody>
<tr>
<td>Mine Pre-Stripping ($ x 1000)</td>
<td>36 747</td>
<td>213 311</td>
<td>248 435</td>
<td>119 755</td>
<td>93 898</td>
<td>256 926</td>
<td>86 165</td>
<td>149 911</td>
<td>279 039</td>
<td>29 796</td>
<td>118 813</td>
<td>51 572</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>1 684 369</td>
<td></td>
</tr>
<tr>
<td>Mine Equipment ($ x 1000)</td>
<td>44 560</td>
<td>12 676</td>
<td>52 500</td>
<td>10 938</td>
<td>16 402</td>
<td>14 435</td>
<td>11 000</td>
<td>1 905</td>
<td>2 355</td>
<td>930</td>
<td>350</td>
<td>100</td>
<td>100</td>
<td></td>
<td></td>
<td></td>
<td>109 736</td>
<td></td>
</tr>
<tr>
<td>Mine Support Equipment ($ x 1000)</td>
<td>2 378</td>
<td>10 893</td>
<td>16 402</td>
<td>13 719</td>
<td>14 435</td>
<td>11 000</td>
<td>1 905</td>
<td>2 355</td>
<td>930</td>
<td>350</td>
<td>100</td>
<td>100</td>
<td>75 164</td>
<td></td>
<td></td>
<td></td>
<td>75 164</td>
<td></td>
</tr>
<tr>
<td>Mine Equipment Maintenance ($ x 1000)</td>
<td>3 858</td>
<td>29 302</td>
<td>35 750</td>
<td>24 998</td>
<td>39 930</td>
<td>39 384</td>
<td>28 998</td>
<td>34 745</td>
<td>32 781</td>
<td>26 898</td>
<td>17 400</td>
<td>7 200</td>
<td>150</td>
<td></td>
<td></td>
<td></td>
<td>321 468</td>
<td></td>
</tr>
<tr>
<td>Mill ($ x 1000)</td>
<td>465</td>
<td>3 197</td>
<td>5 000</td>
<td>29 615</td>
<td>2 500</td>
<td>2 500</td>
<td>2 500</td>
<td>2 655</td>
<td>589</td>
<td>200</td>
<td>1 300</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td>55 521</td>
<td></td>
</tr>
<tr>
<td>Tailing Facility ($ x 1000)</td>
<td>7 244</td>
<td>3 450</td>
<td>5 080</td>
<td>5 000</td>
<td>16 895</td>
<td>8 041</td>
<td>7 361</td>
<td>5 977</td>
<td>6 200</td>
<td>5 000</td>
<td>3 800</td>
<td>1 000</td>
<td>175</td>
<td>50</td>
<td></td>
<td></td>
<td>75 273</td>
<td></td>
</tr>
<tr>
<td>Surface Infrastructure ($ x 1000)</td>
<td>2 006</td>
<td>10 987</td>
<td>30 122</td>
<td>10 000</td>
<td>1 000</td>
<td>500</td>
<td>750</td>
<td>250</td>
<td>175</td>
<td>150</td>
<td>161</td>
<td>118</td>
<td>200</td>
<td>25</td>
<td>28</td>
<td></td>
<td>56 472</td>
<td></td>
</tr>
<tr>
<td>Other ($ x 1000)</td>
<td>1 758</td>
<td>597</td>
<td>975</td>
<td>3 85</td>
<td>500</td>
<td>200</td>
<td>256</td>
<td>175</td>
<td>175</td>
<td>3 500</td>
<td>6 500</td>
<td>2 800</td>
<td>6 207</td>
<td>4 300</td>
<td>3 725</td>
<td>32 053</td>
<td></td>
<td></td>
</tr>
<tr>
<td>TOTAL ($ x 1000)</td>
<td>91 772</td>
<td>288 297</td>
<td>392 634</td>
<td>203 544</td>
<td>157 263</td>
<td>327 405</td>
<td>128 607</td>
<td>197 297</td>
<td>321 577</td>
<td>69 796</td>
<td>148 813</td>
<td>65 790</td>
<td>8 957</td>
<td>4 500</td>
<td>3 803</td>
<td>2 410 056</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

(1) October 1 to December 31, 2012
22 ECONOMIC ANALYSIS

The material economic assumptions used for the calculations presented in this section have been stated in Section 21.2.

22.1 Cash-Flow Forecast

Using a price of gold of $1,350 per ounce as assumed for the mineral resource and reserve estimation process, the open pit LOM plan (Table 20) and the operating and capital cost forecasts (Table 28) have been used to project the net cash flow for the Kumtor Project from October 1, 2012 to the end of 2026. As is shown in Table 30, the total net cash flow discounted at 8% amounts to over $1.9 billion dollars after accounting for all operating costs, capital expenditures related to the open pit operation and taxes under the Restated Investment Agreement. Surface exploration expenditures identified in Table 32, which total $16 million for the fourth quarter of 2012 plus all of 2013, and any additional exploration expenditures in subsequent years have been excluded.
Table 30  Projected Mine Net Cash Flow, October 1, 2012 to 2028

<table>
<thead>
<tr>
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<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>Gold Produced (oz x 1000)</td>
<td>228</td>
<td>660</td>
<td>661</td>
<td>660</td>
<td>663</td>
<td>665</td>
<td>664</td>
<td>658</td>
<td>663</td>
<td>664</td>
<td>526</td>
<td>331</td>
<td>145</td>
<td>107</td>
<td>107</td>
<td>7,875</td>
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<tr>
<td>Gross Revenue From Gold ($ x 1000)</td>
<td>307,731</td>
<td>810,439</td>
<td>891,479</td>
<td>892,122</td>
<td>891,415</td>
<td>894,809</td>
<td>897,317</td>
<td>896,274</td>
<td>888,276</td>
<td>895,019</td>
<td>896,853</td>
<td>891,584</td>
<td>419,250</td>
<td>195,323</td>
<td>145,026</td>
<td>10,631,915</td>
</tr>
<tr>
<td>Estimated Silver Credit ($ x 1000)</td>
<td>1,446</td>
<td>3,809</td>
<td>4,190</td>
<td>4,190</td>
<td>4,206</td>
<td>4,217</td>
<td>4,212</td>
<td>4,175</td>
<td>4,207</td>
<td>4,215</td>
<td>3,340</td>
<td>1,970</td>
<td>918</td>
<td>682</td>
<td>49,970</td>
<td></td>
</tr>
<tr>
<td>Total Gross Revenues ($ x 1000)</td>
<td>309,177</td>
<td>814,248</td>
<td>895,669</td>
<td>896,315</td>
<td>895,604</td>
<td>899,015</td>
<td>901,534</td>
<td>900,486</td>
<td>892,451</td>
<td>899,226</td>
<td>901,068</td>
<td>713,924</td>
<td>421,220</td>
<td>196,241</td>
<td>145,707</td>
<td>10,681,885</td>
</tr>
<tr>
<td>Operating Costs ($ x 1000)</td>
<td>69,102</td>
<td>215,248</td>
<td>207,178</td>
<td>329,374</td>
<td>352,082</td>
<td>187,999</td>
<td>352,077</td>
<td>284,127</td>
<td>168,147</td>
<td>40,662</td>
<td>239,746</td>
<td>179,247</td>
<td>113,117</td>
<td>111,286</td>
<td>109,584</td>
<td>3,324,936</td>
</tr>
<tr>
<td>Capital Costs ($ x 1000)</td>
<td>91,772</td>
<td>288,298</td>
<td>392,634</td>
<td>203,544</td>
<td>157,263</td>
<td>327,405</td>
<td>128,607</td>
<td>197,298</td>
<td>321,577</td>
<td>69,976</td>
<td>148,813</td>
<td>65,790</td>
<td>8,957</td>
<td>4,500</td>
<td>3,803</td>
<td>2,410,056</td>
</tr>
<tr>
<td>14% Revenue Based Taxes ($ x 1000)</td>
<td>43,082</td>
<td>113,461</td>
<td>124,807</td>
<td>124,897</td>
<td>124,798</td>
<td>125,273</td>
<td>125,624</td>
<td>125,478</td>
<td>124,359</td>
<td>125,303</td>
<td>125,559</td>
<td>99,482</td>
<td>58,695</td>
<td>27,345</td>
<td>20,364</td>
<td>1,488,468</td>
</tr>
<tr>
<td>Total Cash Outflow ($ x 1000)</td>
<td>203,956</td>
<td>617,007</td>
<td>724,619</td>
<td>657,815</td>
<td>634,144</td>
<td>640,677</td>
<td>606,308</td>
<td>606,903</td>
<td>614,082</td>
<td>601,720</td>
<td>514,119</td>
<td>344,519</td>
<td>180,769</td>
<td>143,131</td>
<td>133,691</td>
<td>7,223,460</td>
</tr>
<tr>
<td>Net Cash Flow ($ x 1000)</td>
<td>105,221</td>
<td>197,241</td>
<td>171,050</td>
<td>238,500</td>
<td>261,461</td>
<td>258,337</td>
<td>295,226</td>
<td>293,583</td>
<td>278,369</td>
<td>297,505</td>
<td>386,949</td>
<td>369,405</td>
<td>240,452</td>
<td>53,110</td>
<td>12,017</td>
<td>3,458,424</td>
</tr>
<tr>
<td>Cumulative Net Cash Flow ($ x 1000)</td>
<td>105,221</td>
<td>302,461</td>
<td>473,511</td>
<td>712,011</td>
<td>973,471</td>
<td>1,231,809</td>
<td>1,527,035</td>
<td>1,820,618</td>
<td>2,098,987</td>
<td>2,396,492</td>
<td>2,783,441</td>
<td>3,152,847</td>
<td>3,393,298</td>
<td>3,446,408</td>
<td>3,458,424</td>
<td></td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Discount Rate</th>
<th>Net Present Value ($ x 1000)</th>
</tr>
</thead>
<tbody>
<tr>
<td>0%</td>
<td>3,458</td>
</tr>
<tr>
<td>5%</td>
<td>2,383</td>
</tr>
<tr>
<td>8%</td>
<td>1,949</td>
</tr>
<tr>
<td>10%</td>
<td>1,720</td>
</tr>
</tbody>
</table>
22.2 Taxation and Royalties

The Restated Investment Agreement establishes effective January 1, 2008 (and continuing until the termination of the Restated Concession Agreement) a comprehensive tax regime for the Kumtor Project. Except for the payments set out below, the Kumtor Project is exempt from all other present and future taxes.

Except as expressly provided otherwise in the Restated Investment Agreement, the rates, amounts and other terms of any taxes or other payments are not subject to any future change in legislation or treaty provisions which would be more burdensome to the Kumtor Project or Centerra. The Kumtor Project and Centerra are entitled to benefit from any generally applicable future change in legislation or treaty provisions with respect to taxes or other payments payable under sections (2), (7), (8), (10) and (11) below which is beneficial to any of them. To the extent any such rates are capped by the provisions of sections (2), (7), (8), (10) and (11) are decreased due to a change in legislation, they can be increased by a future change in legislation, provided that any such increased rates from time to time shall not exceed the rates in effect on April 24, 2009.

The taxes provided for in the Restated Investment Agreement are as follows:

1. A tax on gross revenue of 13%, payable monthly (the “Gross Proceeds Tax”);

2. A contribution of 1% of gross revenue to the Issyk-Kul Oblast Development Fund (the “Issyk-Kul Contribution”). Together, these taxes constitute the 14% Revenue Based Taxes in Table 30;

3. Customs administration fees at generally applicable rates, which are not to exceed those rates in effect on April 24, 2009;

4. An annual payment of 4% of gross revenue against which all capital and exploration expenditures in the Kyrgyz Republic are fully credited, with expenditures not required for credit in any particular year carried forward for credit in future years;

5. An environmental pollution charge of $310,000 per year;

6. A land-use and access fee of $1,250,000 per quarter, against which the Gross Proceeds Tax and Issyk-Kul Contribution are credited in full;

7. Sales tax at generally applicable rates on goods and services purchased in relation to the Kumtor Project;

8. Value added tax at generally applicable rates on goods and services purchased by KGC and KOC, except for goods and services imported in relation to the Kumtor Project;
9. Generally applicable fees for licenses, registrations, travel visas and other fees for
discrete government services, provided that such fees shall not exceed those in effect
on April 24, 2009;

10. Payroll deductions for all employees subject to Kyrgyz income tax and contributions
to the Social Fund of the Kyrgyz Republic in respect of employees who are Kyrgyz
citizens, in each case at generally applicable rates; and

11. Excise taxes at generally applicable rates except on goods imported in relation to the
Kumtor Project.

The Kumtor Project is exempt from certain other obligations, including the following:

1. KGC and KOC are exempt from all withholding obligations with respect to payments
to third parties, but such third parties are not exempt from the relevant taxes to which
the withholding would otherwise have related, subject to the benefits provided to such
third parties in any applicable international treaties;

2. Centerra and its subsidiaries (including KGC and KOC) are exempt from paying taxes
with respect to intra-group transactions, including for services, dividends, interest and
other distributions or transactions; and

3. The Kumtor Project is exempt from paying customs duties in relation to goods
imported.

Effective June 6, 2009, the management fee payable to Kyrgyzaltyn is fixed at $1 per
ounce, inclusive of any taxes.

22.3 Sensitivity Analysis

Table 30 provides cash flow forecasts for the Kumtor Project from October 1, 2012 to
2026 based on the current LOM plan and a gold price of $1 350 per ounce. Table 31 shows the
sensitivity of the project NPV to gold prices from $1 150 to $2000, discount rates of 0%, 5%, 8%
and 10% and sensitivities to three other variables at the base-case gold price and a 8% discount
rate.
Table 31 Sensitivities of Mine Net Cash Flow
Millions of dollars

<table>
<thead>
<tr>
<th>Gold Price ($/ounce)</th>
<th>Discount Rate</th>
<th>Sensitivity to Gold Price at 0%, 5%, 8% and 10% Discount Rates</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>0%</td>
<td>5%</td>
</tr>
<tr>
<td>$ 1 150</td>
<td>2 096</td>
<td>1 413</td>
</tr>
<tr>
<td>$ 1 350</td>
<td>3 458</td>
<td>2 383</td>
</tr>
<tr>
<td>$ 1 500</td>
<td>4 479</td>
<td>3 111</td>
</tr>
<tr>
<td>$ 1 700</td>
<td>5 841</td>
<td>4 082</td>
</tr>
<tr>
<td>$ 2 000</td>
<td>7 884</td>
<td>5 537</td>
</tr>
</tbody>
</table>

Sensitivities to other Variables at $1 350 per ounce and 8% Discount Rate

<table>
<thead>
<tr>
<th>Variable</th>
<th>Operating Costs</th>
<th>Capital Costs</th>
<th>Gold Grade</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>+10%</td>
<td>-10%</td>
<td></td>
</tr>
<tr>
<td></td>
<td>1 648</td>
<td>2 251</td>
<td>2 584</td>
</tr>
<tr>
<td>Base Case</td>
<td>1 949</td>
<td>1 999</td>
<td>1 949</td>
</tr>
</tbody>
</table>

A gold price of $843 per ounce is required to achieve neutral net cash flow over the presently foreseen life of the mine while meeting all anticipated requirements for operations, capital expenditures, and taxes, but excluding exploration expenditures. At gold prices higher than $834 per ounce, the Kumtor Project has the potential to generate substantial positive cash flow.

Other than the gold price, the only parameter that would have the possibility of having a significant impact on mine cash flow would be a decrease in mill head grade, with a 10% gold grade reduction diminishing cumulative cash flow over the period of the LOM plan by about $634 million at a constant gold price of $1350 per ounce.


23 ADJACENT PROPERTIES

The authors are only aware of one mineral exploration company that has been active in the general Kumtor area. Kentor Gold Limited of Australia (Kenton) had acquired two exploration concessions in 2003. The very large Bashkol concession had originally adjoined the Kumtor Concession Area to the northeast and to the southeast, but appears to now have been substantially reduced in size. The following is quoted from the Kentor web site as it appeared on October 2, 2012:

"Work at Bashkol over the past 2 years has resulted in the identification of substantial areas of gold and copper mineralisation in the order of 1g/t in altered Proterozoic granite at the Bekbulaktor prospect. At Bekbulaktor the mineralisation has been traced on surface in apparent zones over an area of approximately 3 square kilometres, and is still open in three directions. The most substantial zone identified to date is 1,500 metres long and 500 metres. The Bashkol licence area is in the north east of Kyrgyz Republic in the Tien Shan Gold Belt adjacent to a suture line which separates two tectonic units – the Northern Tien Shan and the Middle Tien Shan. This is a similar structural position to the Kumtor deposit which sits adjacent to the same suture line (the Nikolaev Line) some 75km along strike to the south-west."

This description appears to reflect the property status as of year-end 2011, since a work program for 2012 is also announced which was to consist of road construction, follow-up geophysical and geochemical surveys, and a 3 700-metre drill program.

The smaller Akbel concession was located to the southwest of the Concession Area along strike from the Sarytor and Bordoo gold occurrences (Figure 6), some ten to fifteen kilometres to the southwest of the Central Pit (Kenton Gold Ltd., 2005 to 2008). This concession is no longer displayed on the Kentor web site as of October 2, 2012 and appears to have been abandoned.

According to their website, Kentor has conducted surface exploration including mobile metal ion (MMI) soil gold sampling and induced polarization (IP) surveys and diamond drilling on the Akbel concession. A total of nine surface holes were completed between 2005 and 2007. No strongly altered rocks or gold-mineralized zones have apparently been encountered that would indicate the discovery of a mineralized zone. Additional holes were drilled in 2007 and 2008, but no further drilling results have been announced on the Kentor website. Kentor’s 2009 first quarter report states that the company had been in discussions with various parties on the sale of the Akbel concession, but no sale is known to have occurred.

The responsible author is unable to verify the information in respect of the Bashkol and Akbel concessions summarized in this section.
24 OTHER RELEVANT DATA AND INFORMATION

24.1 Mine and Concession Exploration

This section summarizes the remaining exploration targets that are being investigated as potential opportunities to provide mill feed to the existing operation in the future. The targets are grouped into sections describing the investigation of mineral resources potentially mineable by underground methods in the Central Pit area, exploration results and remaining possibilities to add to the open-pit resources in the Central Pit area and along strike from the Central Deposit, and surface exploration activities in the general Kumtor Project area. The target areas discussed in this section are shown on Figure 6 for the entire Concession Area and on Figure 30 for the Central and Southwest Deposits.

24.1.1 Exploration of SB and Stockwork Zone Extensions

24.1.1.1 The Question of Underground Mining

Substantial efforts have been made since 2006 to develop, and ultimately mine by underground methods, those high-grade portions of the SB and Stockwork Zones that fell outside of the ultimate pits of earlier Central Pit mine designs. Separate declines have been completed to access the two zones, but progress of the development has been slow due to very difficult ground conditions. Total development has amounted to approximately 3,000 metres. The two declines have recently been connected for increased ventilation, but trial mining has not yet commenced. Details of this program have been reported in the last technical report (Redmond et al., 2011). The decline in the SB section of the pit has allowed underground definition drilling in the southern part of the SB Zone, which drilling has contributed to the increase in SB Zone mineral resources described in Section 15.13.2. However, much of the underground infrastructure established will be consumed by the new final pit design for the Central Deposit.

The evaluation of the technical and economic feasibility of underground mining will have to concentrate on the SB Zone mineralization with its larger tonnage and higher gold grades (Table 22). A three-stage program is necessary which starts with the in-fill and extension drilling program described below. The second step will be a program of underground trial mining for which access from the SB part of the Central pit will not be possible until early 2018. Assuming that two years of development and trial mining will be required, the third step, the preparation of a feasibility study, could be completed sometime in the first half of 2020. This program requires that the advancing Davidov Glacier is prevented from interfering. Mineral reserves for underground mining can only be estimated after completion of this program.
24.1.1.2. **Planned Drilling Programs**

The additional inferred mineral resources of the SB Zone below the previous KS-12 pit were estimated at 4.0 million tonnes with an average gold grade of 13.6 g/t (Centerra Annual Report for 2011). The new KS-13 final pit will mine out approximately one-half of this tonnage at a higher-than-average gold grade. The resources remaining below the KS-13 pit are therefore of lower grade and also have poorer definition because of a wider drill-hole pattern. To prepare for the eventual evaluation of underground mining, additional drilling is planned from surface and underground to more fully define the SB Zone below the new KS-13 pit bottom to firm up these resources and to test for additional high-grade mineralization that may be mineable by underground methods.

Drilling in the SB Zone in 2012 (through September) totalled 27 219 meters in 67 holes. An additional 25 000 meters of drilling is planned for the remainder of 2012 and for 2013. While the KS-13 ultimate pit will consume the southern parts of the existing underground openings, including the Portal No. 1 in the Davidov Valley, access to the underground workings will be possible until 2017 from the existing Portal No. 2 located within the central part of the Central Pit.

There is no additional drilling planned for those parts of the Stockwork Zone that are below the KS13 design pit.

24.1.2 **Exploration in and around the Central Pit**

24.1.2.1. **Saddle Zone**

The Saddle Zone is located midway between the high-grade mineralization of the Stockwork and SB Zones. Drilling in the Saddle Zone in 2008 to 2011 has identified narrow but consistent zones of high-grade mineralization (e.g. 11.9 g/t over 5.7 metres in drill-hole D1567B) normally located along the hangingwall contact of a broader zone of lower grade mineralization up to 200 metres wide. Further exploration drilling will target these higher grade zones in proximity to the adjacent SB Zone approximately 300 to 400 metres below the KS13 pit design.

24.1.2.2. **Northend Target**

Drilling in 2010 in the Northend target, located at elevation 3 600 metres approximately one kilometre to the northeast of the Stockwork Zone (Figure 30) followed initial results from two drill holes drilled in 2008 and 2009 which had returned gold values of 8.6 g/t Au over 13.9 metres and 9.1 g/t Au over 11.0 metres, respectively). The follow-up drilling did not intersect any extension to this mineralisation. The target has been downgraded due to an overall lack of success as only two out of the twenty three holes drilled since 2005 at a spacing of 80 to 250 metres have intersected any significant mineralization. No further drilling is planned.
24.1.3 Southwest and Sarytor Deposits

The Southwest and Sarytor deposits are satellite open pits located to the southwest of the Kumtor Central Pit and separated from it by the Davidov glacier and valley. At Southwest and Sarytor, the Kumtor mineralization dips gently to the south and becomes quickly concealed beneath steep topography south of the existing open pits. Exploration activities in 2011 and 2012 have included drill tests of anomalies north of the Southwest Deposit and drilling down-dip of the Sarytor deposit has identified local high-grade mineralization (e.g. 7.1 g/t gold over 28.3 metres in drill hole SR-12-202). Drilling north of the Southwest pit has identified narrow intervals of oxide gold mineralization in rocks in the footwall of the Southwest deposit. Additional drilling is planned in both areas for 2013.

24.1.4 Other Exploration Targets on the Concession

This section describes several exploration targets along strike in both the northeast and southwest directions from the Central Deposit. The areas to the southwest of the Central Pit (e.g., Southwest, Sarytor) have the disadvantage of the controlling structures dipping at shallow to intermediate angles to the southeast, with the surface rising in the same direction. Access to the deeper parts of gold mineralization in this area by open-pit mining is therefore limited by the adverse topographical situation. However, owing to the proximity of these areas to the existing mine infrastructure, capital costs to develop a satellite deposit would be low.

The Concession Area and the exploration targets discussed in this section are shown on Figure 6. The targets are described below.

24.1.4.1. Northeast Deposit

To the northeast of the Central Pit is the Northeast Area where surface trenching, diamond drilling and underground sampling were undertaken in the 1980s. New exploration targets were generated in this area following the addition of the Northeast Deposit data into the exploration database and the re-interpretation of the geology and the earlier exploration results. Exploration work in this area has since included, surface trenching, soil sampling, induced polarization (IP) and magnetometer surveys and 65 diamond drill holes completed to the end of 2010.
In 2011, KOC completed a further 5,464 metres in 27 holes at the Northeast Deposit and an initial mineral resource estimate was published by Centerra in early 2012 amounting to 4.1 million tonnes of inferred resources at an average gold grade of 2.1 g/t from this deposit. In addition, the 2011 drilling identified an area of higher gold grades (e.g. 23.4 g/t gold over 9 metres in drill hole DN1566) near the eastern limits of the area covered by the historical exploration. Additional drilling is planned in 2012 and 2013 to expand the knowledge of this deposit further northeast of the current resource.

24.1.4.2. Petrov Prospects

The Petrov prospects are located on the southern and northern shores of Petrov Lake, in the northeast corner of the Concession Area. Soil geochemical and geophysical surveys completed in 2007 identified coincident multi-element geochemical and geophysical anomalies on the south side of Petrov Lake. The anomalies lie on the interpreted north-eastern extension of the Kumtor structure that hosts the mineralization in the Central and Northeast Deposits.

Four diamond drill holes were completed in 2009 to test the anomalies to the south of Petrov Lake. The holes intersected weak alteration on structures but no significant mineralization.

Soil geochemical and geophysical surveys and prospecting surveys completed in 2010 on the northern shore of Petrov Lake identified coincident geochemical and geophysical anomalies on the interpreted north-eastern extension of the KFZ which is normally associated with the mineralized structures at Central Pit, Southwest Pit and Sarytor areas. Three additional holes were completed in 2011, and the results were negative. No additional work is planned.

24.1.4.3. Muzdusuu Prospect

The Muzdusuu target is located to the west of the Central Pit in Carboniferous limestone and sandstone and in conglomerate of Neogene age in the footwall of the Kumtor Fault. Previous Soviet-era drilling and KOC drilling in 1999 intersected low-grade mineralization over intervals ranging from one to five metres.

Geological mapping, soil and rock geochemical sampling and geophysical surveys completed in 2009 identified strong surface gold and multi-element anomalies over a limestone. Six diamond drill holes completed in 2009 and 2010 to test the geochemical anomalies and the stratigraphy in the footwall of the KFZ returned a best intersection of 6.4 g/t Au over 2.9 metres on the contact between hanging wall limestone and footwall Neogene-Paleogene sediments. The mineralization is narrow and discontinuous, and no additional work is planned.
24.1.4.4. **Bordoo Prospect**

Further to the southwest of Sarytor is the Bordoo area, where targets identified by geophysical surveys conducted during the Soviet period were tested in 1999 by surface sampling. The best results of surface chip sampling were 20.3 g/t gold over 5 metres and 3.6 g/t gold over 20 metres. Some 850 metres of trenching and outcrop sampling conducted in this area in 2002 has given initial encouraging results, such as 1.0 g/t gold over 8.0 metres in trench T-BR 2 and 2.4 g/t gold over 5.0 metres in trench T-BR 13. Numerous mineralized outcrop and road cut samples have outlined an 800-metre long by 50 to 70-metre wide zone of generally low-grade gold mineralization, with values from 0.5 to 1.0 g/t gold over 5.0 metres (chip sample) and 8.42 g/t gold over 2.0 metres (chip sample).

An induced polarization (IP) survey over the Sarytor and Bordoo areas, extending the historic IP coverage to the southwest, indicates the KFZ to continue under the moraines covering the northern part of the Sarytor and Bordoo areas. A geo-electrical response similar to that found at Sarytor has also been detected in the Bordoo area, extending the possible structural target area approximately three kilometres along strike to the southwest. The interpreted zone is covered by the moraine. Wide-spaced reconnaissance drilling conducted in late 2007 and 2008 to test the interpreted continuation of the KFZ did not intersect any significant mineralization. Three additional holes drilled in 2011 to test geophysical anomalies failed to intersect altered or mineralized rocks. No additional work is currently planned.

24.1.4.5. **Akbel Prospect**

The Akbel area is situated along strike to the southwest from the Bordoo area. Reconnaissance exploration work which included geophysical surveys, geologic mapping and surface sampling indicated the presence of some gold mineralization, with the best result being an assay of 3.0 g/t gold from a grab sample. Wide-spaced reconnaissance drilling conducted in late 2007 and 2008, (three holes, total 450 metres) to test the interpreted continuation of the KFZ did not intersect any significant mineralization. One additional hole drilled to 486 metres depth also failed to encounter mineralization within a monotonous sequence of black shale and tillite. No additional work is planned at Akbel.

It is of note that the activities of Kentor Gold Limited as described in Section 23 are contiguous with the Akbel area. 2005-2008 activities by Kentor included the drilling of several core holes on a geochemical anomaly some 13 kilometres to the southwest of the processing plant. No significant drilling results have ever been announced by Kentor for the Akbel area.

24.2 **Regional Exploration**

The Kumtor Concession Area is bounded to the northeast by the Sarychat-Ertash Nature Reserve, an area of protected flora and fauna covering some 1 300 km². To the south and west of the mining concession are the Karasay, Koendy and Choloktor license applications filed by Kumtor Gold Company in 2009 and 2010. (Figure 5) Exploration licenses are issued for an initial
two-year period and are renewable for up additional two-year periods. At any time during the 10 years of their validity, they may be converted to a mining licence, valid for 20 years, which can be further renewed. There is no fixed expenditure commitment for an exploration license, but exploration results and future programs by the license holder need to be presented to the Government annually.

The Karasay and Koendy licenses (Figure 5) expired in late 2012 and 2011, respectively, and are the subject of an environmental review being undertaken by the State Commission formed in 2012 (see Section 25.1.2). The Choloktor license application to the southwest of the Kumtor mining concession was filed in 2010 and also remains under review by the Licensing Commission of the Agency for Subsoil and Natural Resources. The review of the exploration licences is now part of the scope of a state commission examining the technical, financial and environmental aspects of the Kumtor operation.

A brief summary of work completed on the two licenses until 2011 and 2012 is presented below.

24.2.1 Karasay Exploration License

KGC was granted a 139.4 square kilometres exploration license on October 6, 2009, over the Karasay prospect, 20 kilometres south of the Central Pit. Soviet-era exploration identified alteration and geochemical anomalies in a geological setting similar to the Central Deposit. Geological mapping and geochemical surveying completed in 2010 identified a number of significant anomalies and zones of mineralization.

Further mapping, sampling, and geophysical surveys identified a three-kilometre trend of stream and soil anomalism in black shale, sandstone and rhyolite analogous to the Kumtor host-rock package. Several additional targets were developed in volcanic rocks and limestone at lower elevations on the license. Exploration work ceased in late 2011 when the license expired. KGC awaits the outcome of the licensing agency’s review and decision on its application to renew the license.

24.2.2 Koendy Exploration License

KGC was granted a 133.6 square kilometres exploration license on June 14, 2010, over the Koendy prospect, 20 kilometres southeast of the Central Deposit Soviet era exploration identified alteration and geochemical anomalies in a geological setting similar to the Central Deposit. In 2010 and 2011, exploration teams completed geologic mapping, stream-sediment and soil sampling, rock-chip sampling, ground magnetics and limited dipole-dipole IP surveys to delineate several targets on the license. Late in 2011, an area of soil anomalism in sheared black shale and sandstone was tested with a series of trenches with negative results. A number of targets remain on the license, and further exploration is pending a decision on KGC’s application for renewal of the Koendy license.
24.2.3 Regional Reconnaissance

Centerra believe that there is significant exploration potential in the district surrounding the Kumtor Project, and therefore commenced a regional assessment program in 2010 including evaluation of regional geological, geochemical and geophysical datasets. KGC submitted two license applications covering regional targets in 2010 and a third application in early 2012. The licensing commission of the Agency for Subsoil and Natural resources has taken no action on these applications.

24.3 2013 Exploration Budget

The description in the previous sections of the exploration opportunities at and around the Kumtor Project provides the justification for additional surface exploration required to fully evaluate the various targets. KOC currently has nine surface diamond drills and four underground diamond drills that, together, can complete up to 50 000 metres of drilling per year. In 2012, five surface rigs and four underground rigs operated from platforms within the pit and from drill bays in the underground.

For the remainder of 2012 and for the year 2013, KOC and Centerra have formulated the exploration programs described in the previous sections and this budget was approved by the Centerra Board of Directors in December 2012. The budgeted expenditures are summarized in Table 32.

Table 32 Planned Exploration Expenditures for 2012 and 2013

<table>
<thead>
<tr>
<th>Location</th>
<th>Number of Holes</th>
<th>Metres</th>
<th>Total Costs (‘000$)</th>
</tr>
</thead>
<tbody>
<tr>
<td>SB Zone Below KS 13 Pit</td>
<td>30</td>
<td>25 000</td>
<td>9 000</td>
</tr>
<tr>
<td>Northeast</td>
<td>10</td>
<td>3 200</td>
<td>1 200</td>
</tr>
<tr>
<td>Southwest &amp; Sarytor Deposits</td>
<td>13</td>
<td>6 000</td>
<td>2 200</td>
</tr>
<tr>
<td>Regional Reconnaissance</td>
<td>20</td>
<td>6 000</td>
<td>3 500</td>
</tr>
<tr>
<td>Capital Items</td>
<td></td>
<td></td>
<td>100</td>
</tr>
<tr>
<td><strong>Total</strong></td>
<td><strong>73</strong></td>
<td><strong>40 200</strong></td>
<td><strong>16 000</strong></td>
</tr>
</tbody>
</table>

The exact number of holes drilled on each of the targets in 2012 and 2013 will depend on a combination of seasonal access, mine operational and target priority considerations, which will change as new results are being obtained.
25 RISK FACTORS

This section addresses risk factors that the authors believe could have a material effect on the Kumtor Project. Some of these risks have already been discussed in previous sections. The risks are both of a technical and non-technical nature. Only risks specific to the Kumtor operation, location or political situation are addressed in this section, while risks affecting the gold mining industry generally are not included.

25.1 Technical Risks

25.1.1 Reserve Estimation Risks

The responsible authors are of the opinion that there is little significant risk with respect to the current estimate of mineral resources or mineral reserves. This is based on the good reconciliation between the KS-13 and SW-2 block models and a substantial tonnage of past production as discussed in Section 15.11.

25.1.2 Geotechnical Risks

Geotechnical risks arise with respect to the stability of the pit walls and with respect to glacier ice movement into the current and future pit. The risks which relate to the two items are summarized below:

The Kumtor open pit is a large man-made excavation that is planned to be enlarged substantially in accordance with the KS-13 LOM plan. The final pit walls will have a vertical extent of up to 700 metres in SB Zone part of the Central Pit (up to 980 metres if the natural slope above is considered) and up to 800 metres in the Stockwork zone area. In general, there is a higher risk associated with higher rock walls.

The pit-wall slopes in rock for the KS-13 pit design follow the recommendations by Golder (2012a), who have provided long-term geotechnical advice to Kumtor. Following two failures of the northeast high wall in 2002 and 2006, a comprehensive program of structural mapping, geotechnical drilling and modelling has resulted in a reduction of the design pit walls to generally between 30° and 35°. However, the slope angles recommended by Golder are still “preliminary” until additional geotechnical drilling can be completed into the walls into which particularly the southern part of the Central Pit will expand. The design slope angles assume that the pit walls are depressurized, and drilling to accomplish depressurization is part of the mine plan.

If the slope angles used for the design of the KS-13 pit need significant flattening or if the pit walls prove to be unstable at the slope angles assumed by KS-13 pit design, there would be a substantial negative impact on the LOM plan and on the current open-pit mineral reserves.

From the information presented in Section 15.3.2, it is obvious that the rate at which Davidov Glacier ice mining has to be accomplished during the operation of the southern part of the Central Pit is uncertain. The volumes of ice mining and the additional mining equipment
required to accomplish this are therefore subject to upward revision, possibly in a substantial way. Should ice mining not keep up with the forward ice movement, interruptions to the LOM plan with respect to mining of the high-grade SB Zone would occur, with negative implications for the mine plan and the project cash flow.

25.1.3 Tailings Facility, Petrov Lake and Waste Dumps

The new LOM plan requires considerable deposition of the waste in large dumps located in the Davidov, Sarytor and Lysii valleys. The waste is placed on top of the moraine layer in both valleys. Based on the experience of the current waste dump in the Davidov valley, deformation expectations of the moraine layers have been incorporated into the waste-dump design. However, should the dumps become unstable, additional haulage distances for the waste may be required which could negatively impact the ore release schedule. Waste-dump placement is therefore considered to be one of the technical risks that can adversely impact the LOM plan and economic performance of the Kumtor operation.

A second geotechnical challenge facing the future Kumtor operation is the interaction of the Central Pit with the surrounding glaciers. The main tasks are to complete the unloading of the historical waste dumps and underlying ice, and the additional removal of a sufficient amount of glacier ice to prevent the advancing Davidov Glacier from entering the Central Pit until 2018, when this part of the pit will be mined out. While the LOM plan has a contingency for additional ice tonnages to be mined, any substantial additional ice mining would negatively impact the mining of ore and the project cash flow; a loss of reserves is not anticipated. However, should Kumtor be prevented from continuing its practice of mining ice, the entire KS-13 mineral reserves and LOM plan would be at risk, leading to an early closure of the operations.

To accommodate the 33 million cubic metres of future tailings that will be produced as a result of the enlarged Central Pit but that cannot be stored in the existing facility as currently approvals, three storage options have been evaluated. Raising of the existing tailings dam by seven metres is the preferred solution that is safe to do, has substantial ecological and economic advantages and should not present any permitting issues on technical grounds. The second option is outside of the Kumtor Concession Area and therefore not practical. The third option, while located inside the concession, requires a large amount of earthworks. If permitting of Option 1 cannot be obtained, additional capital expenditures beyond those in the current capital budget for the new LOM plan would have to be incurred.

Petrov Lake, which is fed by the receding and melting Petrov Glacier and which is therefore increasing in size, is confined by a moraine which is currently stable due to its largely frozen state. It is forecast that the moraine will thaw out prior to 2050 due to continued global warming, and a sudden and potentially damaging glacial lake outflow will occur. This event has the potential to erode a section of the Kumtor tailings management facility and to damage other downstream installations. Any erosion of the tailings dam would have to be considered a serious threat. While the risk of this outflow occurring in the next ten years is considered low, this is a future event that needs to be considered for mine closure. An early warning system is currently being installed to safeguard people working in the path of a potential outflow, and remedial measures to armour and protect the tailings facility are under evaluation. The construction of a
permanent spillway to lower the level of Petrov Lake by three metres is planned to commence in 2014.

25.2 Non-Technical Risks

The Kumtor operation is exposed to a number of non-technical risks which are related to the country of operation or derive from other external factors largely or entirely beyond the control of Centerra or KOC. The segment below covers certain key risks.

25.2.1 Political Risks

The Kyrgyz Republic has experienced political difficulties in recent years including two revolutions (2005 and 2010) that have resulted in the ouster of the then incumbent president; the reconstitution of parliament; and the imprisonment of various political party members. The Kyrgyz economy continues to be impacted by a volatile political environment and lack of economic development.

The Kumtor operation plays an important part in the Kyrgyz economy with a share of 11.7% in the Kyrgyz GDP in 2011. In addition, the Kyrgyz state, through Kyrgyzaltyn, holds nearly one-third of the issued shares of Centerra. The agreement terms between the Kyrgyz Republic and the Kumtor Project have been re-negotiated twice in the past, as discussed in Sections 1.2, 2.1 and 6.1. The initial agreements with Cameco dating from 1994 were re-negotiated in 2003 and then re-negotiated again in 2009. The direct benefits to the Kyrgyz state were increased in each case.

As outlined in Section 2.1, Kumtor has recently been the object of a Parliamentary Commission and the State Commission investigations with the objective of reviewing and assessing all past and present agreements relating to the Kumtor operations; the finances and operational processes of the company, as well as its environmental, health and safety management systems. The State Commission was formed in July 2012 to “assess the environmental, industrial and social damage” caused by the Kumtor project, and to provide a “legal examination of agreements made on the Kumtor Project in terms of protection of the state interests.” (Government Decree as translated). There have also been four recent claims by the Kyrgyz State Inspectorate Office for Environmental and Technical Safety for a total payment of approximately $152 million relating to alleged environmental damages at the Kumtor Project, mainly with respect to the existing waste dumps.

Although both the Prime Minister and the President have indicated through the media that it is not an option, there continues to be voices within the Kyrgyz Republic demanding nationalization of the Kumtor Project. It is conceivable that the recommendations of the State Commission report, predicted to be released in January 2013, will include a recommendation to re-define the terms between the state of Kyrgyzstan and the Kumtor Project. The economic assumptions and projections in this report are based on the existing agreements concluded in 2009. Any changes would alter the economic outcomes of the new LOM plan as presented in this report.
25.2.2 Supply Chain

Kumtor is located in a remote location and long lead times are required for equipment and supplies which partly originate in other countries. Supply-chain risks are associated with the flow of materials, supplies and services to the mine site, as well as timely delivery of large mining equipment (see Section 21.2). Any significant delay in the delivery of equipment and/or materials due to border or customs clearance issues, road blockades or the failure of a key supplier to meet a delivery schedule for critical equipment may negatively affect the timely execution of the LOM plan.
26 INTERPRETATION AND CONCLUSIONS

This review of the Kumtor Project and its September 30, 2012 mineral reserve estimate has resulted in the following main conclusions:

1. The open-pit mineral reserves at Kumtor have continued to expand at a significant rate since the discovery of the SB Zone in 2005. The total known Central Deposit (past production plus mineral reserves) has grown from 72 million tonnes with an average gold grade of 4.0 g/t at the end of 2004 to 164 million tonnes with an average gold grade of 3.6 g/t as of September 30, 2012;

2. As a result, the expected conclusion of open-pit mining has been extended by thirteen years from 2010 as anticipated in 2004 to 2023 according to the current LOM Plan. In addition to the discovery and recurring success of expanding the SB Zone with additional drilling, this increase is the result of continued higher gold prices used for mineral reserve estimation that have offset increasing operating costs;

3. The mineral reserves (tonnes and contained ounces) as of September 30, 2012 have increased by more than 50% compared to the estimate at the end of 2011, with the average gold grade remaining constant at 3.3 g/t. Milling operations are now expected to continue until 2026;

4. Mineral reserve block models for the Kumtor Project continue to evolve with greater production experience and additional drilling information. A review of the production reconciliation to the latest version of the mineral reserve models proves that the models are accurate estimators of the mineral reserves (tonnes and grade) of the gold mineralization at Kumtor. While the Kumtor Project mineral reserve model, based on wide spaced exploration data, does not allow a perfect correlation with actual production tonnages and grades experienced over short time intervals, the variances for annual production tonnages observed over the last nine years are well within industry standards. The mineral reserve estimation risk for the tonnage and grade predicted by the current LOM plan is low;

5. While mineral reserve increases over the last four years have been material, opportunities for further reserve expansions are now limited by the unfavourable topography in the southern part of the new ultimate Central Pit. Centerra and KOC will need to focus their future exploration efforts on the investigation of new, near-surface areas away from the existing pits, on deeper high-grade mineral resources that could be mined from underground, or on new exploration licenses further afield in order to continue to materially add to the Kumtor mineral reserve base;

6. The Kumtor operation will continue to produce ore at a high strip ratio for most of its projected mine life, with the total annual tonnage mined in the range of 170 to 193 million tonnes for the period of 2013 to 2021 with an average waste-to-ore ratio of 19.2. To accomplish this mining rate, the program of significant capital expenditures for additional primary and auxiliary mining equipment of the past years will continue until 2014. Re-
location of surface installations due to the expansion of the Central Pit and because of the encroachment by the Davidov Valley waste dump requires additional capital expenditures until 2015. Other capital projects include the continuing expansion of the tailings facility and the expansion of the mill capacity in the years 2014 and 2015. Centerra has provided for the required mine capital expenditures in its cash-flow forecasts;

7. Gold production from the Central Pit has been negatively impacted on several occasions from 2002 to 2012 by geotechnical issues related to the poor quality of the host rocks resulting from the intensive and complex structural deformation in the area, and the gradual movement of sections of the historical waste dumps that had originally been placed on parts of the adjacent Davidov glacier. While Centerra’s understanding and resulting remedial plans for these issues has progressed significantly, geotechnical issues remain the most significant risk to achieving the gold production and associated cash flow as outlined in the LOM plan;

8. There are no indications of geotechnical issues for the smaller Sarytor and Southwest pits that are planned to be mined in the years 2018 to 2023;

9. Since 2007, Centerra has undertaken a sizeable program of underground development and drilling to evaluate the potential for a possible underground mining operation at Kumtor that would exploit high-grade ore shoots below the final pit bottom to augment the open-pit mining. As of September 30, 2012, approximately $190 million had been spent on the underground project which included costs related to the establishment of portal facilities, underground mining equipment and the completion of approximately 3,000 meters of ramp development in two declines. No underground test mining has yet been completed in what are very difficult ground conditions. Without the planned additional drilling and a comprehensive test mining program, followed by a feasibility study, conversion of these resources into reserves will not be possible.

10. Kumtor has been the object of a State Commission formed in July 2012 with the intention to “assess the environmental, industrial and social damage” caused by the Kumtor Project and to provide a “legal examination of agreements made on the Kumtor Project in terms of protection of the state interests.” It is possible that the State Commission final report expected to be issued in January 2013 will recommend changes to the existing agreements that could put additional economic constraints on the operation. These potential constraints are currently unknown and are not reflected in the economic projections made in this report which is based on the existing agreements concluded in 2009.
27 RECOMMENDATIONS

The authors of this technical report make the following recommendations:

1. The authors endorse the plans by Kumtor to implement additional geotechnical drilling and studies to ascertain that the pit-slope assumptions used for the KS-13 Central Pit are sufficient to prevent any large slope failures in general, and to ensure that there will not be any impact by the expanding pit on the processing plant, which could have very severe consequences;

2. The program of geotechnical analysis of the creeping area of the historical waste dumps deposited on glacier ice should be continued with the goal of further improvements in the understanding of the mechanisms for the movement. These investigations should continue to include the movement of the existing and accumulating waste dumps;

3. Monitoring of the hydraulic pressure of all of the Central Pit walls and particularly of the high walls as recommended by the geotechnical consultants should be continued and completed, including the drilling of depressurization wells where necessary;

4. With the physical limits of open-pit mining having been reached in the southern part of the Central Pit with the new LOM plan based on the KS-13 block model, Centerra should complete studies to determine the economic conditions required for the possible conversion of all or some of the additional mineral resources mineable by open pit into mineral reserves. These investigations would include the determination of incremental strip ratios for the additional resources, capital requirements (the cost of additional waste mining as well as of any additional equipment purchases), the interplay with the current LOM plan and the gold price that would support such conversion. Based on the outcome of such studies, the reporting of additional resources mineable by open pit may change in future;

5. The authors endorse the recommendations made by Bloom (2012) for a modest capital program to improve the space and the equipment of the mine assay laboratory including the sample preparation section;

6. Given the success rate of previous exploration programs at the Kumtor Project and given the remaining exploration possibilities, the authors support the ongoing exploration efforts with a financial commitment of $16 million for 2012/2013 for surface and underground drilling exploration. Further exploration expenditures will likely be required beyond 2013 on the additional new exploration licenses in subsequent years, but details of these, and their justification, are contingent on the results of the 2012/2013 program;

7. The future conversion of mineral resources mineable by underground methods into mineral reserves and subsequent inclusion in the Kumtor LOM plan should be made dependent on the successful completion of the additional drill programs and on a comprehensive underground test mining program that determines the technical and economic parameters of underground mining for a dedicated feasibility study.
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