SUKINDA CHROMITE MINE

(VOLUME-I)

MODIFICATION OF SCHEME OF MINING
(01.04.2011 TO 11.01.2013)

Submitted under Rule No. 10 of MCDR, 1988

&

PROGRESSIVE MINE CLOSURE PLAN
(01.04.2011 TO 11.01.2013)

Submitted under Rule No. 10 of MCDR, 1988

AND

MINING PLAN
(From 12.01.2013 onwards)

Submitted under Rule 24A of MCR, 1960

&

PROGRESSIVE MINE CLOSURE PLAN
(From 12.01.2013 onwards)

Submitted under Rule No. 23B of MCDR, 1988

Lease Area (Ha.)	Forest Area (Ha.)	Non- forest Area (Ha)	Expiry of Lease	Category of Mine
406	73.697	332.303	11.01.2013	"A" (Fully Mechanized)

PREPARED BY : G K GUIN
REGN. NO. RQP/ BBS / 044 / 2003 / A
TATA STEEL LIMITED

Modified Scheme of Mining	• Page 1 to 123
Mining Plan	• Page 124 to 364

This is to certify that:

A) The provisions of Mines Act, 1952 Rules and Regulations made there under have been observed in the preparation of the Modification of Scheme of Mining & Mining Plan of Sukinda Chromite Mine of M/s Tata Steel Limited over an area of 406 ha. in Jajpur district of Orissa and wherever specific permissions are required, the applicant will approach the DGMS.

B) The information furnished in this Modification of Scheme of Mining & Mining Plan of Sukinda Chromite Mine is true to the best of my knowledge.

C) Further standards prescribed by DGMS in respect of miner's health will be implemented.

Name	:	G K Guin
Designation	:	Head (Mine & Production Planning) Sukinda Chromite Mine TATA STEEL LIMITED
RQP	:	Regd. No. RQP/BBS/044/2003/A

Place : Sukinda
Date : 9th May' 2012

C E R T I F I C A T E

This is to certify that the provisions of MCDR 1988 have been observed in the preparation of the Modification of Scheme of Mining & Mining Plan of Sukinda Chromite Mine of M/s Tata Steel Limited in Jajpur district of Orissa and wherever specific permissions are required, the applicant will approach the Indian Bureau of Mines.

Name	:	G K Guin
Designation	:	Head (Mine & Production Planning) Sukinda Chromite Mine TATA STEEL LIMITED
RQP	:	Regd. No. RQP/BBS/044/2003/A

Place : Sukinda
Date : 9th May' 2012

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A INTRODUCTION:

Particulars of approval of Mining Plan / Scheme of Mining:

The Mining Plan for Sukinda Chromite Mine over an area of 406 ha. was submitted under Rule 22 of MCR, 1960 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3) / 97 – MCMM (C) /MP – 7 dated 12.12.97. This is valid till 11.01.2013 concurrent with lease validity period. The second Scheme of Mining for Sukinda Chromite Mine over an area of 406 ha. was submitted under Rule 12 of MCDR , 1988 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3)/2008-MCCM(CZ)/MS-17/97, Dated: 02/02/09 for the period from 01.04.2008 to 11.01.2013. Further a Modified Scheme of Mining & Progressive mine Closure Plan was submitted under Rule 10 of MCDR, 1988 and approved by Controller of Mines (CZ), Nagpur, vide letter no. 314(3)/2011-MCCM (CZ)/MS-08, Dated: 22/07/12for the period from 01.04.2011 to 11.01.2013.

Reasons for Modification of Scheme of Mining:

a) The current approved Scheme of Mining for the Sukinda Chromite Mine is valid till 11.01.2013, i.e. till the validity of the lease period. It is observed that the production from the mine during the past 3 years has been lower than that proposed in the approved Scheme of Mining due to less requirement of ore a down turn in the market place. Since there aren't enough indications for improvement in the prevailing market conditions, there is a need to modify the plans for the current opencast mining operations for the balance 2 years period under the approved Scheme of Mining.

b) As per the recommendation of the Central Government order for 2nd renewal of the lease, we have in the meanwhile duly completed the study for undertaking underground mining. Copy of the Central Government order for 2nd renewal of the lease is enclosed as Annexure-1. This is therefore going to be a brownfield project wherein the mine is proposed to be operated both by opencast & underground means in the near future and then convert to a totally underground operation in the long term. Considering the dwindling mineable reserves for opencast exploitation, lack of space for OB disposal and the long gestation period required for development of the underground mine, the company is keen to commence the underground mining operations at the earliest to sustain chrome ore production and meet its requirements for its plants and also comply to the conditions that may be required for renewal of the lease. The company has undertaken the necessary geological, geo-physical and feasibility studies for underground mining. The Feasibility study has been done with the assistance of SRK Consulting, South Africa and CIMFR, Dhanbad & Nagpur in order to establish the applicable methods of underground mining.

c) It is essential to start the underground mining at the earliest in order to sustain chrome ore production. This essentiality is not only to meet the existing beneficiation and ferro chrome capacities but to also later on cater to the further increased chrome ore requirement for the proposed expansion of existing ferro chrome plants as well as upcoming new ferro alloy units at Nayagarh and Gopalpur and stainless steel unit in the near future.

d) The need for a modification to the Scheme of Mining to incorporate underground mining is also in line with the commitments made as per the future road map given in the present Approved Scheme of Mining under Para 4 on Conceptual Mining Plan (Ref.: page 29).

e) The company has also in the meanwhile duly applied for prior Environmental Clearance to the Ministry of Environment & Forests (MoEF) for undertaking this change of technology to underground mining, expansion, change in project area and renewal of the lease. As per the Terms of Reference for this Environmental Clearance, the MoEF has also demanded for an approved Mining Plan for underground mining.

Accordingly, we are hereby submitting the modification to the Scheme of Mining along with Progressive Mine Closure Plan under Rule 10 of MCDR, 1988 to take care of the changes in opencast operations and the starting of the underground development of the mine during the current plan and lease period. Further, the Mining Plan to establish & complete the balance development of the underground mine, method of mining/ stoping & production thereof together with the proposal for continuing opencast operations for the next 5 year period beyond 12.01.2013, is also being submitted under Rule 22 & 24(A) of MCR, 1960, as a separate section in the document towards part-fulfillment of lease renewal application.

The Modification to the Scheme of Mining & the Mining Plan are being submitted as two separate sections under a single document in order to provide and establish proper linkages and clarity with respect to the future proposal for underground mining.

This modification to the Scheme of Mining is being proposed on the following aspects:

Sl No.	Chapter No.	Chapter Name	Changes Proposed
2.	Chapter-2.1	Name & Address	Change in name and address of the applicant
3.	Chapter-2.4	Area & Expiry of Lease	Details of area and justification been incorporated.
4.	Chapter-3	Reserve	Chrome ore reserve is updated as per UNFC guidelines
5.	Chapter-4	Conceptual Mining Plan	Proposal of underground mining now been incorporated.
6.	Chapter-5	Mining	Proposal for underground mining now being incorporated
7.	Chapter-5.3	Yearly Production Plan	Change in Chromite ore production
8.	Chapter-6	Handling of Waste	Change in Overburden generation.
9.	Chapter-7	Use of Mineral	Change in Chrome ore requirement
10.	Chapter-8	Mineral Beneficiation	Proposed change in production and tailings disposal system.
11.	Chapter-9	Environment Management Plan	Change in Afforestation & Toe wall Construction programme
12.	Chapter-10(e)	Present Land use Pattern	Change in Present Land use Pattern.

13.	Chapter-17	Financial Assurance	Change in Present Land use pattern, at the end of scheme period, at the end of 2017-18.

The paras which have been modified/ introduced and the reasons thereof are described in next pages.

PART – I

REVIEW OF
2nd SCHEME OF MINING
(FROM 2008-09 TO 2010-11)

1. REVIEW OF 2nd SCHEME OF MINING

1.1 Name of Mine:

SUKINDA CHROMITE MINE

1.2 Particulars of Approval of Mining Plan:

The Mining Plan for Sukinda Chromite Mine over an area of 406 ha. was submitted under Rule 22 of MCR, 1960 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3) / 97 – MCMM (C) /MP – 7 dated 12.12.97 (copy of letter enclosed as Annexure-2).

The Scheme of Mining for Sukinda Chromite Mine over an area of 406 ha., was submitted under Rule 12 of MCDR , 1988 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3) / 97 – MCMM (C) /MP – S-6 dated 22.10.2003 (copy of letter enclosed as Annexure-3).

The Progressive Mine Closure Plan for Sukinda Chromite Mine over an area of 406 ha. was submitted under Rule 23B of MCDR , 1988 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3) / 2003 – MCMM (C) /MP – S-6 dated 08.08.2005 (copy of letter enclosed as Annexure-4).

The modification of Scheme of Mining for Sukinda Chromite Mine over an area of 406 ha., was submitted under Rule 10 of MCDR , 1988 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3) / 2007 – MCMM (C) /S-9 dated 05.07.2007 (copy of letter enclosed as Annexure-5).

The 2nd Scheme of Mining and Progressive Mine Closure Plan for Sukinda Chromite Mine over an area of 406 ha. was submitted under Rule 12 and 23B of MCDR , 1988 and was approved by the Controller of Mines (CZ), Nagpur, vide letter no. 314(3)/2008-MCCM(CZ)/MS-17/97, Dated: 02/02/09 (copy of the letter enclosed as Annexure-6).

Modified Scheme of Mining & Progressive Mine Closure Plan was submitted under Rule 10 of MCDR, 1988 and was approved by Controller of Mines (CZ), Nagpur, vide letter no. 314(3)/2011-MCCM(CZ)/MS-08, Dated: 22/07/11 for the period from 01.04.2011 to 11.01.2013 (copy of the letter enclosed as Annexure-7).

1.3 Date of Commencement of Mining Operations:

The company prospected the area under a PL dated 14.4.1952 and subsequently got a ML from the then Raja of Sukinda for a period of 20 years with effect from 22.10.1952. With the enactment of Orissa Estate Abolition Act (Act 1 of 1952), the area got vested with the State Govt. and the same Mining Lease was ratified by the State Govt. for a period of 20 years with effect from 12.1.1953.

Due to remoteness and inaccessibility of the place, non-availability of skilled manpower in the area and lack of market for the friable variety of chrome ore, the mining originally started in a small scale to supply refractory grade lumpy ore to our refractory plant at Jamshedpur & Belpahar during December 1960.

After the establishment of number of charge chrome plants in the country the demand for friable chrome ore went up towards the beginning of the 80's. Accordingly, the mine first went for semi-mechanization and subsequently was developed as a fully mechanized mine in the year 1984.

1.4 Review of salient features of the Approved Mining Plan:

a) Deficiency if any in the Approved Mining plan: None.

b) Review of important chapters of the Approved Mining Plan:

Salient features of the Approved Mining Plan were reviewed and the deviations / changes are enumerated below:

CHAPTER 3.0: Geology & Exploration

Chapter 3.2: Depletion of Reserves

The year-wise production of ROM of Chrome Ore during the period 01.04.2008 to 31.03.11 is given below in Table 1.4.1

Table 1.4.1
(ROM and Finished Ore production)

Year	Quarry	Finished Ore Production (MT)	Total ROM Production of all grade (MT)
2008-09	Friable	301893	929888
	Lumpy	127231	99466
	Pyroxenite	230910	296872
2009-10	Friable	289058	1031722
	Lumpy	104715	90297
	Pyroxenite	229694	223578
2010-11	Friable	299743	1168931
	Lumpy	61171	55914
	Pyroxenite	198220	202088
Grand Total (Friable)		**890694**	**3130541**
Grand Total (Lumpy)		**293117**	**245677**
Grand Total (Pyroxenite)		**658824**	**722538**

Thus 33.76 lakh MT of chrome ore ROM has been mined out during the period of 2008-11.

It may be noted that the finished ore production from Lumpy quarry in the above Table No. 1.4.1 is more than the ROM production from the quarry since it includes the production from the low grade dumps by way of manual segregation/ sorting & sizing of the same. The details of such production have been further shown under the Mineral Beneficiation chapter.

Although the production and development plan from 2011-12 onwards is proposed to be modified as per this modification to the Scheme of Mining, the actuals for 2011-12 is being given below since the period is almost over. This is also required for updating the reserves in the subsequent chapter on Reserves.

Table 1.4.2
(ROM and Finished Ore production)

Year	Quarry	Finished Ore Production (MT)	Total ROM Production of all grade (MT)
2011-12	Friable	343529	1253436
	Lumpy	108977	107437
	Pyroxenite	23284	0

There was no ROM removed from the pyroxenite section during the above period as same is being used temporarily for storage of tailings. Old pyroxenite boulders lying close to the pit were processed to produce the above amount.

Section 3.3.2: Year-wise Exploration proposed:

Table 1.4.3 below gives the exploration figures as proposed in the approved scheme of mining and the actual exploration done. As against the proposal of drilling 6116 m during the period 2008-12, the actual meterage drilled during the period 2008-12 was 22758 m.

Table 1.4.3

Year	**Proposed Bore-holes (No.'s)**		**Actual Bore-holes drilled**	
	Total No	Meterage	Total No	Meterage
2008-09	12	2222.40	12	2222.40
2009-10	13	3893.20	13	3866.60
2010-11	0	0	34	10513.25
2011-12	0	0	18	6155.60
TOTAL	25	6115.60	77	22757.85

Additional drilling of 34 boreholes (7 Exploratory, 9 Geotechnical, 15 Hydro geological & 3 In-situ Stress Measurement boreholes) comprising 10513 m total meterage was done in 2010-11 to serve as further inputs for feasibility of Underground Mining and underground resource estimation. Apart from that, further 18 exploration boreholes, 6155 m were done during the year 2011-12 for upgradation of resource from inferred to indicated category. The logging details of these boreholes have been enclosed as Annexure-8.

CHAPTER 5.0: Mining

Section 4.5: Year-wise Development & Production:

Table 1.4.4 below gives the details of production of chrome ore and pyroxenite with respect to the proposal in the approved scheme of mining.

Table 1.4.4

Year	Plan (MT)			Actual (MT)		
	OB X (Friable)	OB II (Lumpy)	Pyroxenite	OB X (Friable)	OB II (Lumpy)	Pyroxenite
2008-09	552300	174411	225250	301893	127231	230910
2009-10	552300	174411	225250	289058	104715	229694
2010-11	607530	191852	225250	299743	61171	198220
Total	1712130	540674	675750	890694	293117	658824

While the average production / annum of friable ore from OB X quarry has been @ 3 lakh MT/annum only, the average production / annum of lumpy ore from OB II quarry has been marginally less @ 1.00 lakh MT /annum. The variation in year- wise production with respect to the plan has been mainly due to the low market demand. This modification to the Scheme of Mining is accordingly being submitted for taking care of this difference. The production of pyroxenite during this period has been achieved.

Similarly the above data pertaining to the modification period 2011-12 is furnished below under Table 1.4.5 since the period is almost over.

Table 1.4.5

Year	Modified Plan (MT)			Actual (MT)		
	OB X (Friable)	OB II (Lumpy)	Pyroxenite	OB X (Friable)	OB II (Lumpy)	Pyroxenite
2011-12	358020	109980	25000	343529	108977	23284

Table 1.4.6 below gives the overburden quantities with respect to the proposal in the approved scheme of mining.

Table 1.4.6
(all figures in Cum.)

Year	**OB X Quarry**		**OB II Quarry**	
	Proposed OB	Actual OB	Proposed OB	Actual OB
2008-09	6000000	5237585	1000000	627594
2009-10	6700000	4513020	1000000	321061
2010-11	7500000	4156320	1200000	484432
Total	20200000	13906925	3200000	1433087

The yearwise quantity of overburden removal from 2009-10 onwards was scaled down and less than the plan due to less production requirement as stated above. This was done to avoid excessive overburden handling which would have otherwise resulted in an adverse impact on the cost.

Chapter 6.0: Handling of Waste / Sub-grade material

Section: 6.1: Generation and disposal of waste

Table 1.4.7 and 1.4.8 given below gives the deviation with respect to the stripping ratio for the period 2008-11 for the OB-X and OB-II quarries respectively. The stripping ratio is one of the key measures for assessing the extent of development of the mine since the year wise development plans had been conceptualized on the basis of the stripping ratio.

Table 1.4.7
OB X Quarry (Friable)

Year	Plan			Actual		
	Total ROM (lakh MT)	Proposed OB (lakh CuM)	Stripping Ratio	Total ROM (lakh MT)	Overburden (lakh CuM)	Stripping Ratio
2008-09	13.76	60.00	4.36	9.30	52.38	5.36
2009-10	13.76	67.00	4.87	10.32	45.13	4.37
2010-11	14.31	75.00	5.24	11.69	41.56	3.56
Total	41.83	202.00	4.83	31.31	139.07	4.44

The stripping ratio for OB-X quarry achieved during the period 2008-11 has been 4.44 with respect to the plan of 4.83. There has therefore been a marginal deviation in the stripping ratio and overburden removal and an apparent shortfall in the development. The shortfall in overburden removal was mainly due to lack of sufficient space for dumping of the overburden which led to increase in the cycle time of the dumpers. The ultimate stripping ratio for the OB-X quarry had been conceptualized as per the approved Scheme of Mining to be 5.12. However, this stripping ratio after optimization of the ultimate pit design of OB-X quarry has been now re-estimated to be 3.78 as per this modification, as discussed in greater detail in the later chapter on Conceptual Mining Plan. In view of the same there is presently no backlog of development in the OB-X quarry.

Table 1.4.8
OB II Quarry (Lumpy)

Year	Plan			Actual		
	Total ROM (lakh MT)	Proposed OB (lakh CuM)	Stripping Ratio	Total ROM (lakh MT)	Overburden (lakh CuM)	Stripping Ratio
2008-09	1.61	10.00	6.21	0.99	6.28	6.31
2009-10	1.61	10.00	6.21	0.90	3.21	3.56
2010-11	1.84	12.00	6.54	0.56	4.84	8.66
Total	5.06	32.00	6.33	2.45	14.33	5.83

The stripping ratio for OB-II quarry achieved during the period 2008-11 has been 5.83 as against a plan of 6.33. Here again there is a marginal shortfall with respect to the stripping ratio. This shortfall in development was mainly due to an increase in the breakdown of equipment as result of the increasing lead distances and depth of the quarry particularly during 2009-10 leading to less overburden removal. The average pit slope angle is another key measure for assessing the extent of development of the quarry. The maximum permitted overall slope angle for OB-II quarry is 45^0. The ultimate pit design and development plans had been conceptualized on the basis of such permitted slope angles. The present extents of the quarry and assessment of quarry sections for the overall pit slope angles has been reported later under the Conceptual Mining Plan is much less than the permitted overall slope angle. It is therefore evident that though there has been a marginal deviation with respect to the planned stripping ratio, there is no backlog in the development of the OB-II quarry.

Chapter. 8: Mineral Beneficiation:

Table 1.4.7 shows the year wise material planned to be fed in beneficiation process along with grade, material to be produced and tailing to be generated.

Table 1.4.7

YEAR	Feed grade % Cr_2O_3	Feed quantity lakh MT (wet)	Concentrate grade % Cr_2O_3	Concentrate Production lakh MT (wet)	Tailings grade, % Cr_2O_3	Tailings quantity lakh MT	Yield % by weight
2008-09	30-33	8.74	47-50	5.20	12 - 14	2.60	65-67
2009-10	30-33	8.74	47-50	5.20	12 - 14	2.60	65-67
2010-11	30-33	8.74	47-50	5.20	12 - 14	2.60	65-67
2011-12	30-33	8.74	47-50	5.20	12 - 14	2.60	65-67

Table 1.4.8 shows the actuals achieved against the plan as mentioned in Table 1.4.7.

Table 1.4.8

YEAR	Feed grade % Cr_2O_3 (dry)	Feed quantity lakh MT (wet)	Concentrate grade % Cr_2O_3 (dry)	Concentrate Production lakh MT (wet)	Tailings grade, % Cr_2O_3 (dry)	Tailings quantity lakh MT (wet)	Yield % by weight
2008-09	32.52	6.21	49.98	3.04	15.80	3.17	48.92
2009-10	32.30	7.50	50.69	3.67	14.67	3.83	48.94
2010-11	33.55	8.87	49.97	4.82	14.00	4.05	54.36
2011-12	33.71	9.00	50.66	4.85	13.93	4.15	53.84

It can be seen from the above tables that the feed grade as well as the concentrate grade has been consistently met. However the production was less compared to the plan due to slump in market conditions. Taking view of the above the concentrate production plan for the plan period has been reduced.

Chapter 9.0: Environment Management Plan

Afforestation

The year wise proposal of afforestation for last three years and compliance thereof is as given below in Table 1.4.9.

Table 1.4.9
Year wise Afforestation

Year	Plan		Actual	
	Nos	Area (ha)	Nos	Area (ha)
2008-09	11750	4.70	122000	4.70
2009-10	21750	8.70	84000	9.50
2010-11	20000	8.00	85000	8.50
Total	53500	21.4	291000	22.7

It may be seen from the table that the number of saplings planted per ha is at higher side compared to the norms. The reason for the same is that the overburden dumps are mainly of Ni-limonite which contains nickel. As a practice bushy type of plants were being planted in the earlier years in large nos., for stabilization of the dump slopes and allow for its future reuse after establishment of an economically viable technology for nickel extraction.

Total 2150 m of retaining wall using earth, overburden material and boulders were made against the plan of 1700m.

Air & Water Quality:

As planned, ambient air quality is being monitored at a frequency of twice a week by conducting 24 hour sampling and all the parameters are found to be within prescribed limits. Similarly mine water is being treated in effluent treatment plants before discharging out of the lease hold area to ensure that all the water quality parameters are within the permissible limits Water quality at lease end will continue to be monitored at a frequency of once a month for all water quality parameters as prescribed by the Orissa Pollution Control Board. The year wise ambient air and water quality recorded during the period 2008-11 is enclosed as Annexure-9.

c) **Review of compliance position of conditions imposed in the approval letter of the Mining Plan:**

The mining plan was approved subject to the condition that forest clearance from the appropriate authorities should be obtained for the land coming in the forest area as per Forest (Conservation) Act, 1980. The leasehold area of 406 ha. comprises of 73.697 ha. of forest land which was already broken up prior to 25.10.1980. The diversion proposal for the same was duly submitted and Stage II Forest Clearance has been obtained for diversion of the entire 73.697 ha. forest land vide approval letter no. 8-78 / 96 FC dated 27.01.98 (Annexure-10). At the end of every year, plans and sections showing the actual workings had been prepared and submitted.

d) Review of compliance position of violations pointed out by IBM:

i) Concern has been raised during Dec'2010 as regard the lag in proposed development of OB-II quarry by adopting 1: 5 stripping ration compared to planned stripping ratio of 1:6.

During 2008-09 and 2009-10 overburden removal in OB-II quarry was affected due to major breakdown of heavy earth moving vehicles and consequent poor availability. Corrective actions were taken subsequently to make good the lag in development and the stripping ratio for the year 2010-11 was maintained at 1: 8.66 resulting the overall stripping ratio close to 1:6 during 2008-11.

ii) Concern has been raised during Dec'2010 for production of pyroxenite of 4.6 Lakh tonnes during 2008-10 compared to planned production of 4.5 Lakh tonnes.

The total pyroxenite ROM production during 2008-10 is 5.2 lakh tonnes compared to planned ROM production of 5.3 lakh tonnes. The finished pyroxenite ore production of 4.6 lakh tonnes includes 7000 tonnes of fines/ mineral rejects generated earlier which was taken into production after dispatch of the same. Thus net finished ore production has been 4.53 lakh tonnes against the planned production of 4.5 lakh tonnes during 2008-10 on like to like basis. The increase in production is 0.67% only which was resulted due to high recovery and utilization of mineral rejects.

iii)Concern had been raised that back filling over 4.0 ha area on western side of OB-II quarry has been undertaken without making any provision in the approved Scheme of Mining.

Provision for backfilling over 4.05 ha area on western side of OB-II quarry had been duly submitted along with the previous modification of Scheme of Mining for the period 2006-07 to 2007-08. The exact location of backfilling has also been specifically provided in the Drawing no. SCM/MP/17/06 along A-A' section. The above Modification of the Scheme of Mining had been duly approved by the Controller of Mines (CZ) vide his letter no. 314(3)/2007-MCCM(C)/S-9 dated 05.07.2007. Backfilling of the western side of OB-II Quarry has accordingly been carried out thereafter in the year 2007-08. As per Clause (x) of the above approval letter and in accordance with Rule 23E (2) of MCDR, 1988, we have duly submitted the annual report on information on protective measures taken as per PMCP under cover of our letter no. SCM/GM (O)/2947/08 dated 20.10.2008; wherein we have duly confirmed to have done the back-filling over an area of 4.05 ha.

iv)On perusal of the up-dated section along 2600E and 3700E (DRG. No. SCM/MP/13/08) concern was raised that development in northern side of OB-II Quarry is lagging behind the stipulated stripping limit of 2009-10 set in the Approved Scheme of Mining.

As per the section, the stripping limit of the top most bench on the northern side for the year 2009-10 as per the proposal in the Approved Scheme of Mining has a co-ordinate of 1459N and 94mRL. The position of the actual stripping limit achieved for the topmost northern bench represented by the pink line has a co-ordinate of 1524N and 134mRL. Since, the co-ordinates of the actual development done is upto 1524N which is more than the limit of 1459N in the Approved Scheme of Mining for the year 2009-10, no lag in development is therefore evident in the northern side of OB-II Quarry. Moreover, since the overall pit slope of the northern benches at OB-II quarry is also much below the maximum permitted slope angle of 45 degrees as has been permitted by DGMS, no lag in development is presently perceived.

v) Concern was raised that a common person has been appointed as mining engineer and mines manger and agent of the mine.

Separate person was appointed as Mining Engineer in Form-1 vide letter no. SCM/MIN/19/959, dated 29/01/09.

e) Any other points requiring attention:

1. Permission was obtained from IBM, Nagpur to prepare the Surface Plan & Surface Geological plan on a scale 1:4000 vide letter No. N-11013/61/MP/89-CCOM Vol-4, dated 28.08.07 attached as Annexure-11.

2. The Memorandum of Association and Article of Association of Company dated 27th July' 2005 and Dy. Registrar of Companies letter no. 11-260, dated 12th Aug' 2005, the name company was changed from Tata Iron & Steel Co. Ltd. To TATA STEEL LIMITD. Copy of the letter is enclosed as Annexure-12.

3. M/s ORSAC, Bhubaneswar has been declared as the authorized agency by the State Govt. for carrying out the DGPS survey to comply with the CCOM Circular No. 02/2010. Such DGPS survey by ORSAC has been done for the lease during July'2011. However, the superimposed maps have still not been provided by ORSAC although we have been following up on this matter since long. The information in this regard shall be furnished to the IBM as soon as the same is finalized and received from ORSAC. The minutes of the meeting as circulated in the web site of the Dept. of Steel & Mines, Govt. of Odisha vide letter no. 6202/SM, Bhuabaneswar, dated 06.08.2011 in this regard may please be referred to as Annexure-13.

PART – II

PROPOSALS UNDER MODIFIED SCHEME OF MINING (FROM 2011-12 TO 11th JAN'2013)

2.0 PROPOSAL UNDER SCHEME OF MINING FOR THE NEXT FIVE YEARS

2.1 NAME AND ADDRESS OF THE APPLICANT :

The applicant and the present nominated owner of the mine is Mr. H. M Nerurkar. The copy of photo ID of nominated owner along with passport details towards address proof is enclosed as Annexure-14. The details of address are as follows.

Mr. H. M Nerurkar,
Managing Director
Tata Steel Ltd.
At. / P.O. : Jamshedpur
Dist : East Singhbhum
Jharkhand : 831001
Phone : 0657- 2423298
Fax : 0657- 2431818
Email : mdoffice@tatasteel.com

The present members of the Board of Directors of TATA STEEL are:

Name	Pan number	Voter ID number	Citizenship	Passport number / DL number
Ratan N. Tata	AAAPT0002F	MT 04 019 060249	Indian	Z 2177343
B. Muthuraman	AAHPB7489R	JVN 1545873	Indian	H 5574492
Nusli N. Wadia	AAAPW0990M	Not allotted	Indian	F 6352000
S.M. Palia	AABPP2138D	Not allotted	Indian	A 6129072
Suresh Krishna	AABPK3154E	Not allotted	Indian	Z 1759392
Ishaat Hussain	AAVPH 6348F	MT 04 019 057708	Indian	Z 2026139
Jamshed J. Irani	AAFPI1888M	BR 50292 333139	Indian	H 6855092
Subodh Bhargava	AAIPB9290R	MT 08 038 363390	Indian	Z 2007543
Jacobus Schraven	----	Not Applicable	Dutch	NT07K92F5
Andrew M. Robb	----	Not Applicable	British	800240979
Karl-Ulrich Kohler	----	Not Applicable	German	507726111
H.M. Nerurkar	ABGPN0776M	JVN 4233342	Indian	G 9227924

The resolution of Board of Directors for nominated owner, Memorandum of Association & Article of Association issued under Companies Act etc. is enclosed as Annexure-15 & Annexure- 12 respectively.

2.2 NAME, ADDRESS AND REGISTRATION NUMBER OF THE RECOGNIZED QUALIFIED PERSON:

The Modification of Scheme of Mining is prepared by G K Guin, Key Person for the Tata Steel Ltd. Registration No. RQP/BBS/044/2003/A, valid till 03.02.2013 is attached as Annexure- 16. The address of the key person is given below:

Mr. G K Guin
Head (Mine and Production Planning)
Sukinda Chromite Mine, Tata Steel Limited

At/ P.O-	:	Kalarangiatta
Dist	:	Jajpur
State	:	Orissa
Pin Code	:	755028
Phone	:	09238100835
Fax	:	06726 - 268734

The Key Person has been assisted by the officers and staff of Planning Cell. Sri. Haridrumat Behera, RQP, having Registration No. RQP/BBS/093/2010-A valid up to 19.04.2020 (Copy of RQP certificate enclosed as Annexure-17) has assisted in the chapters related with underground mining as per this modification. Services from the departments of Geological Services, Land & Lease, Safety, Environment and the field personnel of Sukinda Chromite Mine were also availed in relevant areas of their expertise.

2.3 MINERALS TO BE MINED :

Chromite shall continue to be mined. Mining of pyroxenite shall be temporarily discontinued till Jan'2013 (i.e till the end of the plan period) as per the approved Modified Scheme of Mining. Shortfall in requirement of flux mineral for the company shall be partially met from purchased pyroxenite ore from the Hata, Chaibasa region and the balance as dolomite ore from the existing Gomardih Dolomite quarry operated by the company in Sundergarh district of Orissa.

2.4 AREA AND DATE OF EXPIRY OF LEASE :

The 2nd renewal over a reduced area of 406 ha. has been granted by Central Govt. vide letter no. 5(22)/95-MIV, dated 17th Aug'1995 (Annexure-1) for a period of 20 years with effect from 12.01.93. The Mining Lease over an area of 406ha is valid till 11.01.2013. Co-terminus with the mining lease we have been allotted 100ha of additional area for overburden dumping contiguous to southern lease boundary. This 100 ha was essentially forest land for which forest clearance has been obtained. Copy of the letter of approval granted by the MoEF is enclosed as Annexure-21. We have further applied for 73.685ha of area south of the additional area of 100ha for tailing disposal.

The Key Plan of the lease area of 406 ha, the allotted area of 100ha contiguous with the southern lease boundary of the existing lease and the applied area of 73.685ha is shown in Drawing No. SCM/MP/01/12. The area falls under the Survey of India Topo Sheet no 73-G/12, 73-G/16, 73-H/9, 73-H/13. The Surface Plan of the leasehold area is shown in drawing no. SCM/MP/02/12 updated as on 01.08.2011. The extent of Latitude and Longitude of the above area is mention below in Table. 2.4.1.

Table: 2.4.1
Details of Latitude & Longitude

Sl No.	Area Type	Area (ha)	Latitude From	Latitude To	Longitude From	Longitude To	Remarks
1	ML Area	406.00	$21^0$00’39.60”	$21^0$02’5.81”	$85^0$44’27.10”	$85^0$46’22.37”	Mining Lease Area
2	Additional Area	100.00	$21^0$00’27.41”	$21^0$01’1.68”	$85^0$45’22.09”	$85^0$46’51.42”	Allotted for dumping of Overburden
3	Applied Area	73.685	$20^0$59’34.88”	$21^0$00’51.00”	$85^0$46’46.84”	$85^0$47’32.69”	Applied for disposal of Tailing

Justification for Applied Area of 73.685 Ha:

The applied area of 73.685 ha. consists of 8.373ha of forest land which shall serve as part of the corridor to reach the tailing disposal area of 65.312ha of non-forest Govt. land. An area of 70 ha was allotted within the existing mining lease area of 406 ha for the disposal of waste during the second renewal of the lease. 10ha out of the 70ha area for waste disposal was earmarked for tailing pond. This 10 ha area is now full and no further tailings can be accommodated within this area since the tailing pond has acquired the maximum safe height. The State Govt. at the time of takeover of the Bamnipal Plant had assured to recommend to Govt. of India for an additional area of 180 ha for the disposal of waste. So far an additional area of 100 ha only have been allotted for waste disposal. The existing mining lease area of 406 ha is surrounded in the north, east and west directions by other lessees. The applied area of 73.685 ha consisting of mostly non-forest land is located further south of the additional area of 100 ha and is therefore the only suitable area available in close proximity to the mine for the operations to be economically viable. The applied area is further south of the Mahagiri Hill range and outside the limits of the Sukinda Ultramafic Complex and hence devoid of any mineral. The requirement of land for tailing disposal has been estimated based on the space required to store the quantum of dry tailing to be generated over the life of the beneficiation plant, runoff management arrangements, storm water handling and water recirculation facilities.

2.5 DATE OF EXPIRY OF THE APPORVED SCHEME OF MINING :

The approved Scheme of Mining and Progressive Mine Closure Plan is valid till 11.03.2013.

3.0 RESERVES :

3.1 CATEGORY-WISE RESERVES INDICATED IN THE APPROVED SCHEME OF MINING :

The mineable reserve including all categories was updated during the second review of approved Scheme of Mining and was estimated to be 9.752 million tonnes as on 31.03.08 as shown in Table no 3.1.1.

Table 3.1.1
Summary of Reserves as on 01.04.2008 (in Million Tonnes)

<table>
<tr><th rowspan="2">Ore body</th><th rowspan="2">Grade (%Cr2O3)</th><th colspan="2">Reserve (31.03.2008)</th><th>Total</th></tr>
<tr><th>111</th><th>122</th><th>111+122</th></tr>
<tr><td rowspan="4">Northern Band</td><td>+ 10 - 40</td><td>0.052</td><td>0.799</td><td rowspan="3"></td></tr>
<tr><td>+ 40 - 52</td><td>0.003</td><td>2.033</td></tr>
<tr><td>>52</td><td>2.667</td><td>0.342</td></tr>
<tr><td>Total</td><td>2.722</td><td>3.174</td><td>5.896</td></tr>
<tr><td rowspan="4">Mid Band</td><td>+ 10 - 40</td><td>0.150</td><td rowspan="3">2.936</td><td rowspan="3"></td></tr>
<tr><td>+ 40 - 52</td><td>0.006</td></tr>
<tr><td>>52</td><td>0.004</td></tr>
<tr><td>Total</td><td>0.160</td><td>2.936</td><td>3.096</td></tr>
<tr><td rowspan="4">Southern Band</td><td>+ 10 - 40</td><td>0.049</td><td>0.347</td><td rowspan="3"></td></tr>
<tr><td>+ 40 - 52</td><td>0.364</td><td>0.000</td></tr>
<tr><td>>52</td><td>0.000</td><td>0.000</td></tr>
<tr><td>Total</td><td>0.413</td><td>0.347</td><td>0.760</td></tr>
<tr><td colspan="2">Total (NB+MB+SB)</td><td>3.295</td><td>6.457</td><td>9.752</td></tr>
</table>

3.2 DEPLETION OF RESERVES :

The year-wise production of ROM of Chrome Ore during the period 01.04.2008 to 31.03.11 is given below in Table 3.2.1. 47.37 lakh MT of chrome ore ROM has been mined out during the period of 2008-11.

Table 3.2.1
ROM production (in million tonnes)

Orebody	Grade (%Cr2O3)	Depletion of Reserves (2008-12)									
		2008-09		2009-10		2010-11		2011-12		Total	
		111	122	111	122	111	122	111	122	111	122
Northern Band	+ 10 - 40	0.006		0.045		0.042		0.008		0.101	0.000
	+ 40 - 52	0.003	0.154	0.000	0.165	0.000	0.194	0.000	0.293	0.003	0.806
	>52	0.145		0.089		0.080		0.051		0.365	0.000
	Total	0.154	0.154	0.133	0.165	0.121	0.194	0.059	0.293	0.468	0.806
Mid Band	+ 10 - 40	0.095		0.055		0.000		0.000		0.150	
	+ 40 - 52	0.006	0.517	0.000	0.679	0.000	0.853	0.000	0.901	0.006	2.950
	>52	0.004		0.000		0.000		0.000		0.004	
	Total	0.105	0.517	0.055	0.679	0.000	0.853	0.000	0.901	0.160	2.950
Southern Band	+ 10 - 40	0.074		0.070		0.041		0.082		0.266	0.000
	+ 40 - 52	0.025		0.021		0.015		0.025		0.087	0.000
	>52	0.00		0.000		0.000		0.00		0.000	0.000
	Total	0.099	0.000	0.090	0.000	0.056	0.000	0.107	0.000	0.353	0.000
Total (NB+MB+SB)		0.359	0.671	0.278	0.844	0.177	1.047	0.167	1.194	0.981	3.756

3.3 ADDITIONAL RESERVES ESTABLISED :

As mentioned in the approved Mining Plan the major part of the core drilling done during the period 1998-02 was essentially for establishing the geological structure of the host rock, hydro-geological investigations, rock strength parameters etc. for the study on open pit optimization. Some of the exploratory drilling undertaken during 2002-12 was for the purpose of gap filling and prove the resources at greater depths. The geological sections & ore body model were accordingly updated. The updated geological sections have been enclosed as drawing no SCM/MP/04/12.

The eastern side of the middle chrome ore band, which was earlier planned to be mined up to 140mRL, is now proposed to be mined up to 20mRL by re-handling of the existing dump.

i) RESOURCE ASSESSMENT OF CHROME ORE

Basis of Resource Estimation

All exploration data were compiled systematically and stored in excel database. Surpac 6.1.3 software was used for modeling and resource estimation. Different steps followed in the modeling of the ore bodies are described below.

Geological Database Creation:
The data consists of coordinates of borehole locations along with their orientation, depth-wise record of lithology and analytical results of samples from boreholes. These were entered

in Excel Worksheet files. Proper care was taken to ensure error free exploration database. The basic data was collected and stored in excel CSV format as per the structure below:

Borehole Collar file:
The final collars of boreholes were properly surveyed and those information were stored in the borehole collar file having the following fields: Borehole ID, Northing, Easting, mRL, Total Depth of borehole and Path of borehole.

Borehole Survey file:
Due to near-vertical nature of the ore body, the boreholes were planned inclined to intersect the ore body at desired levels. The borehole deviation data was checked in the field and later on stored systematically in the database. As the general strike of the orebody is 59°, the azimuth of the boreholes was set to be either 149° (for boreholes looking South) and 329° (for boreholes looking North). As the local mine grid is rotated from the true north at an angle of 31°, the actual azimuths of boreholes were later on converted to grid azimuth by addition of 31° to the actual survey data. The survey file contains the following fields: Borehole ID, Depth (at which survey is taken), Inclination, Azimuth and Remarks.

Borehole Geology file:
All the drilled core was systematically logged and the data was stored in digital format. Geology file contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Primary lithology (PCODE), Secondary lithology (SCODE), Remarks.

Borehole Assay file:
Assay file contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Core Recovery, Code (Ore/Waste), Cr_2O_3%, Fe%, SiO_2%, Al_2O_3%, CaO%, MgO%, Mn%, TiO_2%, Ni%, Co% & Remarks.

Zone file:
Zones were defined with Threshold value of 10% Cr2O3 as per IBM guideline. It contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Cr2O3%, Fe%, SiO2%, Code (Ore/Waste) & Remarks.

Determination Of Specific Gravity:
Core samples of 8 to 30 cm length, averaging 10cm length, free of visible moisture were selected. The samples were dried, weighed in air and the mass in air (Ma) recorded to the nearest 0.1 g. The sample was then suspended in water and the mass in water (Mw) was recorded. Calculation of the density was done by the following formula:

Density = Ma ⁄ (Ma – Mw)

Nearly 2000 specific gravity measurements of chromite ore & host rocks in the hanging wall & foot wall were taken. Correlation of Cr2O3 grade with specific gravity was established for each ore band and correlation equation was developed, which, in turn, was used in block model estimate for specific gravity.

Northern band yielded the following correlation equation:
$y = 0.038x + 1.857$.

Middle band yielded the following correlation equation:
$y = 0.050x + 1.221$.

Southern band yielded the following correlation equation:
$y = 0.028x + 2.578$

Where y = Specific gravity
x = % Cr_2O_3

Solid Body Modeling:

Integrating topographical survey data, geological information and chemical analysis, systematic database was created using SURPAC (6.1.3 version) software. This database was the foundation for 3D modeling of ore body & rock mass, estimation of the grade and calculation of the resources / reserves. The surface topography was sectioned along cross-section lines at 50 m interval (1650E to 4150E) covering area of interest to generate topographical profiles for cross-sections preparation. String files of borehole trace, lithology, assay and ore zones in boreholes along 52 transverse sections from 1650E to 4150E at 50 m interval were extracted from geological database for correlation. The transverse sections thus generated were correlated for continuity of ore body and other lithologies. The chromite body and the other lithologies were correlated upto the present topography as per the topographical profiles along each section and the boundary strings for each lithology were digitised.

This was followed by litho-wise stitching of the ore/lithologies to have a model of each ore band and individual lithologies. All ore/litho solid were combined to get the 3D litho-model of the entire leasehold. The snapshot of the orebody model showing all three orebodies is given below in Fig 3.5.

Determination of hard-soft Boundary in the Northern & Middle ore bands:

The character of the weathered material in the Northern & Middle chromite ore bands goes from soil-like near the surface to weak friable ore. The unweathered ore is fairly hard and lumpy and there is a transition zone where the ore goes from friable to lumpy. The soft powdery ore above the lumpy ore is un-mineable by conventional underground method of mining.

As no methodology has been suggested till date for mining the friable ore from underground, for Northern and Middle Band, one more exercise was carried out to determine the boundary between Friable & Lumpy Ore. For determining the same, all available drilled core in the ore zone was critically re-logged to see whether the ore is of friable or lumpy in nature. The collected data was plotted in the Longitudinal Vertical Section. The boundary between friable and lumpy portion of the individual ore body were delineated from the LVS which was followed by creation of the Friable-Lumpy Boundary DTM. The depth of the weathering for the North and Middle Bands varies from about 0 mRL in the west to -250 mRL in the east for the North Band, while the same for the Middle Band is less the -200 mRL.

The schematic diagrams of the longitudinal sections showing the above weathering profile in the Northern and Middle chromite ore bands is given below in Fig.'s 3.6 to 3.9.

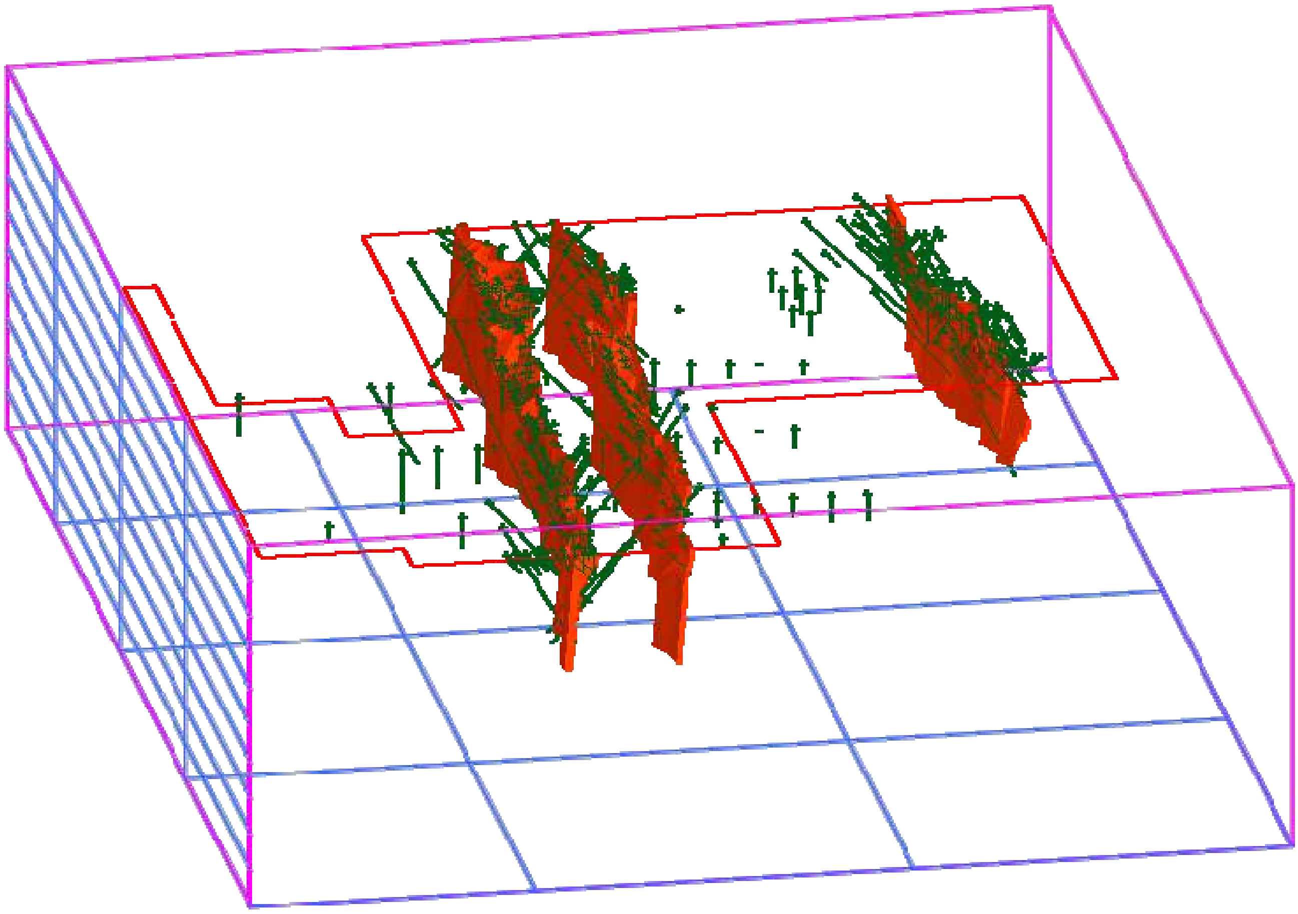


Figure 3.3.1: Snapshot of the ore body model showing all three chrome ore bands

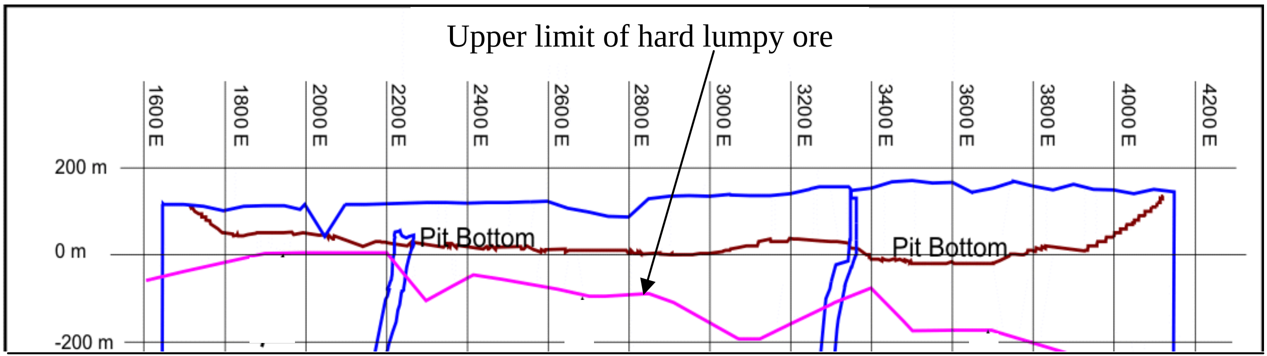


Fig. 3.6: Longitudinal Section of North Band showing Weathering Profile

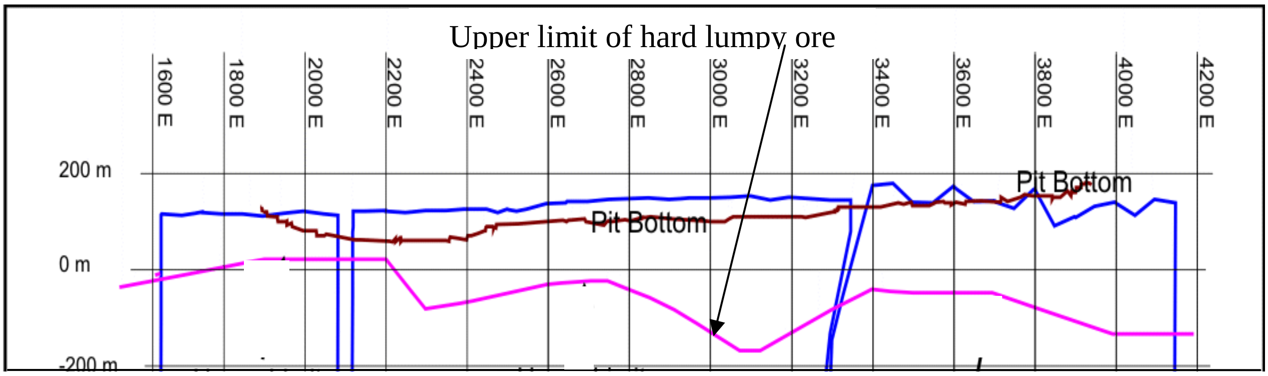


Fig. 3.7: Longitudinal Section of Middle Band showing Weathering Profile

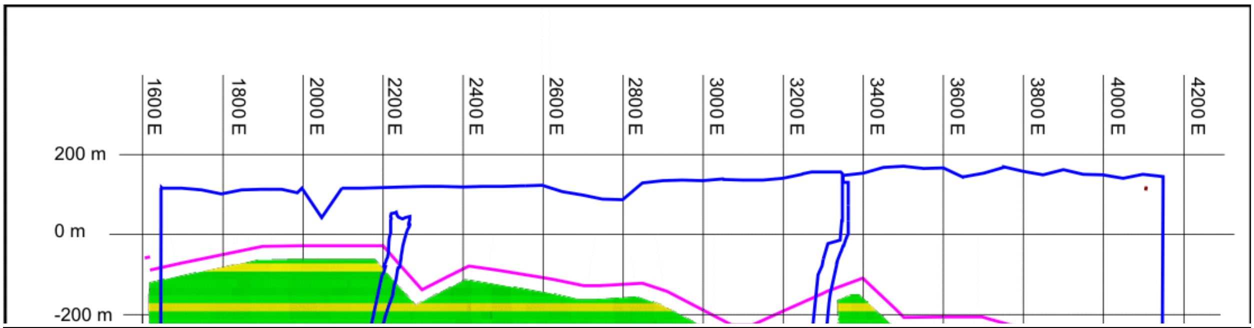


Fig. 3.8: Longitudinal Section of North Band Hard Lumpy Ore Zone (in green)

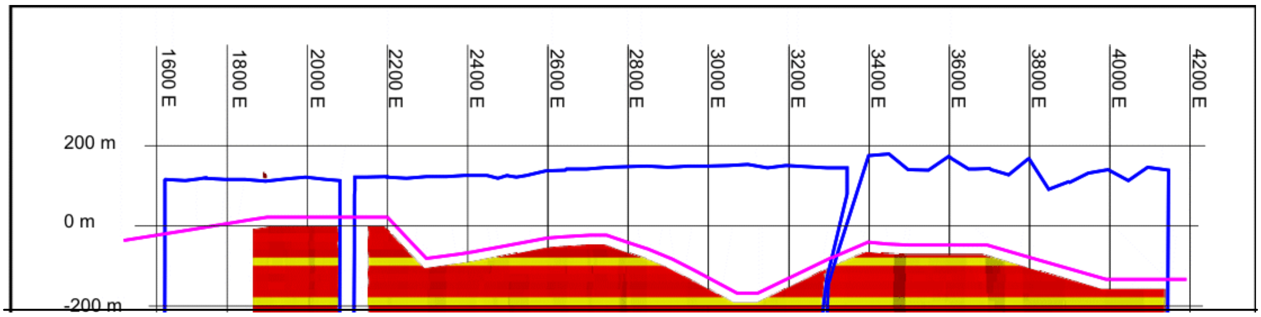


Fig. 3.9: Longitudinal Section of Middle Band Hard Lumpy Ore Zone (in red)

Preparation Of Block Model:

For block model estimation, composite samples were made at 2 m intervals inside the wireframe model and basic statistical values were determined. Geostatistical analysis was carried out to understand the distribution type, variance, standard deviation, etc, followed by variogram modeling. For each variable (i.e. Cr_2O_3, Fe and SiO_2), variogram along three directions i.e. along strike, along dip of the ore body and across strike of the ore body was generated. Based on the experimental curve generated by the data points, a model curve was fitted to smoothen the experimental curve and range, nugget (C0) and sill (C1) were calculated in each direction of the variogram.

These variogram parameters obtained were used in the estimation by Ordinary Kriging. While defining the search ellipse parameter for estimation, the search ellipse was defined considering the disposition of the ore body. Because the ore body is striking in E-W direction and dipping approximately 80^0 towards north, bearing of the search ellipse was considered 90^0 and dip at 80^0, so that the search ellipse move in down dip direction while estimation.

ii) BASIS OF RESOURCE CATEGORISATION AS PER UNFC:

For delineation of the resource boundaries field guidelines of UNFC were followed. The following criteria were taken under consideration

For Measured Resource: 100 m (laterally) x 50 m (vertically) grid spacing
For Indicated Resource: 200-300m (laterally) x 100–150 m (vertically) grid spacing
For Inferred Resource: 300 m grid spacing

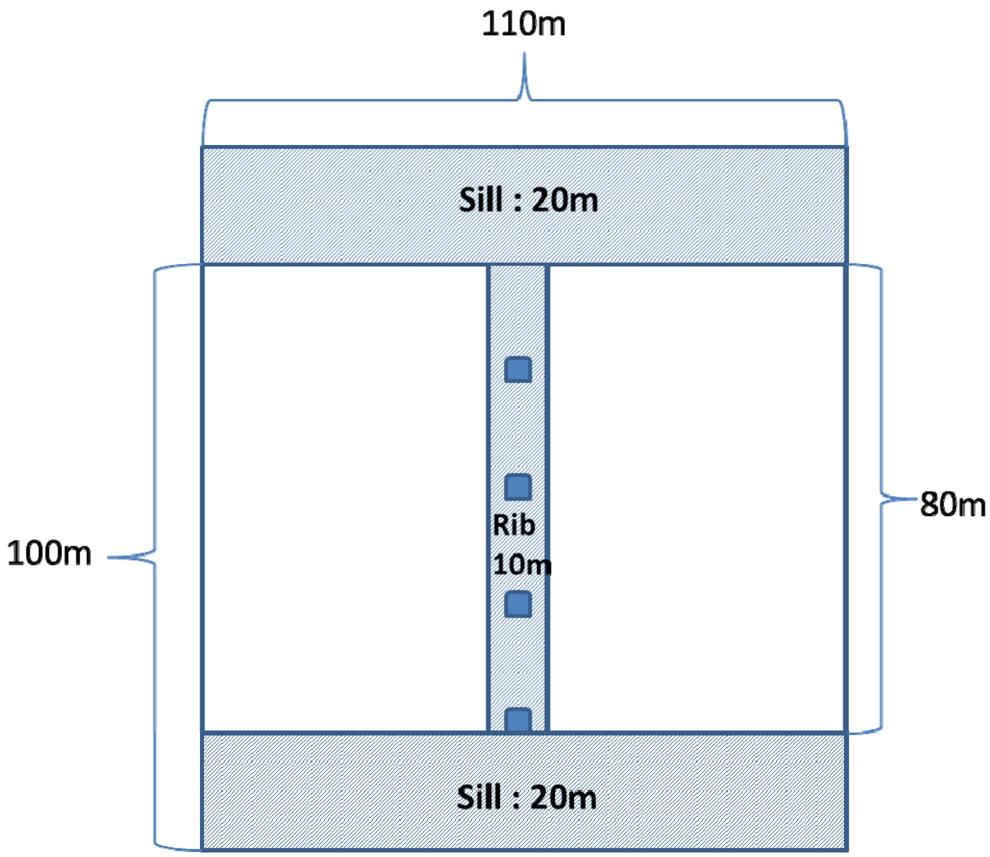
The depth extension parameters for resource estimation have been considered to be 25m below the positive intersection depth for the Measured Resources. For Indicated resources, where there is lack of borehole intersection, depth influence of 25 m below the measured resource boundary has been considered. Similarly for Inferred resources, a depth influence of 25 m below the indicated resource boundary has been considered. For all the categories, 10% Cr2O3 cut-off grade has been considered for the resource estimation.

Thereafter, the resultant mineable reserves and non-mineable resources has been derived out of resources after due consideration of the feasibility studies and with consideration of modifying factors (UPL, Statutory barriers ,pillars, geotechnical reasons, environmental/forest clearances etc.) for both U/G and O/C.

Crown Pillar:

The development within the orebody is proposed to commence below the crown pillars which has been shown in the longitudinal geological section for all 3 orebodies (Drawing no.'s SCM/MP/21/12). As per geotechnical investigation report (enclosed as Annexure-22, Ref. pp-53) it is stated that a crown pillar will be unstable in the soft friable chrome ore material and pillars can therefore not be developed in this material. Accordingly, the crown pillar has been designed to be below the weathered horizon of the Northern and Middle ore bands and within the hard and competent ore. Presently the thickness of the crown pillar for all three ore bodies has been proposed to be 60m in line with the requirements of the statute.

Sill & Rib Pillar:



Total Area of the Block = 110m * 100m =11000m^2
The Area blocked in Sill & Rib Pillar = (110m * 20m) + (80m * 10m) = 3000m^2
Hence the % of area blocked in Sill & Rib Pillar = (3000/110000) *100% =72.87%

As a conservative approach, the % of resource blocked in Sill & Rib pillars has been taken as 30%.

The conceptual longitudinal vertical section to elaborate the basis of categorization as per UNFC for the Northern & Middle chromite ore bands are given below in Fig 3.3.2 and for Southern band is given below in Fig 3.3.3.

Northern & Middle Band

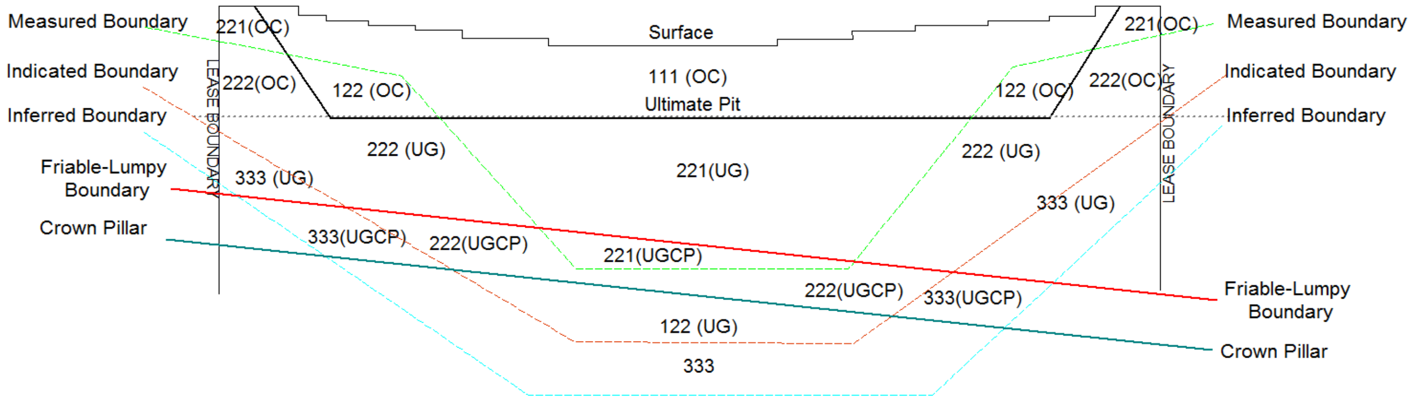


Fig. 3.3.2: Conceptual Longitudinal Vertical Section of Northern & Middle Band

Southern Band

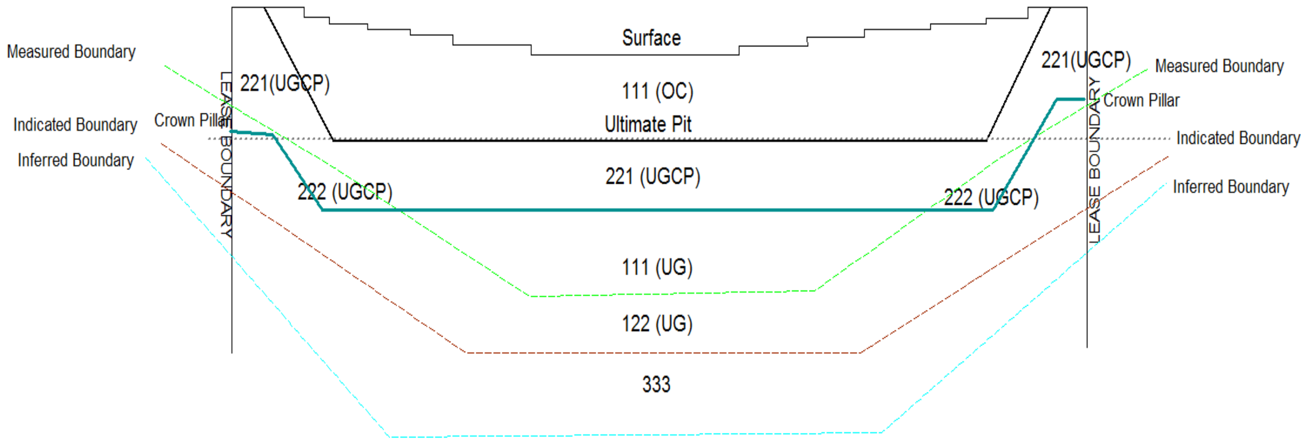


Fig. 3.3,3: Conceptual Longitudinal Vertical Section of Southern Band

Based on above facts and information/data generated during the course of exploration and proposed change in mining practices, additional resource for chrome ore was established as given in Table 3.3.1.

Table 3.3.1
Addition of Resources (2008-12)
(in Million Tonnes)

Ore Body	Category	UNFC Category	Grade Range, $\%Cr_2O_3$	Addition of Resources (Million tonnes)
Northern Band	Proved Mineral Reserve - Open Cast	111	10-40	0.332
			40 - 52	1.588
			>52	-0.520
	Probable Mineral Reserve - Open Cast	122	10-40	-0.596
			40 - 52	-0.364
			>52	-0.063
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	211	10-40	0.314
			40 - 52	0.710
			>52	0.567
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	222	10-40	0.169
			40 - 52	0.372
			>52	0.038
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	333	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	221	10-40	0.171
			40 - 52	1.422
			>52	5.397
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	222	10-40	1.723
			40 - 52	6.295
			>52	4.162
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	333	10-40	0.290
			40 - 52	1.857
			>52	2.798
	Prefeasibility Mineral Resource blocked in crown pillar	221	10-40	0.000
			40 - 52	0.067
			>52	0.174
	Prefeasibility Mineral Resource blocked in crown pillar	222	10-40	0.234
			40 - 52	2.229
			>52	1.728
	Inferred Mineral Resource blocked in crown pillar	333	10-40	0.134
			40 - 52	1.296
			>52	1.241

	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.406
			40 - 52	0.903
			>52	0.367
	Proved Mineral Reserve - Underground	**111**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	0.947
			40 - 52	2.106
			>52	0.857
	Inferred Mineral Resource - Underground	**333**	10-40	0.540
			40 - 52	2.684
			>52	1.631
	TOTAL RESERVE + RESOURCES			**44.209**
Middle Band	Proved Mineral Reserve - Open Cast	**111**	10-40	2.027
			40 - 52	1.221
			>52	0.078
	Probable Mineral Reserve - Open Cast	**122**	10-40	1.617
			40 - 52	0.808
			>52	0.013
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**211**	10-40	0.299
			40 - 52	0.023
			>52	0.000
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**222**	10-40	0.753
			40 - 52	0.088
			>52	0.000
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	**333**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**221**	10-40	0.003
			40 - 52	0.126
			>52	0.033
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**222**	10-40	2.027
			40 - 52	4.998
			>52	0.586
	Underground Inferred Mineral Resource: Presently non-mineable	**333**	10-40	0.054
			40 - 52	0.208

	because of friable nature of the formation		>52	0.001
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	1.703
			40 - 52	2.754
			>52	0.225
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.122
			40 - 52	0.083
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.686
			40 - 52	0.890
			>52	0.025
	Proved Mineral Reserve - Underground	**111**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	1.601
			40 - 52	2.076
			>52	0.059
	Inferred Mineral Resource - Underground	**333**	10-40	0.640
			40 - 52	1.177
			>52	0.020
	TOTAL RESERVE + RESOURCES			**27.024**
Southern Band	Proved Mineral Reserve - Open Cast	**111**	10-40	0.482
			40 - 52	-0.190
			>52	0.000
	Probable Mineral Reserve - Open Cast	**122**	10-40	-0.347
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	1.328
			40 - 52	0.364
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	0.482
			40 - 52	0.010
			>52	0.000
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.000
			40 - 52	0.000

			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.070
			40 - 52	0.005
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	1.145
			40 - 52	0.077
			>52	0.000
	Proved Mineral Reserve - Underground	**111**	10-40	0.163
			40 - 52	0.011
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	2.671
			40 - 52	0.180
			>52	0.000
	Inferred Mineral Resource - Underground	**333**	10-40	1.530
			40 - 52	0.090
			>52	0.000
	TOTAL RESERVE + RESOURCES			**8.070**

3.4 CATEGORY-WISE UPDATED RESOURCES :

The category – wise updated reserves for chrome ore as on 31.3.2008 was 9.752 million tonnes as given in Table- 3.1.1. This was the recoverable reserve considering the resource within the conceptualized ultimate open pit limits only. The band-wise, category-wise mineral resource as on 31.03.2012 for chrome ore, considering the depletion during 2008-12 and subsequent addition of resource was calculated and given in Table - 3.4.1 (a), (b) & (c) below.

The cut-off grade for chrome ore is considered to be 10% Cr_2O_3. Chemical analysis of ore/associated mineral/sub grade/waste from Odisha Govt. lab at Jajpur Road & M/s Shiva Analyticals (India) Ltd., Bangalore (NABL accredited lab) is enclosed as Annexure-32. The company's chemical analysis lab at the mine has also been recognized as a standard reference laboratory by the Directorate of Geology, Govt. of Odisha.

The criteria G1, G2 & G3 level as per UNFC has been synthesized on the basis exploration carried out over entire lease area on spatial and level-wise till date is as follows

Detailed Exploration (G1): 100 m (laterally) x 50 m (vertically) grid spacing
General exploration (G2): 200-300m (laterally) x 100–150 m (vertically) grid spacing
Prospecting (G3): 300 m grid spacing

Accordingly corresponding resource categories have been estimated. The depth extension parameters for resource estimation have been considered to be 25m below the positive intersection depth for the Measured Resources. For Indicated resources, where there is lack of borehole intersection, depth influence of 25 m below the measured resource boundary has been considered. Similarly for Inferred resources, a depth influence of 25 m below the indicated resource boundary has been considered. For all the categories, 10% Cr2O3 cut-off grade has been considered for the resource estimation. The resources under G1, G2 & G3 axis have been further categorized to reserves and remaining resources is given in Table 3.4.1 (a) below.

Table 3.4.1 (a)

Resource category	Northern Band	Middle Band	Southern Band	All total
Measured Mineral Resources (331)	12.476	3.810	2.293	18.579
Indicated Mineral Resources (332)	23.882	20.895	4.565	49.342
Inferred Mineral Resources (333)	12.472	2.305	1.619	16.396

Thereafter, the resultant mineable reserves and presently non-mineable resources has been derived out of resources after due consideration of the feasibility studies and with consideration of modifying factors (UPL, Statutory barriers ,pillars, geotechnical reasons, environmental/forest clearances etc.) for both underground and opencast. The distribution of mineable reserves and remaining resources (non-mineable part) for opencast and underground working as derived from Measured and Indicated resources are as follows:

Table 3.4.1 (b)

RESOURCES IDENTIFIED *(By degree of exploration)*				RESERVES AND REMAINING RESOURCES AFTER CONDUCTING FEASIBLITY STUDIES for O/C *(with consideration of modifying factors)*					
Resource category	Northern Band	Middle Band	Southern Band	Description	UNFC STD	Remarks	Northern Band	Middle Band	Southern Band
Measured Mineral Resources (331)	12.476	3.810	2.293	Above Ultimate Pit(O/C)	111	Open Cast	3.654	3.326	0.351
					211	Lease Boundary constraint	1.592	0.323	0.000
				Below Ultimate Pit (U/G)	121	Underground	0.000	0.000	0.174
					221	Soft Friable Ore	6.991	0.161	0.000
						Crown Pillar	0.241	0.000	1.693
						Sill & Rib Pillar	0.000	0.000	0.075
Indicated Mineral Resources (332)	23.882	20.895	4.565	Above Ultimate Pit(O/C)	122	Open Cast	1.345	2.424	0.000
					222	Lease Boundary constraint	0.579	0.841	0.000
				Below Ultimate Pit(U/G)	122	Underground	3.911	3.736	2.852
					222	Soft Friable Ore	12.181	7.611	0.000
						Crown Pillar	4.190	4.682	0.491
						Sill & Rib Pillar	1.676	1.601	1.222

Table 3.4.1 (c)
Resource of Chrome Ore as on 31.03.2012, (in Million Tonnes)

Ore Body	Category	UNFC Category	Grade Range, $\%Cr_2O_3$	Updated Resources (As on 31.03.2012)	
				Tonnage	Total
Northern Band	Proved Mineral Reserve - Open Cast	111	10-40	0.283	3.653
			40 - 52	1.588	
			>52	1.782	
	Probable Mineral Reserve - Open Cast	122	10-40	0.203	1.345
			40 - 52	0.863	
			>52	0.279	
	Open cast Feasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	211	10-40	0.314	1.591
			40 - 52	0.710	
			>52	0.567	
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	222	10-40	0.169	0.579
			40 - 52	0.372	
			>52	0.038	
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	333	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	221	10-40	0.171	6.990
			40 - 52	1.422	
			>52	5.397	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	222	10-40	1.723	12.180
			40 - 52	6.295	
			>52	4.162	
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	333	10-40	0.290	4.945
			40 - 52	1.857	
			>52	2.798	
	Prefeasibility Mineral Resource blocked in crown pillar	221	10-40	0.000	0.241
			40 - 52	0.067	
			>52	0.174	
	Prefeasibility Mineral Resource blocked in crown pillar	222	10-40	0.234	4.191
			40 - 52	2.229	
			>52	1.728	
	Inferred Mineral Resource blocked in crown pillar	333	10-40	0.134	2.671
			40 - 52	1.296	
			>52	1.241	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	221	10-40	0.000	0.000
			40 - 52	0.000	

			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.406	1.676
			40 - 52	0.903	
			>52	0.367	
	Proved Mineral Reserve - Underground	**111**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	0.947	3.910
			40 - 52	2.106	
			>52	0.857	
	Inferred Mineral Resource - Underground	**333**	10-40	0.540	4.855
			40 - 52	2.684	
			>52	1.631	
	TOTAL RESERVE + RESOURCES				**48.83**
Middle Band	Proved Mineral Reserve - Open Cast	**111**	10-40	2.027	3.326
			40 - 52	1.221	
			>52	0.078	
	Probable Mineral Reserve - Open Cast	**122**	10-40	1.604	2.425
			40 - 52	0.808	
			>52	0.013	
	Open cast Feasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**211**	10-40	0.299	0.322
			40 - 52	0.023	
			>52	0.000	
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**222**	10-40	0.753	0.841
			40 - 52	0.088	
			>52	0.000	
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	**333**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**221**	10-40	0.003	0.162
			40 - 52	0.126	
			>52	0.033	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**222**	10-40	2.027	7.611
			40 - 52	4.998	
			>52	0.586	
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	**333**	10-40	0.054	0.263
			40 - 52	0.208	
			>52	0.001	
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	

	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	1.703	4.682
			40 - 52	2.754	
			>52	0.225	
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.122	0.205
			40 - 52	0.083	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.686	1.601
			40 - 52	0.890	
			>52	0.025	
	Proved Mineral Reserve - Underground	**111**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	1.601	3.736
			40 - 52	2.076	
			>52	0.059	
	Inferred Mineral Resource - Underground	**333**	10-40	0.640	1.837
			40 - 52	1.177	
			>52	0.020	
	TOTAL RESERVE + RESOURCES				**27.01**
Southern Band	Proved Mineral Reserve - Open Cast	**111**	10-40	0.264	0.351
			40 - 52	0.087	
			>52	0.000	
	Probable Mineral Reserve - Open Cast	**122**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	1.328	1.692
			40 - 52	0.364	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	0.482	0.492
			40 - 52	0.010	
			>52	0.000	
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.070	0.075
			40 - 52	0.005	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in	**222**	10-40	1.145	1.222

	stope (rib/sill) pillar		40 - 52	0.077	
			>52	0.000	
	Proved Mineral Reserve - Underground	**111**	10-40	0.163	0.174
			40 - 52	0.011	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	2.671	2.851
			40 - 52	0.180	
			>52	0.000	
	Inferred Mineral Resource - Underground	**333**	10-40	1.530	1.620
			40 - 52	0.090	
			>52	0.000	
	TOTAL RESERVE + RESOURCES				**8.477**

The level wise reserve and resource for all three ore bodies is given in Table 3.4.1.1 to 3.4.1.3.

Table 3.4.1.1

NORTHERN BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Mining Method	From (mRL)	To (mRL)	Grade (% Cr_2O_3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable		Total	Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 211	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total		Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	**Sub Total**		
					A		B	**C=(A+B)**	D	E	F	G	**H=(D+E+F+G)**	I	J	K	L	**M=(I+J+K+L)**	N	O	P	Q	**R=(N+O+P+Q)**	**S=(H+M+R)**	**T=(C+S)**
Open Cast			+10 - 40						0.000																
	145	135	+40 - 52						0.000				**0.000**					**0.000**					**0.000**	0.000	**0.000**
			+52						0.000																
			+10 - 40						0.000																
	135	125	+40 - 52						0.009				**0.009**					**0.000**					**0.000**	0.009	**0.009**
			+52						0.000																
			+10 - 40						0.001																
	125	115	+40 - 52						0.027				**0.028**					**0.000**					**0.000**	0.028	**0.028**
			+52						0.000																
			+10 - 40						0.020																
	115	105	+40 - 52						0.041				**0.060**					**0.000**					**0.000**	0.060	**0.060**
			+52						0.000																
			+10 - 40	0.003					0.026																
	105	95	+40 - 52	0.003	**0.009**			**0.009**	0.074				**0.113**					**0.000**					**0.000**	0.113	**0.122**
			+52	0.002					0.012																
			+10 - 40	0.001					0.039																
	95	85	+40 - 52	0.007	**0.022**			**0.022**	0.072				**0.163**					**0.000**					**0.000**	0.163	**0.185**
			+52	0.014					0.051																
			+10 - 40	0.000					0.040																
	85	75	+40 - 52	0.008	**0.021**			**0.021**	0.090				**0.194**					**0.000**					**0.000**	0.194	**0.216**
			+52	0.014					0.065																
			+10 - 40	0.012		0.006			0.051																
	75	65	+40 -	0.037	**0.072**	0.006	**0.012**	**0.084**	0.085				**0.213**					**0.000**					**0.000**	0.213	**0.297**
			+52	0.022		0.000			0.077																
			+10 -	0.049		0.013			0.050					0.004											
	65	55	+40 -	0.062	**0.157**	0.033	**0.052**	**0.209**	0.082				**0.220**	0.006				**0.010**					**0.000**	0.230	**0.439**
			+52	0.046		0.006			0.087					0.000											
			+10 -	0.074		0.020			0.027					0.035											
	55	45	+40 -	0.215	**0.510**	0.105	**0.162**	**0.672**	0.072				**0.181**	0.026				**0.061**					**0.000**	0.242	**0.915**
			+52	0.221		0.037			0.083					0.000											
			+10 -	0.057		0.019			0.022					0.039											
	45	35	+40 -	0.359	**0.631**	0.182	**0.300**	**0.931**	0.070				**0.158**	0.057				**0.098**					**0.000**	0.255	**1.186**
			+52	0.215		0.099			0.065					0.001											

			+10 -	0.043		0.051			0.020					0.046											
	35	25	+40 -	0.378	**0.820**	0.251	**0.389**	**1.210**	0.055				**0.144**	0.093				**0.142**					**0.000**	0.286	**1.495**
			+52	0.400		0.087			0.069					0.003											
			+10 -	0.043		0.095			0.018					0.045											
	25	15	+40 -	0.497	**1.375**	0.282	**0.425**	**1.800**	0.033				**0.108**	0.189				**0.268**					**0.000**	0.376	**2.176**
			+52	0.834		0.048			0.057					0.034											
			+10 -	0.000		0.000			0.000					0.000											
	15	10	+40 -	0.021	**0.036**	0.003	**0.004**	**0.041**	0.000				**0.000**	0.000				**0.000**					**0.000**	0.000	**0.041**
			+52	0.015		0.001			0.000					0.000											
Sub Total			**49.13**		**3.654**		**1.345**	**4.999**	**1.592**	**0.000**	**0.000**	**0.000**	**1.592**	**0.579**	**0.000**	**0.000**	**0.000**	**0.579**	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	**2.171**	**7.170**
Underground			+10 -							0.087					0.224					0.011					
	15	0	+40 -							0.659			**1.861**		0.753			**1.381**		0.011			**0.025**	3.267	**3.267**
			+52							1.115					0.404					0.002					
			+10 -							0.084	0.000				0.651	0.165				0.147	0.000	0.003			
	0	-60	+40 -							0.755	0.009		**4.355**		2.685	0.617		**6.604**		0.555	0.008	0.014	**1.259**	12.218	**12.218**
			+52							3.471	0.035				1.862	0.624				0.512	0.020	0.001			
			+10 -			0.048				0.000	0.000				0.033	0.007	0.020			0.069	0.000	0.000			
	-60	-71	+40 -			0.038	**0.146**	**0.146**		0.005	0.016		**0.372**		0.368	0.093	0.016	**1.057**		0.165	0.022	0.006	**0.491**	1.921	**2.067**
			+52			0.060				0.320	0.032				0.448	0.047	0.026			0.226	0.003	0.001			
			+10 -			0.119				0.000	0.000				0.133	0.012	0.051			0.060	0.015	0.021			
	-71	-95	+40 -			0.172	**0.397**	**0.397**		0.000	0.031		**0.510**		0.678	0.185	0.074	**2.234**		0.231	0.019	0.022	**0.978**	3.722	**4.119**
			+52			0.106				0.425	0.053				0.848	0.209	0.045			0.560	0.047	0.002			
			+10 -			0.617				0.000	0.000				0.683	0.046	0.264			0.002	0.009	0.195			
	-95	-195	+40 -			1.464	**2.383**	**2.383**		0.003	0.011		**0.133**		1.811	1.335	0.627	**6.338**		0.828	0.668	0.382	**4.627**	11.098	**13.482**
			+52			0.302				0.065	0.053				0.601	0.840	0.130			1.414	0.773	0.359			
			+10 -			0.163									0.000	0.003	0.070			0.001	0.111	0.322			
	-195	-295	+40 -			0.432	**0.984**	**0.984**					**0.000**		0.000	0.000	0.185	**0.433**		0.066	0.579	1.605	**4.225**	4.658	**5.642**
			+52			0.389									0.000	0.007	0.167			0.083	0.399	1.058			
			+10 -																			0.000			
	-295	-381	+40 -										**0.000**					**0.000**				0.655	**0.866**	0.866	**0.866**
			+52																			0.211			
Sub Total			**45.49**				**3.911**	**3.911**	0.000	6.991	0.241	0.000	**7.231**	0.000	12.181	4.190	1.676	**18.04**	0.000	4.945	2.672	4.855	**12.47**	**37.750**	**41.661**
GRAND TOTAL								**8.909**	1.592	6.991	0.241	0.000	**8.823**	0.579	12.181	4.190	1.676	**18.62**	0.000	4.945	2.672	4.855	**12.47**	**39.921**	**48.830**

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Table No.. 3.4.1.2

MIDDLE BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Middle Band	From (mRL)	To (mRL)	Grade (%Cr2O3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable			Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 211	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	**Sub Total**	**Total**	**Total Resources**
					A		B	**C=(A+B)**	D	E	F	G	**H=(D+E+F+G)**	I	J	K	L	**M=(I+J+K+L)**	N	O	P	Q	**R=(N+O+P+Q)**	**S=(H+M+R)**	**T=(C+S)**
	170	160	+10 - 40	0.000																					
			+40 - 52	0.000	**0.000**			**0.000**					**0.000**					**0.000**					**0.000**	**0.000**	**0.000**
			+52	0.000																					
	160	150	+10 - 40	0.003																					
			+40 - 52	0.000	**0.004**			**0.004**					**0.000**					**0.000**					**0.000**	**0.000**	**0.004**
			+52	0.000																					
	150	140	+10 - 40	0.021					0.000																
			+40 - 52	0.007	**0.028**			**0.028**	0.000				**0.000**					**0.000**					**0.000**	**0.000**	**0.028**
			+52	0.000					0.000																
	140	130	+10 - 40	0.084					0.012					0.000											
			+40 - 52	0.034	**0.118**			**0.118**	0.000				**0.012**	0.000				**0.000**					**0.000**	**0.013**	**0.130**
			+52	0.000					0.000					0.000											
	130	120	+10 - 40	0.148		0.001			0.006					0.013											
Open Cast			+40 - 52	0.053	**0.201**	0.000	**0.001**	**0.201**	0.000				**0.006**	0.000				**0.013**					**0.000**	**0.019**	**0.220**
			+52	0.000		0.000			0.000					0.000											
	120	110	+10 - 40	0.176		0.007			0.020					0.025											
			+40 - 52	0.066	**0.242**	0.000	**0.007**	**0.249**	0.002				**0.022**	0.000				**0.025**					**0.000**	**0.048**	**0.297**
			+52	0.000		0.000			0.000					0.000											
	110	100	+10 - 40	0.195		0.019			0.043					0.031											
			+40 - 52	0.096	**0.291**	0.000	**0.019**	**0.310**	0.001				**0.043**	0.000				**0.031**					**0.000**	**0.075**	**0.384**
			+52	0.000		0.000			0.000					0.000											
	100	90	+10 - 40	0.189		0.044			0.047					0.039											
			+40 - 52	0.097	**0.286**	0.000	**0.044**	**0.330**	0.002				**0.049**	0.000				**0.039**					**0.000**	**0.088**	**0.418**
			+52	0.000		0.000			0.000					0.000											
	90	80	+10 - 40	0.165		0.073			0.048					0.049											
			+40 - 52	0.116	**0.282**	0.000	**0.074**	**0.356**	0.002				**0.050**	0.000				**0.049**					**0.000**	**0.099**	**0.455**
			+52	0.002		0.000			0.000					0.000											
	80	70	+10 - 40	0.228		0.107			0.040					0.051											
			+40 - 52	0.116	**0.355**	0.022	**0.131**	**0.486**	0.011				**0.051**	0.004				**0.055**					**0.000**	**0.106**	**0.592**
			+52	0.011		0.002			0.000					0.000											

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			+10 - 40	0.251		0.161			0.044					0.054											
	70	60	+40 - 52	0.156	**0.420**	0.050	**0.211**	**0.631**	0.002				**0.046**	0.011				**0.065**					**0.000**	**0.112**	**0.742**
			+52	0.013		0.000			0.000					0.000											
			+10 - 40	0.226		0.226			0.033					0.069											
	60	50	+40 - 52	0.152	**0.386**	0.113	**0.339**	**0.725**	0.001				**0.034**	0.014				**0.083**					**0.000**	**0.117**	**0.842**
			+52	0.008		0.000			0.000					0.000											
			+10 - 40	0.211		0.284			0.006					0.109											
	50	40	+40 - 52	0.145	**0.368**	0.153	**0.437**	**0.805**	0.000				**0.007**	0.009				**0.118**					**0.000**	**0.125**	**0.930**
			+52	0.012		0.000			0.000					0.000											
			+10 - 40	0.109		0.329			0.000					0.137											
	40	30	+40 - 52	0.106	**0.230**	0.210	**0.539**	**0.769**	0.000				**0.000**	0.016				**0.153**					**0.000**	**0.153**	**0.922**
			+52	0.015		0.000			0.000					0.000											
			+10 - 40	0.020		0.340			0.000					0.168											
	30	20	+40 - 52	0.074	**0.112**	0.246	**0.596**	**0.708**	0.001				**0.001**	0.033				**0.201**					**0.000**	**0.202**	**0.910**
			+52	0.017		0.010			0.000					0.000											
			+10 - 40	0.000		0.013			0.000					0.006											
	20	15	+40 - 52	0.002	**0.003**	0.012	**0.027**	**0.030**	0.000				**0.000**	0.002				**0.008**					**0.000**	0.008	**0.038**
			+52	0.001		0.001			0.000					0.000											
Sub Total			**37.59**		**3.326**		**2.424**	**5.750**	0.323	0.000	0.000	0.000	**0.323**	0.841	0.000	0.000	0.000	**0.841**	0.000	0.000	0.000	0.000	**0.000**	**1.163**	**6.914**
			+10 - 40							0.003					0.613	0.151									
	20	0	+40 - 52				**0.000**	**0.000**		0.096			**0.120**		0.751	0.050		**1.665**					**0.000**	**1.785**	**1.785**
			+52							0.021					0.100	0.000									
			+10 - 40			0.026				0.000					0.939	0.446	0.011			0.000	0.007	0.000			
	0	-51	+40 - 52			0.028	**0.056**	**0.056**		0.030			**0.041**		2.451	0.355	0.012	**4.636**		0.042	0.000	0.002	**0.050**	**4.728**	**4.784**
			+52			0.002				0.012					0.328	0.092	0.001			0.000	0.000	0.000			
			+10 - 40			0.152									0.386	0.514	0.065			0.036	0.012	0.010			
Underground	-51	-95	+40 - 52			0.239	**0.402**	**0.402**					**0.000**		1.042	1.083	0.102	**3.328**		0.125	0.002	0.020	**0.206**	**3.534**	**3.937**
			+52			0.011									0.080	0.051	0.005			0.001	0.000	0.000			
			+10 - 40			0.968									0.089	0.590	0.415			0.018	0.103	0.193			
	-95	-195	+40 - 52			1.393	**2.377**	**2.377**					**0.000**		0.753	1.048	0.597	**3.658**		0.041	0.068	0.441	**0.865**	**4.522**	**6.899**
			+52			0.016									0.077	0.083	0.007			0.000	0.000	0.000			
			+10 - 40			0.455										0.002	0.195				0.000	0.437			
	-195	-285	+40 - 52			0.416	**0.901**	**0.901**					**0.000**			0.218	0.178	**0.606**			0.013	0.714	**1.184**	**1.791**	**2.691**
			+52			0.030										0.000	0.013				0.000	0.020			
Sub Total			**40.96**				**3.736**	**3.736**	0.000	0.161	0.000	0.000	**0.161**	0.000	7.611	4.682	1.601	**13.894**	0.000	0.264	0.205	1.837	**2.305**	**16.360**	**20.096**
GRAND TOTAL								**9.486**	0.323	0.161	0.000	0.000	**0.484**	0.841	7.611	4.682	1.601	**14.735**	0.000	0.264	0.205	1.837	**2.305**	**17.524**	**27.010**

* Presently Non Mineable Resource

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Table No. 3.4.1.3

SOUTHERN BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Southern Band	From (mRL)	To (mRL)	Grade (%Cr2O3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable			Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 221	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	**Sub Total**		
					A		B	**C=(A+B)**	D	E	F	G	**H=(D+E+F+G)**	I	J	K	L	**M=(I+J+K+L)**	N	O	P	Q	**R=(N+O+P+Q)**	**S=(H+M+R)**	**T=(C+S)**
Open Cast	115	105	+10 - 40	0.000																					
			+40 - 52	0.000	**0.000**			**0.000**																0.000	**0.000**
			+52	0.000																					
	105	95	+10 - 40	0.003																					
			+40 - 52	0.002	**0.005**			**0.005**																0.000	**0.005**
			+52	0.000																					
	95	85	+10 - 40	0.005																					
			+40 - 52	0.010	**0.015**			**0.015**																0.000	**0.015**
			+52	0.000																					
	85	75	+10 - 40	0.004																					
			+40 - 52	0.016	**0.020**			**0.020**																0.000	**0.020**
			+52	0.000																					
	75	65	+10 - 40	0.028																					
			+40 - 52	0.006	**0.034**			**0.034**																0.000	**0.034**
			+52	0.000																					
	65	55	+10 - 40	0.018																					
			+40 - 52	0.001	**0.019**			**0.019**																0.000	**0.019**
			+52	0.000																					
	55	45	+10 - 40	0.036																					
			+40 - 52	0.005	**0.041**			**0.041**																0.000	**0.041**
			+52	0.000																					
	45	35	+10 - 40	0.128																					
			+40 - 52	0.036	**0.164**			**0.164**																0.000	**0.164**
			+52	0.000																					
	35	30	+10 - 40	0.043																					
			+40 - 52	0.010	**0.053**			**0.053**																0.000	**0.053**
			+52	0.000																					
Sub Total			34.85		**0.351**			**0.351**					**0.000**					**0.000**					**0.000**	**0.000**	**0.351**
ergr oun	175	105	+10 - 40								0.242														
			+40 - 52		**0.000**			**0.000**			0.027		**0.269**											0.269	**0.269**
			+52								0.000														

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		+10 - 40	0.025		0.002					0.091	0.011					0.001								
105	89	+40 - 52	0.000	**0.025**	0.000	**0.002**	**0.027**			0.034	0.000	**0.135**				0.000	**0.001**						0.136	**0.163**
		+52	0.000		0.000					0.000	0.000					0.000								
		+10 - 40	0.107		0.063					0.235	0.046					0.027								
89	35	+40 - 52	0.007	**0.113**	0.000	**0.063**	**0.176**			0.127	0.003	**0.411**				0.000	**0.027**						0.438	**0.614**
		+52	0.000		0.000					0.000	0.000					0.000								
		+10 - 40	0.006		0.013					0.511	0.003				0.068	0.005					0.001			
35	0	+40 - 52	0.004	**0.011**	0.000	**0.013**	**0.023**			0.157	0.002	**0.672**			0.002	0.000	**0.075**				0.000	**0.001**	0.749	**0.772**
		+52	0.000		0.000					0.000	0.000				0.000	0.000					0.000			
		+10 - 40	0.012		0.115					0.249	0.005				0.414	0.049					0.028			
0	-31	+40 - 52	0.000	**0.012**	0.007	**0.122**	**0.135**			0.020	0.000	**0.274**			0.007	0.003	**0.474**				0.000	**0.028**	0.777	**0.911**
		+52	0.000		0.000					0.000	0.000				0.000	0.000					0.000			
		+10 - 40	0.013		1.613						0.006					0.691					0.225			
-31	-95	+40 - 52	0.000	**0.013**	0.151	**1.765**	**1.778**				0.000	**0.006**				0.065	**0.756**				0.000	**0.225**	0.987	**2.765**
		+52	0.000		0.000						0.000					0.000					0.000			
		+10 - 40			0.866											0.371					1.268			
-95	-195	+40 - 52		**0.000**	0.022	**0.887**	**0.887**					**0.000**				0.009	**0.380**				0.089	**1.357**	1.737	**2.624**
		+52			0.000											0.000					0.000			
		+10 - 40																			0.007			
-195	-206	+40 - 52		**0.000**		**0.000**	**0.000**					**0.000**					**0.000**				0.001	**0.008**	0.008	**0.008**
		+52																			0.000			
Sub Total		32.5		**0.174**		**2.852**	**3.026**	0.000	0.000	1.693	0.075	1.767	0.000	0.000	0.491	1.222	1.714	0.000	0.000	0.000	1.619	1.619	5.100	**8.126**
GRAND TOTAL							**3.377**	0.000	0.000	1.693	0.075	**1.767**	0.000	0.000	0.491	1.222	**1.714**	0.000	0.000	0.000	1.619	**1.619**	**5.100**	**8.477**

* **Presently Non mineable**

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SUMMARY OF RESOURCES:

The summary of the mineral reserve & remaining resources that can be exploited by both opencast as well as underground means is given below in Table 3.4.2.

Table 3.4.2 (a)
Summary Of Chrome Ore Reserves & Resources As on 01.04.2012 (In Million tonnes)

Ore Body	Method of Mining	Reserves						Remaining Resources					Total Resources	
		Proved		Probable		Total Reserves		Prefeasibility		Inferred	Total Remaining Resources			
		STD 111		STD 122				STD 211 &221	STD 222	STD 333				
		A		B		C=(A+B)		D	E	F	G=(D+E+F)		H= (C+G)	
Northern Band	Open cast	3.654	**3.654**	1.345	5.256	4.999	**8.909**	1.592	0.579	0.000	2.171	**39.921**	7.170	**48.830**
	Under ground	0.000		3.911		3.911		7.231	18.047	12.472	37.750		41.661	
Middle Band	Open cast	3.326	**3.326**	2.424	6.160	5.750	**9.486**	0.323	0.841	0.000	1.163	**17.524**	6.914	**27.010**
	Under ground	0.000		3.736		3.736		0.161	13.894	2.305	16.360		20.096	
Southern Band	Open cast	0.351	**0.525**	0.000	2.852	0.351	**3.377**	0.000	0.000	0.000	0.000	**5.100**	0.351	**8.477**
	Under ground	0.174		2.852		3.026		1.767	1.714	1.619	5.100		8.126	
GRAND TOTAL	Open cast	**7.331**	**7.505**	**3.769**	**14.267**	**11.100**	**21.773**	**1.914**	**1.420**	**0.000**	**3.334**	**62.545**	**14.435**	**84.317**
	Under ground	**0.174**		**10.498**		**10.672**		**9.160**	**33.655**	**16.396**	**59.211**		**69.883**	

The above reserves and resources as per UNFC standards are given in Table 3.4.2 (b) below.

Table No. 3.4.2 (b)
Total Chromite Resources as Per UNFC

	Category/Code	Total	Description	Quantity in Million tonnes (+10% Cr_2O_3)
Reserves (a)	Proved **(111)**	7.331	Open Cast	7.331
			Underground	0
	Probable **(121)**	0.174	Open Cast	0
			Underground	0.174
	Probable **(122)**	14.267	Open Cast	3.769
			Underground	10.498
Remaining Resources (b)	Feasibility Resources **(211)**	1.914	Open Cast	1.914
	Pre-feasibility Resources **(221)**	9.160	Underground	9.160
	Pre-feasibility Resources **(222)**	35.075	Open Cast +Underground	35.075
	Measured Resources **(331)**	--		
	Indicated Resources **(332)**	--		
	Inferred Resources **(333)**	16.396	Underground	16.396
Total Resources (a+b)		**84.317**		84.317

Pyroxenite Reserve:

The proved mineral resource of pyroxenite was 37.98 lakh tonnes as on 31.03.08. During the year 2008-09, 2009-10 total 5.20 lakh tonnes of pyroxenite ROM was mined. So the updated pyroxenite reserve as on 01.04.2010 was 32.78 Lakh tonnes.

Based on the geological mapping data & the boreholes drilled for pyroxenite, the Pyroxenite body was remodeled using Surpac software. A solid body model of the Pyroxenite was created. Considering the proximity of the tailing dam the ultimate pit was redesigned upto 73mRL. From the block model, the different UNFC categories of Pyroxenite resource was calculated using a fixed specific gravity of 2.5. The details of Pyroxenite resource is tabulated below in Table 3.4.3:

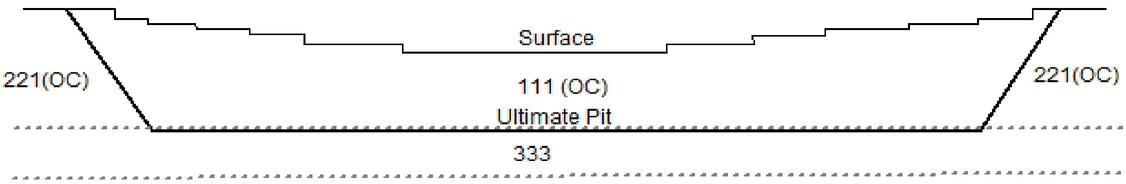


Table 3.4.3
Updated Reserve & Resources for Pyroxenite
(as on 31.03.2012)

Reserves					Remaining Resources					Total Resources
Proved		Probable		**Total (Million Tonnes)**	Prefeasibility		Inferred		**Total**	
STD 111		STD 122			STD 221		STD 333			
Million Tonne	Total	Million Tonne	Total		Resource blocked in boundary	**Sub Total**	Inferred Resource	**Sub Total**		
	A		B	**C=(A+B)**	D	**E**	F	**G**	**H = E+G**	**I = C+H**
3.080	**3.080**			3.080	5.74	5.74	3	3	8.74	**11.820**

Other Resources:

The stock of resources estimated within mineralized stacks of LG friable ore and lumpy ore together with mineral rejects in form of fines of lumpy ore, pyroxenite ore, as well as tailings stored in the tailing pond and other areas as on 01.04.2012 is given below in Table 3.4.4.

Table: 3.4.4

Sl No.	Type of Material	Tonnage (Lakh MT)	Location
1	LG Ore (Friable)	3.00	LG Dump (2),(6)
2	LG Ore (Lumpy)	14.00	LG Dump (4),(4A)
3	Mineral Rejects (Lumpy Fines)	1.20	MR (8), MR (Lumpy fines)
4	Mineral Rejects (Pyroxenite Fines)	0.75	MR (Pyrox fines) in 1(A) dump
5	Tailings	75.00	In the Tailing pond
6	Total	93.95	

3.5 EXPLORATION PROPOSED DURING 2011-12 to 2012-13:

The morphology of the Chromite ore body as existent in Sukinda valley has been thoroughly studied during the long exploration history of Sukinda. Sukinda Chromite deposit is a stratiform, stratabound and Tabular deposit of regular habit. This observation has also been made by GSI in their Detailed Information Dossier on Chromite Ore in India, January, 2008 (Chapter2, pg-9) where they have clearly mentioned that **"The Chromite deposit of Sukinda belt is mainly stratiform type"**. Chromite body exists in a continuous stretch intermittently intercepted by dykes. Within the leasehold area, these intercepting dykes (four in numbers) occur naturally and cut across the orebody at 4 places in the Friable bands and penetration through them has not disturbed the neighbouring ore morphology nor it is expected that sudden dykes will appear throughout the stretch which is evident from the present exploration. The nature of the dyke in the Friable bands are well established. There is no apprehension of sudden occurrences of dyke as seen during exploration. The drilling results also support the correlation.

The entire leasehold area has been well explored although more emphasis has been given for detailed exploration for three established chromite bands. Ore body being tabular / vertical detailed exploration at depth has been done. To rule out occurrences of other ore bodies , exploration has been done in the surrounding quarry areas laterally. This apart, Hydrogeological, Geotechnical and Infrastructure holes have been drilled. We have not detected presence of any other mineral / orebody through these holes were made. Geophysical study on "Locating Probable Groundwater zones around Lease hold area of Tata Steel" conducted by Faculties from IIT, Kharagpur does not indicate presence of any other ore body except the three being currently discussed. GSI has also found and taken only these three bands into consideration while conducting their study in this belt / lease-hold area from time to time through Geological mapping and other spectral data.

As may be observed from the updated surface geological plan, all the ore bands have been explored by drilling mostly inclined boreholes to cover the total strike length from extremities in the west to the east of the lease hold. As is evident from the updated longitudinal geological section, the depth of proving for the opencast mining upto the ultimate pits limits has been more than adequate with a closely spaced grid interval of 50 – 100m. Most of the mineable reserves for opencast workings are in the proved category in all the three ore bodies. In view of the dwindling resources of chrome ore for opencast mining, the company has since long taken up further core drilling program to assess the resources at greater depths. Feasibility studies for underground mining have been completed based on the resources so established.

So far till 01.04.2012 the deepest intersections in the northern, middle and southern ore bands have been (-) 375, (-) 235 & (-) 175 mRL respectively. The average depth upto which the resources have already been explored and brought to the indicated (G2) category has been (-) 150, (-) 190 & (-) 125 mRL respectively.

The future exploration has been conceptualized to further prove the depth persistency of all the three orebodies and progressively bring the inferred or (G3) resources to the (G2) category by way of drilling more number of boreholes from the surface. The remaining gaps in exploration of the orebodies have been identified. Accordingly the future proposals of drilling deeper boreholes from the surface to fill these gaps and also further explore the depth persistency during the next 5 years have been proposed, to be in consonance with MOM directives for conditions under 27(3) of MCR, 1960. Considering the near vertical nature of the orebodies, drilling to still greater depths from the surface would require the drilling to be done at fairly steep angles due to lack of space laterally. Moreover, there is also a likelihood of missing the orebody while attempting to drill very long holes at steep angles. Accordingly, the maximum depths of such surface boreholes have been conceptualized. The deepest intersections proposed for the northern, middle and southern ore bands are (-) 450, (-) 450 & (-) 400 mRL respectively. Further, in order to bring the indicated (G2) category of resources at the deeper horizons to the measured (G1) category of resources, the exploratory drilling at lateral intervals of 50-100m and 30-50m vertical depth has be proposed to be undertaken from the proposed underground haulage drives/ drifts. The targeted intersection points for the proposed boreholes have been indicated in the longitudinal geological section for all three orebodies.

Besides the above, boreholes for infrastructure development of the major underground infrastructures like declines and vertical shafts have also been planned to be done at the earliest.

There was no proposal for further exploration for the period 2010-13 in the approved Scheme of Mining. A further proposal for carrying out 8360m of core drilling during the period 2012-13 is being made through this modification to the Scheme of Mining to take care of the following:

- For finalization of detailed design of major underground mine infrastructures like shaft & declines
- Infill drilling in Middle Band to increase confidence level of resource & bring the probable reserve to proved category for underground mining in the Southern Band (OB-II) up to a depth of 185m below the surface.

The details of the proposed drilling have been given in Table 3.5.1. The location of the proposed boreholes has been indicated in drawing No. SCM/MP/03/12. The details of proposed surface exploratory boreholes mentioned above are given in Table 3.5.2.

Table 3.5.1
Year wise Exploration Programme (2012-13)

Year	Proposed Boreholes			
	Infrastructural	Exploratory	Total Nos.	Meterage
2012-13	15	17	32	8360
TOTAL	15	17	32	8360

Table 3.5.2

Details of exploratory boreholes (2012-13)

Year	Type	Ore Body	BH No	Northing	Easting	RL	Planned Level Of Intersection, mRL	Dip	Azimuth	Appx. Length, M
2013-14	Exploration	Northern Band	PBH33	2717	2990	124	-300	68	180	500
			PBH34	2333	2000	119	-350	59	0	575
			PBH35	2800	2100	114	-350	61	180	575
			PBH36	2709	3700	134	-350	66	180	600
			PBH37	2704	3900	137	-350	73	180	575
		Middle Band	PBH38	2464	3900	72	-300	60	180	475
			PBH39	2407	3494	63	-300	66	180	450
			PBH40	2443	3110	48	-300	67	180	450
			PBH41	2408	2900	61	-300	73	180	500
			PBH42	2441	3300	53	-350	58	180	525
		Southern Band	PBH43	1464	3250	121	-200	49	180	450
			PBH44	1477	3300	121.6	-200	73	180	475
			PBH45	1350	3450	56.11	-225	57	180	400
			PBH46	1540	3500	131.5	-200	55	180	525
			PBH47	1517	3700	130.4	-200	59	180	400
			PBH48	1513	4006	147.3	-200	63	180	500
			PBH49	1428	3150	121	-300	66	180	500
			PBH50	1516	3750	131	-300	65	180	600
			PBH51	1513	3897	136	-350	66	180	600
			PBH52	1534	3595	133	-300	68	180	600
									Total	**10275**
2014-15	Exploration	Northern Band	PBH53	2728	2805	123	-400	70	180	625
			PBH54	2926	1900	112	-450	61	180	700
			PBH55	2746	2600	121	-450	66	180	650
			PBH56	2716	3700	136	-450	70	180	650
			PBH57	2718	4000	136	-450	77	180	675
			PBH58	2757	3200	138	-400	72	180	800
		Middle Band	PBH59	2542	1800	70	-400	68	180	475
			PBH60	2442	2503	57	-400	66	180	550
			PBH61	2453	2700	43	-450	65	180	475
			PBH62	2411	3702	64	-450	70	180	625
			PBH63	2482	2000	63	-400	73	180	500
		Southern Band	PBH64	1534	3500	131	-400	69	180	700
			PBH65	1508	4000	147	-400	72	180	650
			PBH66	1433	3050	121	-400	75	180	700
									Total	**8775**

As per UNFC guide lines borehole interval is permissible upto 50-100m in case of Measured resource and upto 300 m in case of Indicated / inferred resource. Presently, the reserve submitted to IBM for underground mining are geologically dominantly in indicated category (G2) and therefore drilling at 30/50 m interval is not essential right now. Close interval drilling will however be done when underground opening will start to bring the resource into G1 category.

The proposed boreholes have been shown in drawing no SCM/MP/03/12. Since most of the boreholes are proposed to be drilled within the active quarry, the locations may change marginally to suit the site conditions within the quarry.

4.0 CONCEPTUAL MINING PLAN

Present Method of Mining and allied activities:

Mining:

Presently conventional opencast method of mining with shovel-dumper combination is being practiced. As there is no further space for disposal of the overburden/ associated minerals (consisting of nickelliferous limonite, serpentinite and quartzite) within the mining lease of 406 ha, the same is being stacked within the additional area of 100 ha that has been allotted for the purpose (Ref. para 2.4) since early 2006. As per conditions stipulated in the approval for its forest clearance, this additional area was released in phases of 25 ha each and so far 75 ha has already been released and almost fully saturated. We have applied for the balance 25 ha to the concerned authority and the same is expected to be obtained shortly.

The rate of production from the respective pits is decided on the basis of the available grades of ore (physical & chemical characteristics) and its end use requirements and plant capacities.

Deep hole drilling and controlled blasting techniques like presplit blasting, use of Nonel, SME (Site Mixed Emulsion) is being practiced to minimize vibration, back breaks and fly rock generation.

Based on geotechnical studies on slope stability conducted by CMRI, Dhanbad and SRK, South Africa approval was obtained from DGMS vide letter no. BCU/CH-7/P-106/2005, dated 03.07.2006 (enclosed as Annexure-18) for a permitted maximum overall pit slope angle of 35 degree in OB-X quarry and 45 degree in OB-II quarry. While the recommendations of CMRI, Dhanbad was same as the above permitted overall slope angles, the recommendations of SRK after a more detailed slope stability analysis was 5 degrees higher than the above permitted overall slope angles. Copy of the relevant portions of the slope stability analysis report of SRK, South Africa is enclosed as Annexures-19. The relevant portions of the CMRI report of March 2000 & SRK report of November 2004 for recommended pit slope dimensions is enclosed as Annexure-20 & 19 respectively. Accordingly, the ultimate pit depth had been conceptualized as (–) 25 mRL in OB-X quarry and 25 mRL in OB-II quarry as per approved Scheme of Mining.

Regular slope monitoring through precise level surveys is being carried out both departmentally as well as with the assistance of CIMFR, Dhanbad to ensure the safety of the opencast workings. Horizontal holes (length- 50 m) are being drilled at intervals along the entire strike length of the lower benches to dewater and depressurize the slopes and improving its stability condition. The depth of the water table is also being monitored with the aid of piezometers that have been installed for the purpose. High capacity electric pumps that have been mounted on pontoons have been provided to effectively dewater the pits. The present extents of the quarries are given in Table No. 4.1.1 below.

The stripping ratios as per the approved Scheme of Mining for the period 2008-13 was planned to be:

Friable Ore band: 5.12
Lumpy Ore band: 7.53

Table No. 4.1.1
Extent of Present Workings

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL				No. of benches								Overall slope angle, degrees							
			North side		South side		Northern Band		Middle Band		Northern Band				Middle Band				Northern Band				Middle Band			
											North side		South side		North side		South side		North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-X	2503	572	124.3	139	125	204	20	52	52	85	13	15	6	9	5	7	5	11	23	33	18	28	15	21	16	24

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL		No. of benches				Overall slope angle, degrees			
			North side		South side		Min.	Max.	North side		South side		North side		South side	
			Min.	Max.	Min.	Max.			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-II	1151	449	118	132	152	230	33	45	12	15	16	21	22	33	24	32
Pyroxenite	329	207	136	146	123	132	102	107	5	7	5	7	17	22	15	21

However as mentioned in the Review chapter, the above planned stripping ratios could not be partially achieved. This was mainly due to higher operational losses. The availability of the dumpers was lower than the plan and was adversely impacted due to the increasing depth, gradients and lead distance of the mine. But more importantly, the operational efficiency and cycle time was affected as a result of cueing of the dumpers due to the restricted bench widths, space available in the dumping grounds and slower speeds of the dumpers due to traffic congestion. Such constraints in the space for overburden dumping arose as the area was released in a phased manner of 25 ha only as per the condition no. 11 stipulated in the approval letter for forest clearance (Ref. to Annexure-21). This also led to difficulties in stacking of the different types of associated minerals separately. Although the company had time and again requested the MoEF siting the above constraints for the modification of such condition, no clear order was passed. However, despite the same the overall slope angles of the pits as evident from the supporting transverse sections of the pits enclosed and enumerated above under Table No. 4.1, are still well within the maximum permitted slope angles approved by the DGMS. The safety of the mine and operations have therefore not been jeopardized.

Mineral Beneficiation:

Company has installed a chrome ore beneficiation plant since 1991 which is based on gravity separation principle. The various processes in beneficiation process involve feed preparation by crushing in jaw crusher and cone crusher, grinding in ball mill, removal of gangue using scrubber. In the beneficiation circuit hydrocyclones, screw classifiers, floatex density separators to recover the finer concentrate and to segregate the product based on size. The process water is recycled back into the process after treatment in thickener. The underflow of thickener is discharged into the tailing pond; clean water from the same is recycled back and reused in the COB plant.

Similarly ROM from OB-II quarry is processed at LOPP and N Kumar Plant and +10mm marketable grade material is shifted to stackyard for onward dispatch to various customers through road. The -10mm fines material is stacked separately for future use.

Environment Management:

Presently ambient air, noise and water quality is being monitored twice a week at six locations within the core zone and ambient air quality monitoring is done once in a quarter in 10 villages in buffer zone. Water quality within the core zone is monitored at ETP, STP, WTP, tube wells and oil & grease separation pit.

Plantation is being taken up in dumps progressively after it reaches its ultimate capacity as per the method stated in the approved scheme of mining. Adequate measures are being taken towards post plantation care, watering, watch and ward. Many a places the dump slopes are being coir matted using biodegradable geotextiles to prevent erosion and formation of rain cuts and gullies.

Conceptual Exploration:

As may be observed from the updated surface geological plan, all the ore bands have been explored by drilling mostly inclined boreholes to cover the total strike length from extremities in the west to the east of the lease hold. As is evident from the updated longitudinal geological section, the depth of proving for the opencast mining upto the ultimate pits limits has been more than adequate with a closely spaced grid interval of 50-100m. Most of the mineable reserves for opencast workings are in the proved category in all the three ore bodies. In view of the dwindling resources of chrome ore for opencast mining, the company has since long taken up further core drilling program to assess the resources at greater depths. Feasibility studies for underground mining have been completed based on the resources so established.

So far till 01.04.2012 the deepest intersections in the northern, middle and southern ore bands have been (-) 375, (-) 235 & (-) 175 mRL respectively. The average depth up to which the resources have already been explored and brought to the indicated (G2) category has been (-) 150, (-) 190 & (-) 125 mRL respectively.

The future exploration has been conceptualized to further prove the depth persistency of all the three ore bodies and progressively bring the inferred or (G3) resources to the (G2) category by way of drilling more number of boreholes from the surface. The remaining gaps in exploration of the ore bodies have been identified. Accordingly the future proposals of drilling deeper boreholes from the surface to fill these gaps and also further explore the depth persistency during the next 5 years have been proposed, to be in consonance with MOM directives for conditions under 27(3) of MCR, 1960. Considering the near vertical nature of the ore bodies, drilling to still greater depths from the surface would require the drilling to be done at fairly steep angles due to lack of space laterally. Moreover, there is also a likelihood of missing the ore body while attempting to drill very long holes at steep angles. Accordingly, the maximum depths of such surface boreholes have been conceptualized. The deepest intersections proposed for the northern, middle and southern ore bands are (-) 450, (-) 450 & (-) 400 mRL respectively. Further, in order to bring the indicated (G2) category of resources at the deeper horizons to the measured (G1) category of resources, the exploratory drilling at lateral intervals of 50-100m and 30-50m vertical depth has be proposed to be undertaken from the proposed underground haulage drives/ drifts. The targeted intersection points for the proposed boreholes have been indicated in the longitudinal geological section for all three orebodies.

Besides the above, boreholes for infrastructure development of the major underground infrastructures like declines and vertical shafts have also been planned to be done at the earliest.

Conceptual Method of Mining and allied activities:

Opencast Mining:

While the general method of opencast mining shall remain the same, a change in the design parameters of the benches, ultimate pit bottom limits and conceptual plan as per this modification to the Scheme of Mining & subsequent Mining Plan is enumerated below.

Conceptual mining plans have been prepared to cover the life of the mine as per the present scenario and have been shown in Drawing no.'s SCM/MP/05/12. The conceptual mining plan is developed considering the long-term future requirements of ore for the end users and the method of Mining to be followed. Considering the reserves, it is envisaged that this requirement shall be fully met from the existing opencast operations. The open pits are designed from the bottom upwards to arrive at the pit limits and accordingly year-wise excavation plans are developed.

It may be observed that the design of pit slopes for OB-II quarry as recommended by CMRI, Dhanbad and SRK, South Africa was meant to have a lower bench width than the bench height in order to achieve the recommended ultimate pit slope of 45^0 (enclosed as Annexures 19 & 20). However, the DGMS permission letter for relaxation under Reg. 106 2(b) under MMR, 1961 (enclosed as Annexure-18) stipulates that the overall pit slope shall not exceed 45^0 and the bench width shall not be less than the bench height. In accordance with the said order of the Director of Mines Safety, the ultimate pit for OB-II quarry has been redesigned. The bench widths for the future mining to be undertaken have been now considered to be same as the bench height. The conceptual ultimate pit design has been shown in the Conceptual Plan and Conceptual Transverse Section (Drawing No.'s SCM/MP/05/12 & SCM/MP/05A/12). With this the overall pit slope angle of the ultimate pit shall be less than 45^0. With this modification, the ultimate pit bottom depth of OB-II quarry (after providing for a sump for dewatering purposes) is 30 mRL and no longer 25 mRL as proposed earlier in the approved Scheme of Mining.

Standalone, the opencast mining limits is typically constrained by the following factors:

a) The available reserves and resources,
b) The lease boundary limits for the lateral extension of the pit top,
c) The ultimate safe & permitted maximum overall pit slope angles for the depth extension of the pits
d) The stripping ratios and the economic viability with respect to the lead distances and depth extension of the pits
e) The expected rate of generation of overburden and the holding capacity of the area available for its disposal.
f) The balance areas that shall be available within the lease after the future extension of the pits for providing for the other allied activities viz. mineral and mineral rejects storage, future infrastructures etc.

The major part of the overburden to be generated in the course of future mining of the chrome ore by opencast method is going to be from the OB-X quarry. Further, considering the dwindling reserves of the low/ beneficiable grade ores which is mostly available from the middle friable ore band, it was felt necessary to rehandle a part of the overburden dump (Dump No. 2A) lying in the south eastern part of the OB-X quarry in order to increase the ultimate pit depths of the mid band area.

The balance holding capacity within the 100 ha allotted for disposal of the overburden as on 01.04.2011 is only 178 lakh cum. The estimates for further overburden generation from the opencast workings are given in the Table no. 4.1.2 below.

Table No. 4.1.2

Estimates for further overburden generation upto ultimate pit bottom limits

(as on 01.04.2011)

Quarry/ Section	Ultimate Pit bottom Depth, mRL	Volume of OB, lakh cum
OB-II	30	24
OB-X	(-)25	337
OB-X (addl. from deepened mid band section)	15	197
Rehandling of OB from the SE old OB dump		26
Total		584

It is evident from the above that the constraints in the area for overburden disposal and the area required for other allied activities and future infrastructure development emerges as the predominant factor for deciding the ultimate pit bottom limits.

Thus, considering the dwindling resources of OB-II quarry it has been first conceptualized to exhaust the mineable reserves and then resort to backfilling of the OB-II pit with the overburden to be generated from the OB-X quarry. Thereafter it is proposed to exhaust the middle ore band section of the OB-X quarry to backfill the same with the remaining overburden to be generated from the northern band of the OB-X quarry.

The holding capacity of the OB-II quarry area and mid band section of the OB-X quarry by way of backfilling has been estimated as given in Table 4.1.3 below.

Table 4.1.3

Estimates for holding capacity for overburden by backfilling of the pits

Quarry	Backfill holding capacity, lakh cum
OB-II	272
OB-X (Mid band section upto 70 mRL)	60
Total	332

The net holding capacity for the overburden including the remaining area within the allotted 100 ha for overburden dumping and the backfilling opportunities within the exhausted pits is ~ 509 lakh cum.

It is evident from the above, that the volume of the overburden to be generated (584 lakh cum) far exceeds the holding capacities of the remaining area of 100 ha and the pits/ sections proposed to be backfilled.

In view of the above constraints, it is now conceptualized as per this modification to the Scheme of Mining and Mining Plan to optimize the future overburden generation from the north band section of the OB-X quarry in order to accommodate the same by way of backfilling within the OB-II and still later in the mid band sections of the OB-X quarry.

Accordingly, the ultimate pit for OB-X quarry has been re-designed and the proposed ultimate pit bottom of the north band section of the OB-X quarry is now 10 mRL (after providing for the sump for dewatering purposes). This would also allow for some space that would be required in future for accommodating the waste from the future underground development work. With this change the revised estimates for OB generation shall be

Quarry/ Section	Ultimate Pit bottom Depth, mRL	Volume of OB, lakh cum
OB-II	30	24
OB-X	10	209
OB-X (addl. from deepened mid band section)	15	197
Rehandling of OB from the SE old OB dump		26
Total		456

The manner in which the overburden dumping shall be done during the balance Scheme of Mining period is shown in Drawing No.'s SCM/MP/08/12.

The manner in which the backfilling of the pits shall be done up to the end of the mining plan period of 5 years after this scheme period and thereafter at intervals of every 5 years till the end of life of the open pit operations have been indicated in the drawings for Conceptual Plan & Conceptual Section (Drawing no.s SCM/MP/05/12 & SCM/MP/05A/12)

No backfilling is however being contemplated in the northern band section of the OB-X quarry at this stage. This is with the view of allowing for further studies to be undertaken for establishment of any novel methods of mining the remaining resources of the higher grades of soft friable chrome blocked between the presently defined ultimate pit limits and the crown pillar proposed to be left in the hard & competent ore for future underground mining as shown in the longitudinal geological section.

As there is no change in the prevailing hydro geological condition and monsoon pattern no change is being proposed in the dewatering mechanism of the quarry.

The chrome ore deposit is dipping almost vertically. The grade of chrome ore changes both in the strike and dip direction as well as with depth. In order to cater to the stringent chemical specifications of the user industries, it is necessary to mine all along the entire strike direction to meet the grade specifications for its end use. The specifications of the user industries are thereby met by suitable blending.

Stripping Ratio: Consequent to the changes in the ultimate pit designs as well as updating of the geological sections as on 01.04.2012 as stated above, a re-assessment of the reserves & resources for opencast mining as per the guidelines of the UNFC was done. Similarly, based on the fresh estimates of the overburden to be generated for future opencast mining upto the proposed ultimate pit limits and rehandling of the old dump no. 2A, the revised stripping ratio for the quarries as on 01.04.2012 are given below:

OB-X quarry: 3.78
OB-II quarry: 6.88

From the above it is evident that the overall stripping ratio of 4.44 at OB-X quarry achieved during the period 2008-11 (as mentioned in the Review chapter), was thus more than adequate for its development. The year-wise mine development plan from 2011-12 onwards has accordingly been conceptualized and modified with the above stripping ratio of 3.78 for OB-X quarry.

The estimates for the stripping ratio of OB-II quarry has come down from 7.53 to a level of 6.88 as per this modification. The overall stripping ratio achieved during the period 2008-11 was 5.85 which were marginally lower than the planned overall stripping ratio of 6.32 during the above period. The year-wise mine development plans from 2011-12 onwards has accordingly been conceptualized and modified with the above stripping ratio of 6.88 for OB-II quarry.

The backfilling of the OB-II quarry has been conceptualized to begin from 2014-15 after the external dumping area of 100 ha is saturated/ full. This backfilling during 2014-15 of the Mining Plan period shall be done between the sections 3100E to 3600E as shown in the Drawing No. SCM/MP/07/12. The OB-II quarry at its finishing stages of mining has also been proposed to be carried out during this year. The mining has been conceptualized and accordingly planned to be restricted between the sections 3650E to 4000E and thus synchronized with the backfilling to be undertaken simultaneously during this year. A safety margin of 50m has been left between the proposed backfill and mining areas. Further care shall be taken to do the backfilling from the bottom upwards in a retreating manner.

Though the mining operation shall not attain its ultimate pit depth during the plan period till 2013, the ultimate pit dimension as envisaged at the end of opencast mining is given below in Table 4.1.4(a) & 4.1.4 (b).

Table 4.1.4 (a)
Ultimate Pit Dimension of OB-X Quarry

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL			
			North side		South side		Northern Band		Middle Band	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-X	2503	615	114	137	120	225	10	30	15	40
			No. of benches							
			Northern Band				Middle Band			
			North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
			14	16	6	9	5	7	5	21
			Overall slope angle, degrees							
			Northern Band				Middle Band			
			North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
			30	35	34	35	27	35	35	35

Table 4.1.4 (a)
Ultimate Pit Dimension of OB-II Quarry & Pyroxenite Section

<table>
<tr><th rowspan="3">Quarry</th><th rowspan="3">Avg. Length (m)</th><th rowspan="3">Avg. Breadth (m)</th><th colspan="4">Pit top, mRL</th><th colspan="2">Pit bottom, mRL</th></tr>
<tr><th colspan="2">North side</th><th colspan="2">South side</th><th rowspan="2">Min.</th><th rowspan="2">Max.</th></tr>
<tr><th>Min.</th><th>Max.</th><th>Min.</th><th>Max.</th></tr>
<tr><td>OB-II</td><td>1151</td><td>449</td><td>118</td><td>132</td><td>152</td><td>230</td><td>30</td><td>35</td></tr>
<tr><td>Pyroxenite</td><td>467</td><td>224</td><td>136</td><td>145</td><td>123</td><td>129</td><td>73</td><td>73</td></tr>
<tr><td rowspan="3">Quarry</td><td colspan="4">No. of benches</td><td colspan="4">Overall slope angle, degrees</td></tr>
<tr><td colspan="2">North side</td><td colspan="2">South side</td><td colspan="2">North side</td><td colspan="2">South side</td></tr>
<tr><td>Min.</td><td>Max.</td><td>Min.</td><td>Max.</td><td>Min.</td><td>Max.</td><td>Min.</td><td>Max.</td></tr>
<tr><td>OB-II</td><td>12</td><td>15</td><td>16</td><td>21</td><td>30</td><td>33</td><td>35</td><td>40</td></tr>
<tr><td>Pyroxenite</td><td>10</td><td>11</td><td>8</td><td>9</td><td>35</td><td>36</td><td>35</td><td>36</td></tr>
</table>

However, in order to meet the chrome ore requirement for end use in a sustained manner keeping in view the proposed expansion, necessary infrastructure development for underground mining is required to be started at the earliest.

Life of the Deposit:

While open cast mining can continue for another 10 years at the present level of production, certain pits like the OB-II lumpy ore quarry and the mid band section of OB-X quarry which is producing beneficiable grade chrome ore shall be depleted much earlier. This is therefore going to be a brown field project wherein the mine is proposed to be operated both by opencast & underground means in the near future and then convert to a totally underground operation in the long term.

Extending the life of mine & Underground Mining:

From inception of mining operation at Sukinda Chromite Mine, both the OB-X and OB-II quarry were planned to operate at overall pit slope of 25^0. Slope stability study was undertaken through SRK, South Africa and CMRI, Dhanbad. Based on their recommendations, various measures were implemented in mines and permission for higher slope angle of 30^0 in OB-X and 40^0 in OB-II was obtained from DGMS. After successful implementation of the recommendation, the stability of pit slope was again reassessed by CMRI, Dhanbad and subsequently permission for higher pit slope angle of 35^0 for OB-X and 45^0 for OB-II was obtained from DGMS. Due to the above increase in overall pit slope angle, mining can now be carried out at greater depth with similar lease boundary restrictions as earlier which in turn has increased the life of mine.

Feasibility study for underground mining was conducted by SRK, South Africa. This study includes structural mapping and modeling, geotechnical assessment, hydrogeological study, insitu stress measurement, numerical modeling for the design of mine entries and its layout and design, stoping method with support system, hoisting/ haulage, ventilation assessment, drainage etc. and techno-economic assessment. The relevant portions of the above reports are enclosed as Annexure-23.

An independent assessment was also conducted by CIMFR, Nagpur for the geotechnical design of underground stoping methods as well as for subsidence prediction. Numerical modeling has been done for all such assessments. The relevant portions of the above reports are enclosed as Annexure-36.

Both these studies advocate the feasibility of underground mining in all three ore bodies. The recommendations of both these studies are more or less on the same lines and were found to complement each other. Accordingly, the design parameters for the underground mine, crown & sill pillars, stope design have been conceptualized considering both these independent studies.

The long-term ROM production requirement from the mine has been conceptualized to be around 2.4 MTPA of chrome ore. While this ROM production level is proposed to be achieved by the end of the Mining Plan period (i.e. 2017-18) and met from both opencast and underground workings put together, the underground mine is proposed to have an ultimate rated capacity of ~ 2.4 MTPA. In view of the same, the underground mine is proposed to be a fully mechanized mine with trackless method of mining. The underground loading and hauling equipment shall be diesel operated.

The location of the mine entries and other underground infrastructures of the underground mine has been shown in the conceptual underground plan & sections in drawing no.'s SCM/MP/15/12 & SCM/MP/22/12.

The access to the underground mine has been proposed to be made by 2 declines and a common vertical shaft to service the 3 ore bodies. The location of the shaft has been decided on the basis of the centroid of the resource for all 3 ore bands as well as the available vacant space to accommodate the other ancillary infrastructure. Similarly, the location of the north decline has been conceptualized based on the available vacant space near the shaft so as to also facilitate the sharing of the common surface infrastructures to be built up around the shaft. In the absence of any suitable vacant space for locating the southern decline it is proposed to re-handle the low grade dump situated at the north western corner of the OB-II quarry to accommodate the same.

The incidental ore production during the development period shall mostly be through the declines. After commissioning of the vertical shaft it shall serve for both production and man and material winding. The cross cuts connecting the shaft to all three ore bodies are proposed to be developed at 100 m intervals at (–) 95 and (–) 195 mRL. The shaft is proposed to be sunk to a depth of 365m (i.e upto – 240 mRL) to cater to the two main levels as stated above. The shaft may be further deepened at a later point of time after further proving of the ore bodies at deeper levels. The surface declines are proposed to be driven upto a depth of (–) 35 mRL. Thereafter, access to lower levels and sub-levels shall be through a network of ramps as shown in the conceptual plan and sections.

The development schedule of the underground mine has been conceptualized taking into account the life of the opencast workings in all 3 ore bodies & its ultimate pit limits. The time required for developing the underground mine for production build up from the stopes have been synchronized to have a smooth transition from opencast to underground mining.

The tentative road map for the programme being initiated for shifting over to underground mining is given below as Table.4.1.5.

Table: 4.1.5

Tentative Road map for Underground Mining

Activity	**H1'11**	**H2'11**	**Jan' 12**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**July**	**Aug**	**Sept**	**Oct**	**Nov**	**Dec**
Feasibility Report														
Approval of Mining Plan														
Prior Environmental Clearance& CTO														
DGMS Approval														
Capital Scheme Approval														
Detail Design														
Pretender														
Tender														
Pre project Activity														
Start Decline(S)														

*Statutory jobs include approval of Modified Scheme of Mining, Environment Clearance, and Consent to Operate from State Pollution Control Board etc.

Considering the extents of the orebodies along the strike direction and the use of trackless diesel equipment for its development, adequate ventilation is perceived to be an important criterion in the development stages of the underground mine. In order to address this issue quickly during the development stage of the mine, two drifts in the waste rocks are proposed to be made at (–) 35 mRL to serve the northern and southern sections of the underground mine respectively. These ventilation drifts shall be connected to the surface at intervals along the strike direction by ~ 3.5m dia. ventilation shafts and shall thus serve as the main ventilation outlet for the mine. As these drifts are meant to serve throughout the life of the underground mine, it is proposed to be adequately supported to ensure its long-term stability.

The development within the orebody is proposed to commence below the crown pillars which has been shown in the longitudinal geological section for all 3 orebodies (Drawing no.'s SCM/MP/21/12).

A geotechnical study and subsequent numerical modeling work has been carried out to determine the safe crown pillar dimensions for all three ore bands. Insitu stress measurement and physical and engineering properties of the ore has been studied. As per geotechnical investigation report (enclosed as Annexure-22, Ref. pp-53) it is stated that a crown pillar will be unstable in the soft friable chrome ore material and pillars can therefore not be developed in this material. Accordingly, the crown pillar has been designed to be below the weathered horizon of the northern and middle ore bands and within the hard and competent ore. As per the numerical modeling report (enclosed as Annexure-23, Ref.: pp-59) the ratio of the required crown pillar size to orebody width was determined to be 1:1 based on the insitu stress measurement done by MeSy. The same ratio was determined to be 1:2.5 based on the stress regime postulated by SRK's desktop study. However, presently the thickness of the crown pillar for all three orebodies has been proposed to be 60m in line with the requirements of the statute. The possibility of reduction of the crown pillar thickness shall be taken up later after further insitu stress measurements and detailed investigations from the underground workings and seeking permission for the same from the appropriate authority.

All the three orebodies are steeply dipping (80^0 to 90^0). The ore bodies are sufficiently far apart with the distance between the northern and middle ore bands being 250-270m. Essentially therefore there is no footwall and hanging wall side for these near vertical ore bands. With a view to optimize the development requirement, the ramps and the haulage drives/ drifts have been proposed to be located between the northern and middle orebodies. With this the number of intersections through the orebody shall be reduced to a great extent and shall thereby reduce the blockage of ore reserves in pillars and shall also avoid intersections through weak rock conditions. Since the distance between the northern and middle orebodies is far apart, the influence of mining (considering back filling of the stopes) on such infrastructure is therefore not likely. The advantages of the proposal of placing these infrastructures between the two ore bands far outweigh the disadvantages for the given geo-mining conditions.

The southern side of the southern body is having inferior strata conditions due to the presence of the Mahagiri fault between the quartzites and ultramafics together with shears and dykes along the strike direction of the orebody. In view of the presence of this zone of faulting and shears on the southern flank along the orebody, the haulage drives and ramp have been proposed to be located in the relatively more competent strata towards the north of the near vertical southern ore body. Moreover, the northern side is also closest to the vertical shaft position and avoids the cross cuts from the shaft to intersect the orebody. The above proposal of locating these infrastructures has been ratified from the geo-technical aspect by CIMFR, Nagpur in their report.

Methods of Underground Mining:

The selection of the mining method was based on extensive geotechnical studies and numerical modeling done by SRK, South Africa and CIMFR, Nagpur. The recommendations of both the studies were:

- The friable and soft chrome ore with weak soil like hanging and footwalls in the Northern & Middle friable ore bands up to the weathering horizons should not be mined from underground as it will be extremely unsafe;
- Below the weathering zone in the Northern & Middle friable ore bands both Cut and Fill method and Sub-level Open Stoping with post filling is possible depending upon the hardness of the orebody, and; similarly
- Both Sub-Level Open Stoping with post filling and Cut and Fill methods shall be applicable in the Southern lumpy ore band depending upon the extent of disturbance/ fracturing and soundness of the orebody and its wall rocks.

Sub-level Open Stoping:

The schematic diagram showing the method of Sub-level Open Stoping is given as Fig-4.1. The stope block is mined out from the "bottom-up", as a panel between sub-levels is mined and filled, the fill is then used as a platform for the above panel is be mined. Before the panel is mined the in-stope development, on the top and bottom, to the limits of the block must be completed. A raise or slot is developed up from the bottom ore drive to the top ore drive. This slot provides a free-breaking face for the ring blasting and ventilation.

The sub-level open stoping and its many derivatives, is relatively flexible in that it can accommodate varying widths down to a minimum of 2 m. The maximum mineable width is dependent on the rock mass characteristics of the ore body and the surrounding host rock.

Cut & Fill Stoping:

The cut and fill mining method is based on longitudinal drifts mined on strike. The drifts shall be 5 m wide x 5 m high and the length dependent of distance apart of the access crosscuts. Fig 4.2 shows a typical drift and fill mining with long anchor support for the hanging wall.

Access to the ore body is from ramps and the ore is mined in horizontal cuts and carried out in a "bottom-up" sequence. The drift is filled and mining always takes place on top of the fill material. Drift and fill mining is generally used in rock conditions that do not allow the opening up of the stope to the full width of the ore body.

The filling of the stopes shall be done with the rejects generated from the underground development drives as well as with the indigenously available overburden and mill tailings as a paste fill. A separate plant shall be set up in the surface to pump the fill material for the stowing of the voids in the underground stopes.

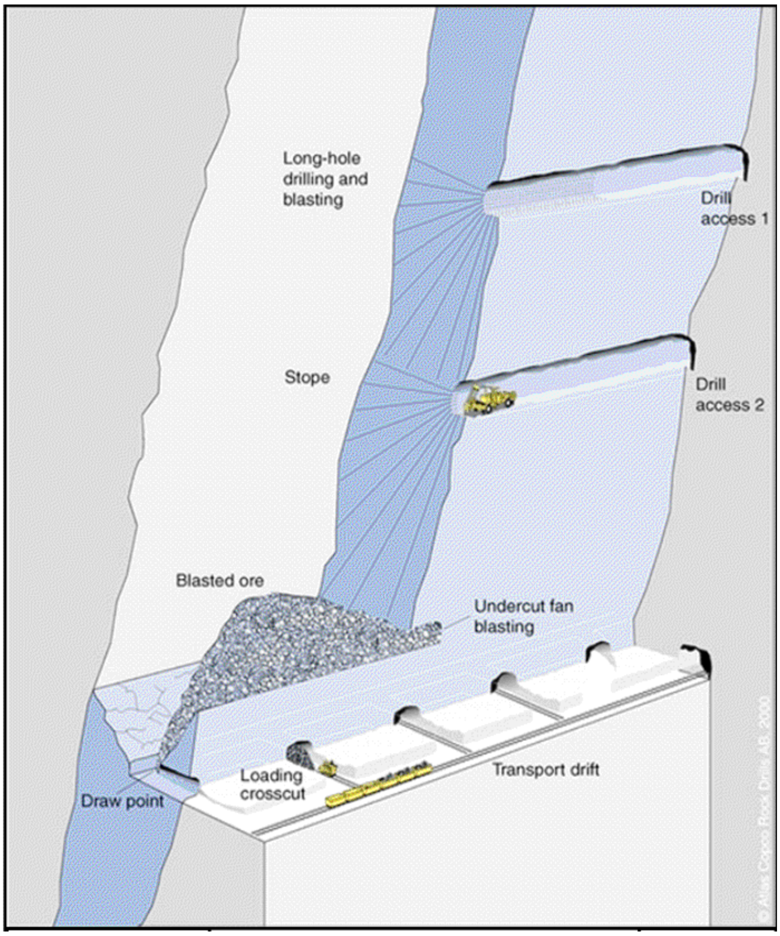


Fig. 4.1: Typical layout for Sub-level Open Stoping Method

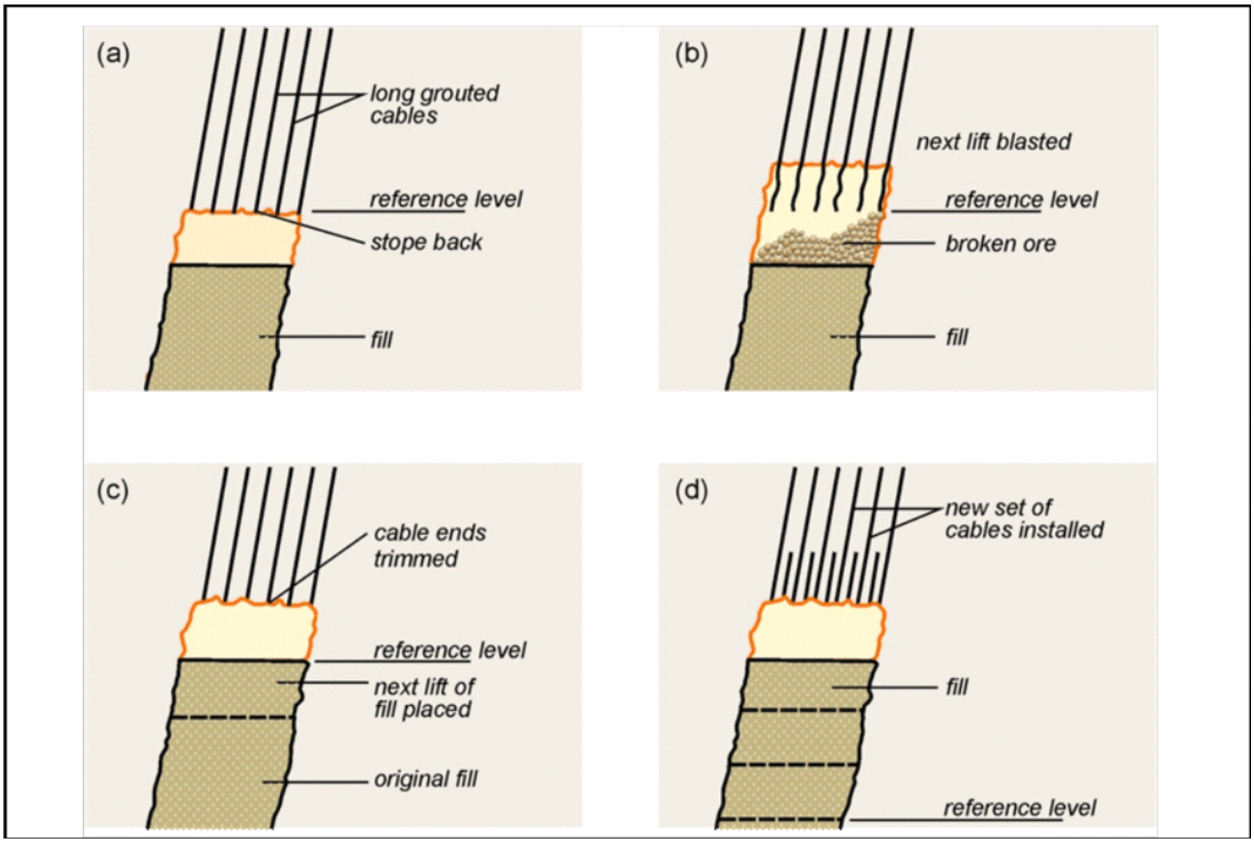


Fig. 4.2: Mining sequence in typical Cut & Fill Method in poor ground

The cross cuts from the haulage drives/ drifts are proposed to be driven to the orebodies during the Mining Plan period at intervals of 110m to sub-divide the ore body between the levels into panels. The dimension of the ore drives and sub-level intervals within the panels to be subsequently stoped out are proposed to be the same for both the above stoping methods, as further detailed in the Mining chapter. Presently, all stoping is being proposed within the ore horizons which are in the probable reserve category. Based on the feasibility study, the proportions of cut and fill mining in all three ore bodies have been estimated to be as follows:

- Northern Band: 50%
- Middle Band: 22%
- Southern Band: 40%

In order to save on time and cost of drilling the > 300-500m long boreholes at close interval from the surface, further exploratory drilling have been proposed to be undertaken from the proposed underground haulage drives/ drifts at a strike interval of 50-100m and ~ 30-50 m in the dip direction to bring the ore reserves to the proved category. The company proposes to engage its own team of geologists and geo-technical engineers to make a further detailed study of the geo-mining conditions of each of these panels after such levels are developed, drilling and geo-technical logging work is completed. Accordingly, the panels within the two levels shall be divided into different geo-technical domains for selecting the appropriate support system and any one of the above two applicable stoping methods. The cut and filling stoping method shall be applied mostly in the comparatively weaker zones. The company as per Rule 26 of the MCDR, 1988 and Rule 107 of MMR, 1961 shall accordingly apply for stoping permission well in advance (not less than 60 days). Stoping shall commence only after receiving such permission. The areas in which stoping shall be undertaken during the plan period to meet the year wise production requirements have been indicated in different colours in the longitudinal section for each of the three ore bodies.

Post Mining Land Use:

The Conceptual Mining Plan has been furnished as shown in Drawing No. SCM/MP/05/12. As mentioned in approved Scheme of Mining, back filling of quarries and its subsequent reclamation and rehabilitation shall be taken up after the economic pit bottom depths are reached along the entire strike of the deposit. Overburden/ associated mineral generated in 2011-12 to 2013-14 shall be dumped in the additional area of 100ha. Similarly the overburden/ associated mineral generated during 2014-15 to 2017-18 shall be backfilled in OB II quarry in a manner as shown in the drawing no SCM/MP/08/12. The land use pattern at the end of planned period (2012-13) and at end of life (opencast operation) is given below in Table 4.1.6; as per Drawing No. SCM/MP/13/12 & SCM/MP/13A/12.

Table 4.1.6
Land use Pattern

Activity	As at the end of plan period (2012-13) in ha.	As at the end of conceptual period (life of the mine) in ha.
Area to be excavated	201.50	175.49**
Storage of Top Soil	0.00	0.00
Over Burden Dump	37.47	76.11
Mineral Storage	30.70	32.86
Infrastructure (Workshop, magazine etc)	7.92	7.92
Roads (Present L= 7812m, W= 25m)	19.05	19.05
Railways	0.00	0.00
Greenbelt	38.53	29.69
Tailing Pond	15.95***	10.00#
Effluent Treatment Plant	0.32	0.32
Mineral Separation Plant	18.38	18.38
Township Area*	36.18	36.18
Others (to be specified)	0.00	0.00
Area which will remain untouched	0.00	0.00
Total	406.00	406.00

Remarks:
* - includes roads within the colony area
** - includes pyroxenite section & excludes OB-II area as backfilled
*** - includes area of pyroxenite section for temporary storage of tailings
# - excludes area of pyroxenite section

Mineral Beneficiation:

No such change is being proposed in the mineral beneficiation process during the plan period till 11.01.2013 apart from what has been proposed in the tailing management process is as enumerated below.

The water from the tailings is further proposed to be filtered by the introduction of parallel plate press filters as shown in Fig 8.2 of chapter-8. 3 such units having a capacity of 27 t/ hr each is proposed to be installed. The tails after dewatering shall then be in cake form and shall then be transported in dumpers to the proposed disposal site. The schematic layout of the manner in which these parallel plate press filters shall be installed within the Chrome Ore Beneficiation Plant as described in chapter-8.

The tailing dam constructed within the lease to store the tailings in slurry form within the pond so created is almost full. The stability study conducted by IIT, Kharagpur suggests that further heightening of the dam would be unsafe. In the absence of any further vacant space within the leasehold to store the tailings from the COB Plant, the company has identified and applied for an additional area of 73.685 ha having 65.315 ha of non-forest Government land for future solid disposal of the tailing after filtration of the water within the plant, which is under process. Till such time pyroxenite section of OB-II quarry had been temporarily discontinued to store tailings from the COB Plant. The Modified Scheme of Mining in this respect was approved vide letter no. 314(3)/2011-MCCM(CZ)/MS-08, dated: 22/07/11 (Annexure-7). While the installation of the filter presses is going as per schedule, it may take a further period of at least two years to acquire the above mentioned applied land after seeking forest clearance for the proposed road leading to the same. In order to take care of such eventuality, it is therefore necessary to make alternate arrangements to store the tailings for another two years beyond the end of this scheme period so that the production from the mine is not hindered. Accordingly, it is further proposed to store the solid tailings (after filtration in the filter presses proposed to be installed) in the present pyroxenite section area till 2014-15. It is expected that it shall take a year to evacuate the tailings so stored and rehandle the same to the applied area. Thus, the reopening of the pyroxenite section for the production of pyroxenite is proposed to be deferred to 2016-17.

Conceptual Environment Management:

The various environment parameters shall continue to be monitored as per conditions laid in the EC, Consent to Operate, and as per notifications issued from time to time. The following environmental protective measures that are being taken and further proposed to be maintained and continued in future are:

(i) Water sprinkling and wet drilling for dust suppression,
(ii) Wet beneficiation process and water jet system at feed hopper of processing plants to allay the dust,
(iii) Garland drains and toe wall at the foot of the overburden dumps to prevent soil erosion,
(iv) Treatment of mine discharge water and surface run-offs in the effluent treatment plants to ensure the discharge to be hexavalent chrome free,
(v) Recycling and reuse of the water from the tailings pond,
(vi) Oil and grease trap at the workshop areas to prevent soil contamination,
(vii) Treatment of domestic effluents in the Sewage treatment plant.
(viii) Acoustic enclosures for DG sets, rubber lining in the transfer points & discharge chutes to reduce noise levels
(ix) Development of the green barriers between the mining and colony area as well as along the lease boundary
(x) Safe storage and disposal of hazardous wastes such as used/ waste oil, old batteries, waste containing oil and ETP sludge to authorized recyclers and facility operators.

Conceptual reclamation

The Steel Company proposes to reclaim the open pits by back filling after the same reaches its ultimate pit bottom depths and the mineable reserves by opencast methods are exhausted. Wherever possible such backfilling shall be done from the bottom upwards in a retreating fashion. Thereafter, plantation is proposed to be taken up over such completely backfilled areas shall be taken up to rehabilitate the area. The mined out areas and benches that have not been proposed to be backfilled shall be reclaimed by plantation. Similarly, the overburden dumps which are proposed to be made in a retreating manner shall be reclaimed and rehabilitated by afforestation in a phased manner after it attains the designed extent. The areas proposed to be so reclaimed and rehabilitated have been shown in the conceptual plan.

The afforestation of the dumps and mining benches will be done by the following methods.

a) Method of pitting and planting.

(i) The worked out benches of the quarry and dumps will be reclaimed by making pits 0.5 m x 0.5 m x 0.5 m size 2m apart. The pits will then be filled with sweet earth, sand and cow-dung.

(ii) Neem cake powder is proposed to be applied in the pit to protect the plants from white ants.

(iii) Such ground preparation is proposed to be done before the monsoons after which appropriate varieties of saplings will be planted during the monsoon.

b) Method of planting by contour trenching

This method is proposed for slopes wherein contour trenches are dug at 3m interval along the contour. The excavated earth is stacked on the edge of the trench on the lower slope side to arrest the water flow that comes due to rains and accumulates on the trenches and gradually seeps through the strata enabling the planted saplings to get water and nutrients regularly for healthy growth.

Time bound action plan shall be drawn to reclaim the dead dumps, vacant area within the colony and to develop greenbelt. Year wise reclamation progarmme within the mining lease, additional area of 100ha and applied area of 73.685 ha is as given below in Table 4.1.7

Table 4.1.7

Year wise Reclamation Plan

Year	Within the Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total Plantation	
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)
2011-13	6.4	15875	12.1	30125	0.0	0.0	18.4	46000
2013-18	13.5	33750	39.9	99750	8.5	21125	61.9	154625
2018-23	42.0	105000	13.0	32500			55.00	137500
2023-28	68.4	170875					68.35	170875
2028-33	85.2	213025					85.21	213025
Total	215.4	538525	65.0	162375	8.5	21125	288.8	722025

The phase wise reclamation programme as per the approved FDP to cover the entire additional allotted area of 100 ha. is as follows:

Phase	Area (ha)	No. of saplings to be planted
Phase – 1	25	62500
Phase – 2	25	62500
Phase – 3	25	62500
Phase – 4	25	62500

The phase wise stabilization and rehabilitation programme for 25 ha. each shall be strictly adhered to, the annual programme may undergo slight changes depending upon the field conditions and market situations encountered. However company shall try to plant more trees to match the existing greenery of Mahagiri hills.

5.0 MINING

A. Opencast Mining:

The basic opencast mining method remains same as that given in the approved Mining Plan under para 4.0 to 4.2 and under para 5.0 of approved Scheme of Mining. The updated average pit dimension of OB-X and OB-II quarry as on 01.04.2012 is mentioned in previous chapter under Table 4.1.1.

There is no change proposed in the present opencast mining process, mineral processing. However some changes are being proposed w.r.t the ultimate depth of mining as envisaged in approved Scheme of Mining. The changes are enumerated in details in subsequent paras.

OB X quarry:

It is proposed to design the benches in a manner to achieve an overall ultimate pit slope of 35 degree as stipulated by the Director of Mines Safety, Bhubaneswar as per the approved machine mining bye-laws. The quarry had been planned to go up to a depth of -25 mRL with the bench height being 8 to 10 m and 10 to 12 m width with an individual bench slope of 85 degree. But due factors related to DGMS permission and overburden disposal space as discussed in the conceptual mining plan the change in the ultimate pit bottom is proposed to be 10mRL in Northern ore band (at sump) and 15mRL in Middle ore band (at sump). Other parameters related to ramp etc shall continue to remain same as per the approved Scheme of Mining.

OB II quarry (for Lumpy Chrome ore & pyroxenite):

The strata being encountered presently in quartzites and the ultrabasic rocks in the lower horizons are hard. It is proposed to design the benches in a manner to achieve an overall ultimate pit slope of 45 degree as stipulated by the Director of Mines Safety, Bhubaneswar as per the approved machine mining bye-laws (Annexure-18) and approved Scheme of Mining. The quarry had been planned to go up to a depth of 25 mRL. The benches had been designed to have a height of 8 to 10 m and 8 to 10 m width with an individual bench slope of 85 degree. The ramp shall be of a switchback type located along the northern slope and shall have a gradient of 1:16 and width of 18 to 20 m. But due factors related to DGMS permission and overburden disposal space as discussed in the conceptual mining plan the change in the ultimate pit bottom is proposed to be 30mRL (at sump area) in the Southern Lumpy Ore band.

The strata being encountered in pyroxenite band is hard. It is proposed to design the benches in a manner to achieve an overall ultimate pit slope of 45 degree as stipulated by the Director of Mines Safety, Bhubaneswar as per the approved machine mining bye-laws during the life of mine. The quarry has been planned to go ultimately up to a depth of 57 mRL. The benches shall be designed to have a height of 8 to 10 m and 8 to 10 m width with an individual bench slope of 70-75 degree. The ramp shall be of a switchback type located along the southern slope and shall have a gradient of 1:16 and width of 15 to 18 m.

The pyroxinite section of OB-II quarry shall be temporarily discontinued to store the tailings generated from the chrome ore beneficiation plant. The holding capacity of the pit is around 16.5 lakh tonnes of tailings. The water shall be re-circulated back to the plant by continuous pumping.

B. Underground Mining:

i. Mode of entry:

The schematic layout of the current opencast mining operation in respect to the three ore bodies is shown in Fig.5.1. Being a brown field project, the entries to the proposed underground mine have been located considering the available space, ultimate open pit limits, and existing infrastructures. The number, type and size of the entries to the underground mine have been determined in order to service the ultimate underground production capacity of 2.4 Mtpa and also cater to the needs of ventilation and transport of men and material to the underground mine. Trackless mining method, as further elaborated below has been considered necessary to achieve the required/ designed levels of production from the underground mine.

Based on the above considerations, the access to the underground operations shall be serviced by the following infrastructure:

Surface Declines: Two declines shall provide the initial access to develop the underground infrastructure. While one decline shall serve the Northern and Middle Bands, another decline shall serve as an access to the Southern Band. The declines shall be of 5 m x 5 m X-section at an angle of 7° to 8°;

Vertical Shaft: A single vertical shaft of 6.3 m diameter shall be sunk to provide central rock hoisting facilities with access for men and material as required. The shaft shall service the underground operations via two main platform levels at Level -95 and Level -195 mRL's. With the increase in depth of the underground mine in the distant future, this shaft shall also be equipped and be capable of hoisting the entire production from the underground mine;

Ramps: The underground workings shall be serviced by four ramps. These ramps or underground declines shall be connected to the main surface declines at Level -35 mRL. 3 of such underground declines shall service the Northern and Middle Bands and 1 for the Southern Band.
The schematic diagram showing the location of the main entries to the underground mine is given in Fig.5.2. The drawing showing the location of these main entries is enclosed as Drawing No. SCM/ MP/ 15/ 11.

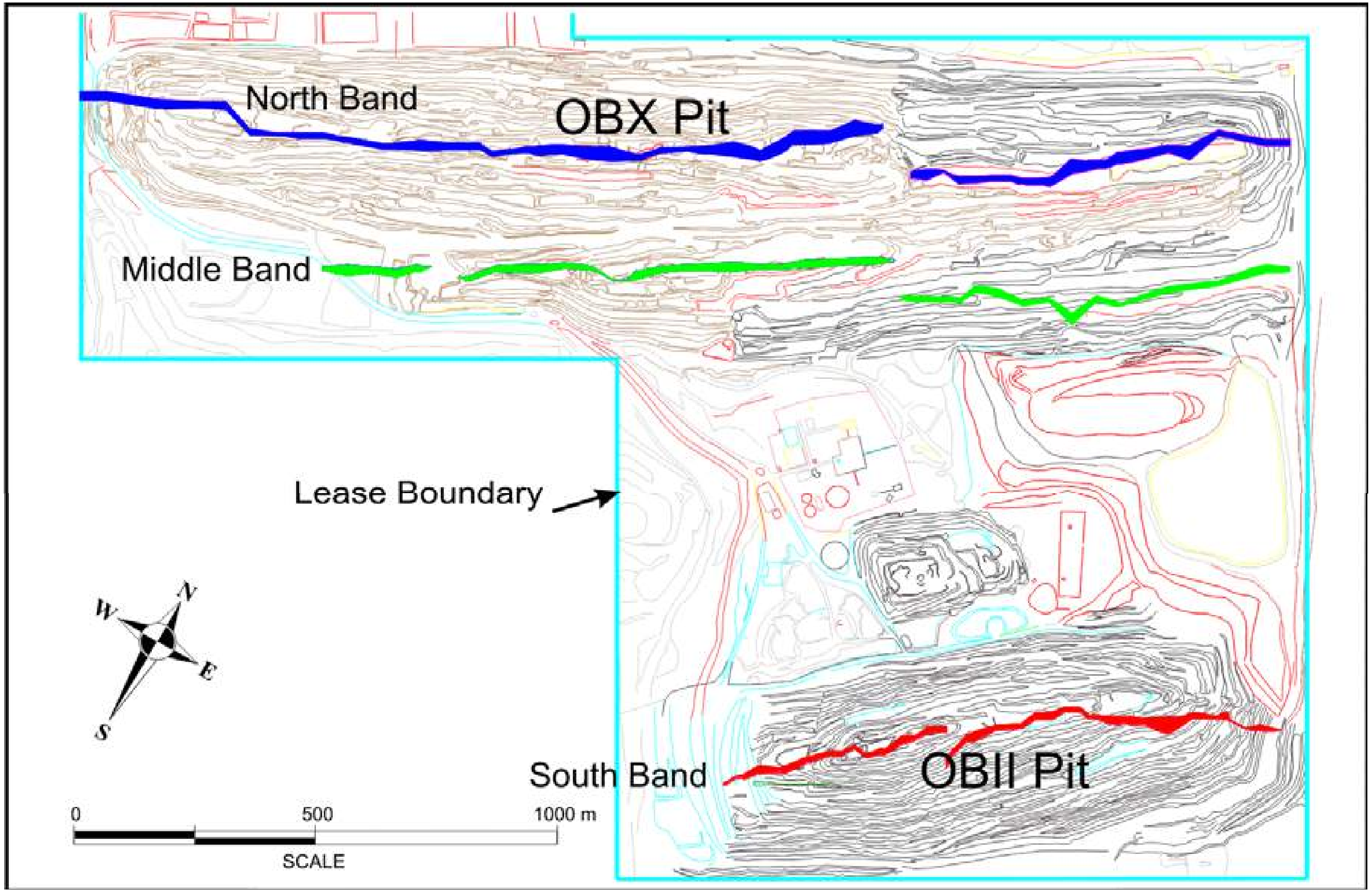


Fig.5.1: Schematic Plan View of the Mining Operations

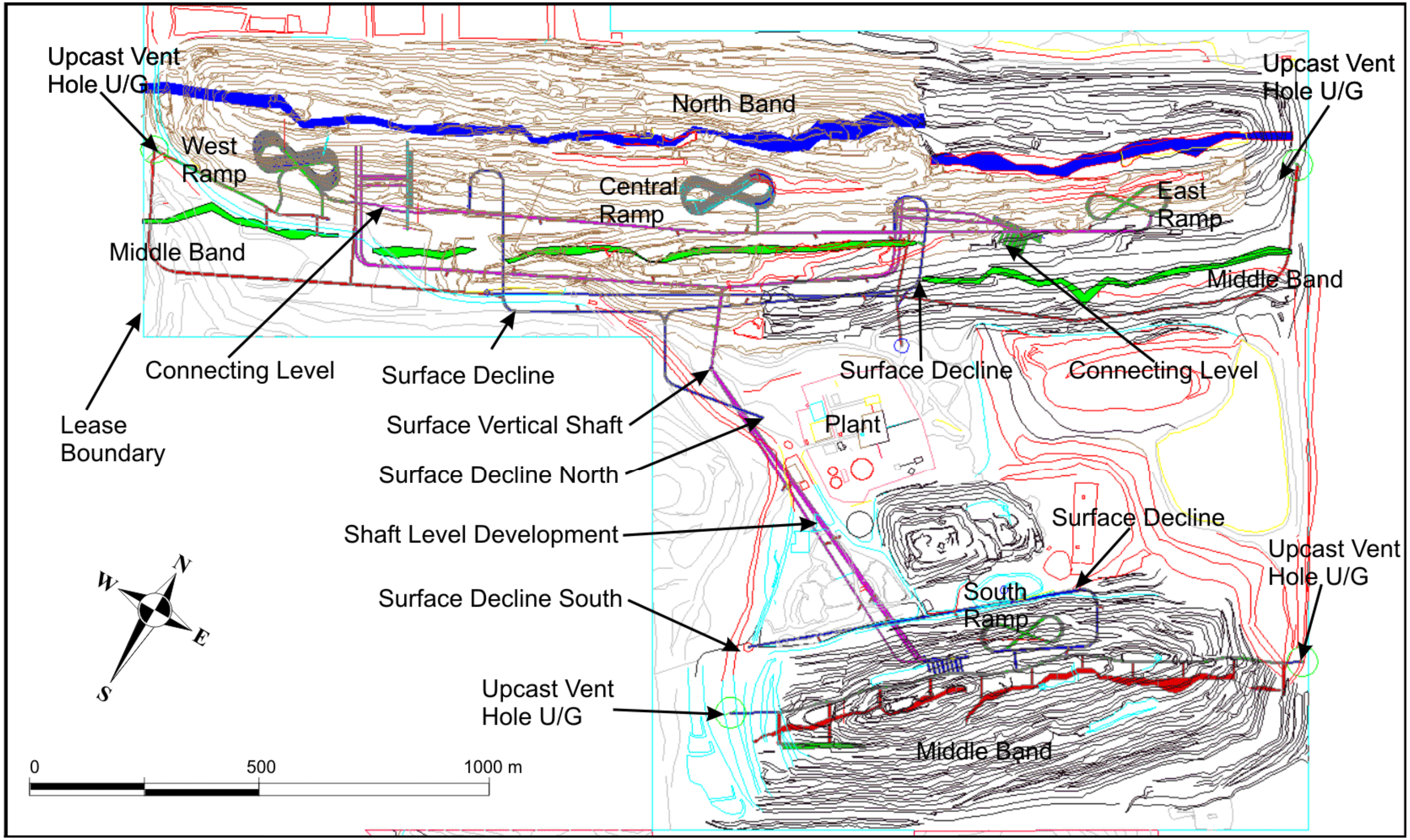


Fig. 5.2: Schematic Plan View of Mine Entries for Underground Mine

The coordinates of the main entries to the underground mine is given in Table No. 5.1.2 below.

Table No.5.1.2
Coordinates of the Main Entries to the Underground Mine

Description	Northing/ Y-coordinate	Easting/ X-coordinate
Declines & Shaft		
North Decline	1900.0	3000.0
South Decline	1450.0	3100.0
Vertical Shaft	2007.4	2887.7

There may be minor changes in the above location of the mine entries during the actual construction of these openings depending of the site conditions.

Surface Declines:

The schematic cross section of the declines is given below in Fig. 5.3.

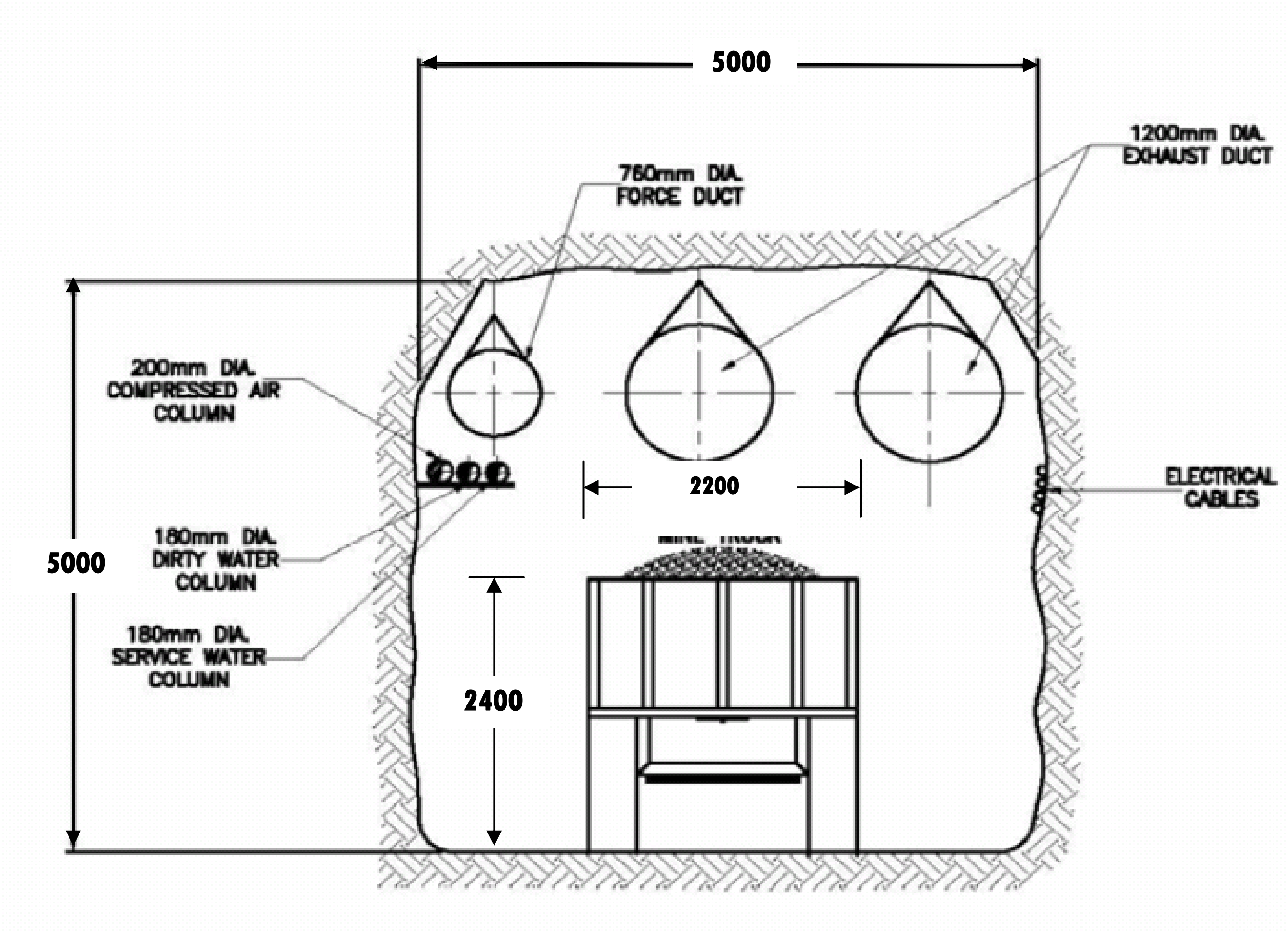
Trackless mining method has been considered for meeting the desired production levels. Surface declines has been considered as the most effective and quick means of entry to the underground mine In order provide easy access of the tyre mounted mining equipments which has been further elaborated below. The use of trackless equipments allows for greater flexibility of operations. The transport of waste and ore during development phase of the underground mine shall be through these declines. The declines shall be of 5 m x 5 m X-section and driven at an angle of 7^0 to 8^0. These surface declines shall also serve as intakes for the fresh air to the underground workings.

Vertical Shaft:

The schematic cross section of the vertical shaft is given below in Fig. 5.4.

This common vertical shaft of 6.3 m diameter shall be sunk to provide access for men and material as well as central rock hoisting facilities. The shaft shall service the underground operations via two main platform levels at Level -95 and Level -195 mRL’s for all three ore bands. All the main levels shall be connected by shaft cross cuts which shall serve as access to the shaft and ore transfer stations in each main level. This main vertical shaft opening shall be a downcast shaft and serve as one of the main intakes of fresh air to the underground workings.

The shaft shall be equipped to accommodate 2 cages of 10 t payload or 35 persons capacity each. It shall also be provided with two skips of 12 t capacity each.



(All dimensions in mm)

Fig. 5.3: Schematic X-section of the Declines

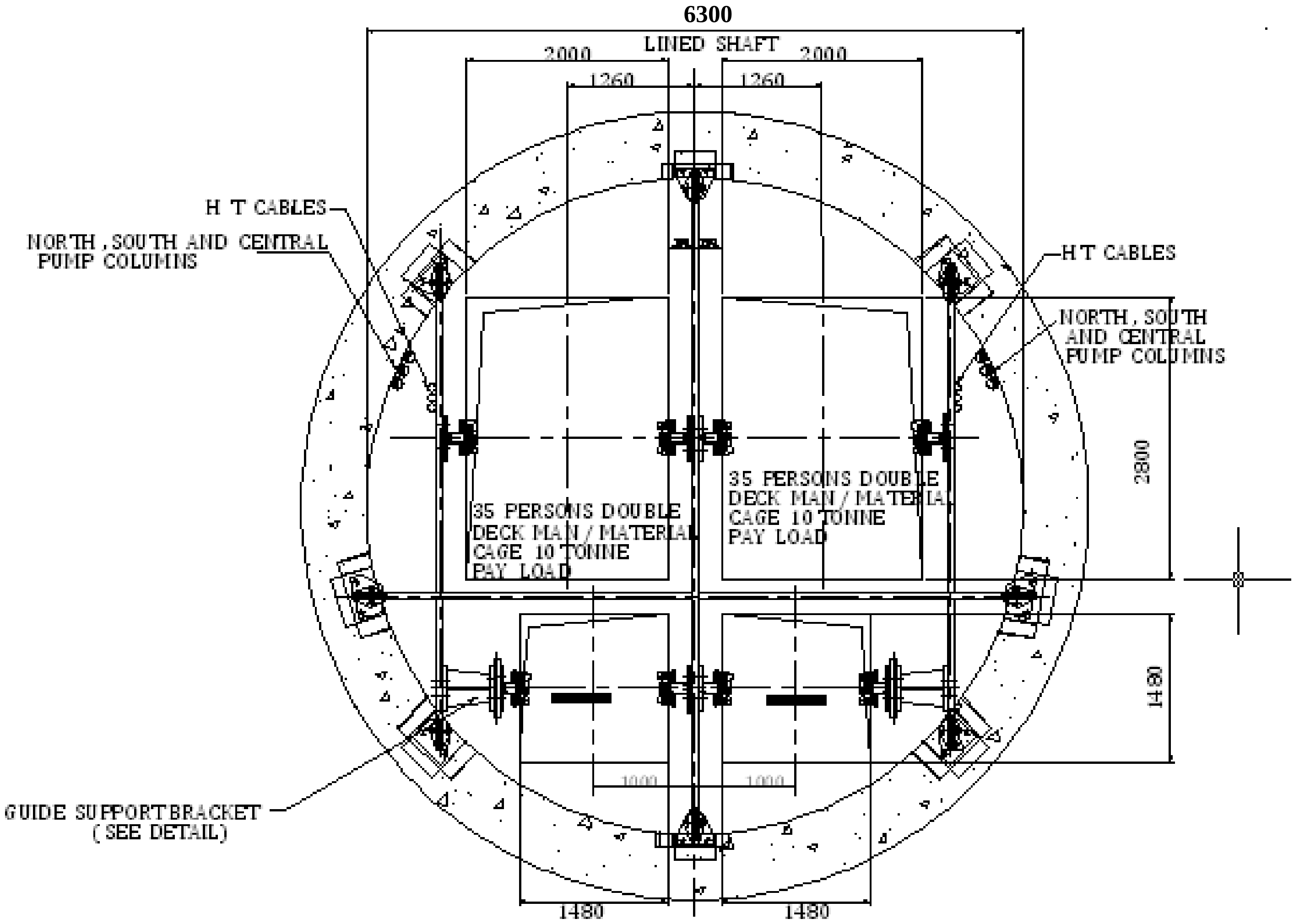


Fig. 5.4: Schematic X-section of the Main Vertical Shaft

Ramps:

The underground workings shall be serviced by four ramps, 3 of which shall service the Northern and Middle Bands and 1 for the Southern Band. The access surface declines shall connect the main ventilation drive at -35mRL which is around 160m below the general surface ground level. This is the starting level for the ramps for all the three bands. Three ramps shall be developed to access different level and sublevel horizons of the Northern & Middle ore bands. The size of the ramps shall be 5m x 5m both for the development as well as production stages of the mine. During the development stage two exhaust ducts and one forcing duct is to be established in the ramps to simultaneously allow the movement of the mine trucks within it. The waste rocks to be transported by mine trucks shall be through these ramps. Compressed air, drilling water and drinking water lines shall be laid along these ramps as permanent fixtures, to serve the service requirements for the underground faces and stopes.

The ramps shall be developed in a figure of eight pattern as shown in the schematic diagram in Fig.5.5 given below.

ii. System Of Winding And Hoisting:

The vertical shaft shall be initially sunk to a depth of 365 m and shall be fitted with two counterbalancing cages and skips. Considering the constraints of space for surface infrastructure and future increased depths of the underground workings, tower mounted Koepe winding system is proposed to be adopted. The winding and hoisting system of the mine shall follow the following sequence from development stage to the final production stage.

a. During the development phase of the mine the entire ore and waste shall be transported through the declines with trackless equipments like LHD (Load Haul Dump) of 4-7 m3 and LPDT (Low Profile Dump Truck) of 15-20 m3 capacity. Loaded LPDTs from each sublevel shall move to the nearest level drive to reach the decline.

b. After commissioning of the shaft all ore hoisting shall be done by the shaft skips of 12 t capacity. There shall be two such skips of counterbalancing type. The skips shall be loaded at the skip loading station located at -215 mRL which is connected by ore passes from the grizzly of each main production level at -95 and -195 mRL's. Two 10 tonne pay load capacity cages (35 person capacity) shall also facilitate the lowering and hoisting of man and material from/to the underground workings.

The schematic vertical section of the shaft and its main levels is given in Fig. 5.6.

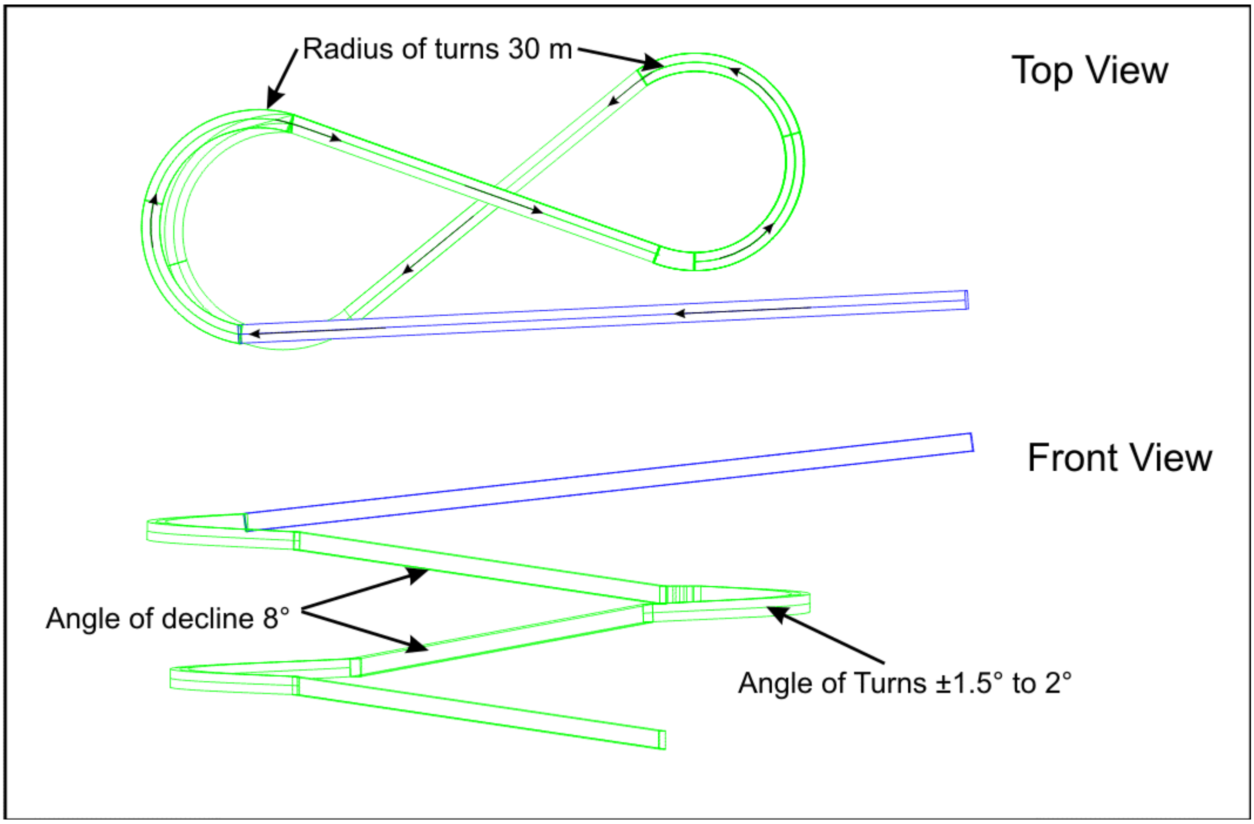


Fig. 5: Schematic Plan & Elevation View of the Ramps

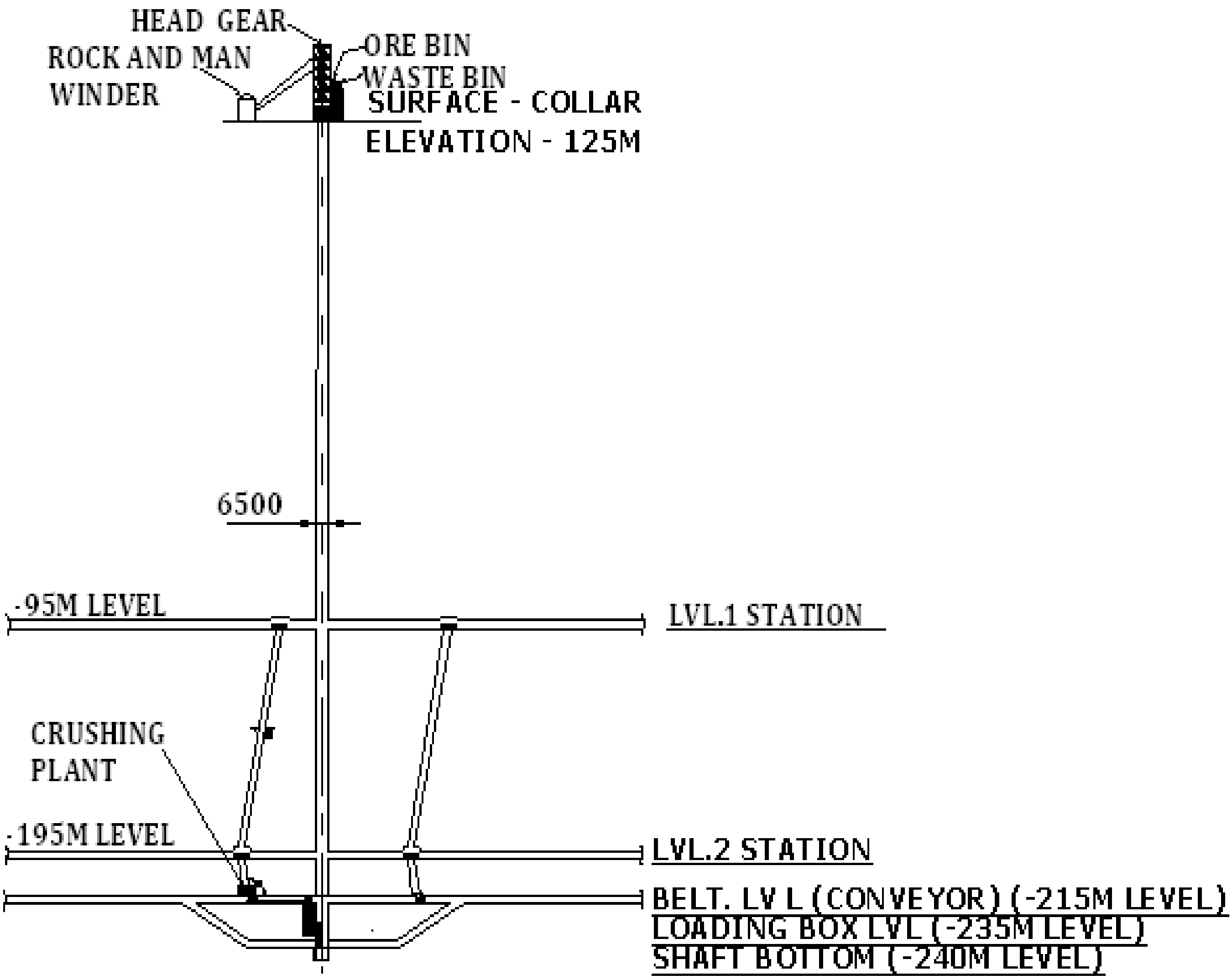


Fig.5.6: Schematic view of the Vertical Section of the Shaft

Skip and cage hoisting capacity:

2 No.'s of Double drum tower mounted Koepe winders shall be installed having a rating of 2 MW each. Both the cage and skips shall be counterbalancing type, so no counterweights shall be there. The main shaft & winder specifications are as follows:

Required skip hoist capacity = 8000 tons per day of ore.

Assuming,

2 number of 12 t skips each at an operating speed = 15 m/ sec
2 number of cages (Type- double deck) with 35 men carrying capacity (10 t payload) each at an operating speed = 5.5 m/ sec

Skip capacity calculation:

- Loading time- 20 sec
- Acceleration time -10 sec
- Travel time- 20 sec
- Deceleration time -10 sec
- Dumping in surface ore bin - 10 sec
- Total time for one cycle - 70 sec, i.e. 1 min 10 sec
- Trips in an hour - 51 trips
- Running hour in a day - 2 shifts - 14 hr, (assuming, I hr maintenance in each shift)
- Total trips in a day- 51 x 14 = 714 trips
- Tonnage in a day- 714 x 12 t(Skip capacity) = 8568 tonnes
- Steady state production capacity = 8000 t /day ~ 2.4 Mtpa

Cage capacity calculation:

.

- Acceleration time- 20 sec
- Deceleration time- 20 sec
- Traveling time(to -195 m level)- 60 sec
- Total traveling time for one down trip- 100sec
- Trips in an hour(½ mode cage travel time, i.e. 30 min)- 18 trips
- Man shifting capacity- 35 x 18 = 630 persons/hr
- Proposed highest direct underground manpower in a shift ~ 600

Deployment of Machinery (Underground):

Trackless method of mining is being proposed. The proposed equipments for underground operation along with their make and capacity to be used during the Scheme Period is as given below in Table-5.1.3.

Table-5.1.3
List of Underground Equipment

Equipment	OB-X	OB-II	Total	Make	Capacity
Twin Boom Drill Rig	1	1	2	Sandvik/ Atlas Copco	76-127mm
Single Boom Drill Rig	1	1	2		
Rock Bolter	1	1	2		33-45mm
Development LHD (5m³)	2	2	4		4-7 cum
Primary Dump Truck(30t)	2	1	3		15 - 20 cum
Shotcreter	1	1	2		
ANFO Loaders	1	1	2		
Service Vehicles	4	3	7		
Total	**13**	**11**	**24**		

Process flow of Underground Mining:

Broadly the mining operation can be divided into two categories, i.e. development and stoping. The process flow chart for underground mining is given below in Fig 5.7.

Development essentially consists of the following steps:

1. Construction of the entry headings to the deposit by declines for trackless equipments and shaft from the surface level.
2. Construction of the horizontal levels and sublevels as per the design.
3. Establishing the ventilation connection to each level and stope blocks through ventilation raises and stope raises.

The development of openings generally follows the following cyclic pattern which consists of the following operations like:

a) Electro hydraulic jumbo drilling
b) Charging and blasting
c) Roof dressing, supporting and stabilization
d) Removal of the blasted rock by trackless equipments like LHD and LPDTs and preparation for next round of drilling.

The process flow chart for the stoping operations is below given in Fig 5.8.

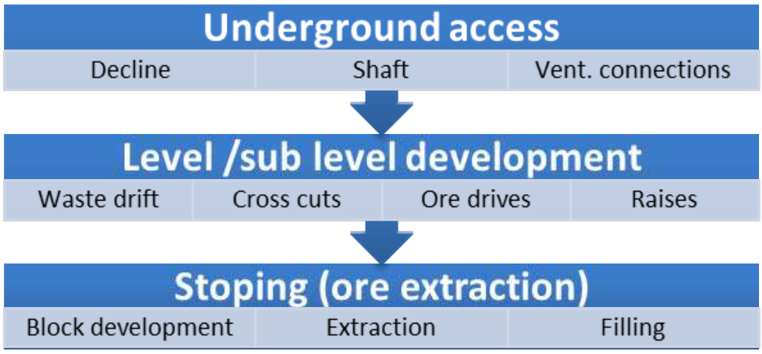


Fig. 5.7: Process Flow Chart for Underground Mining

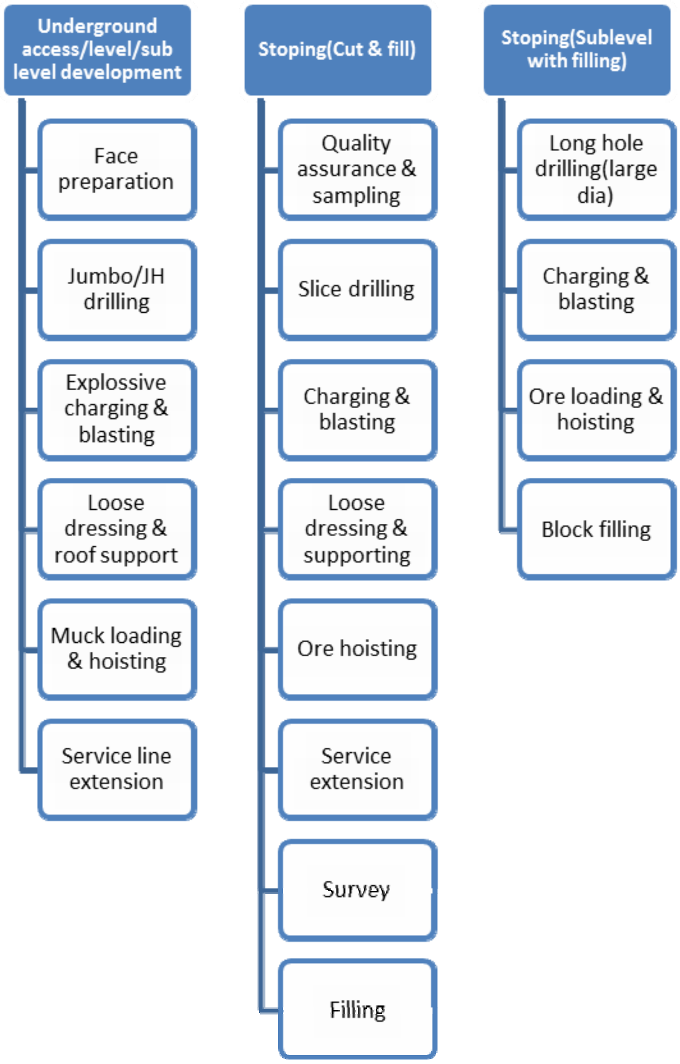


Fig. 5.8: Process Flow Chart of the Stoping Operations

5.1 SALIENT DESCRIPTION OF PRESENT MINING METHOD:

No modification is being proposed for opencast mining method. As per the extensive geotechnical studies and numerical modeling done by SRK, South Africa and CIMFR, Nagpur underground mining shall be carried out in the manner as briefly described below.

- The friable and soft chrome ore with weak soil like hanging and footwalls in the Northern & Middle friable ore bands up to the weathering horizons should not be mined from underground as it will be extremely unsafe;

- Below the weathering zone in the Northern & Middle friable ore bands both Cut and Fill method and Sub-level Open Stoping with post filling is possible depending upon the hardness of the orebody, and; similarly

- Both Sub-Level Open Stoping with post filling and Cut and Fill methods shall be applicable in the Southern lumpy ore band depending upon the extent of disturbance/ fracturing and soundness of the orebody and its wall rocks.

Development of underground mining shall commence after getting prior environmental and other relevant clearances. Development essentially consists of the following steps:

1. Construction of the entry headings to the deposit by declines for trackless equipments and shaft from the surface level.
2. Construction of the horizontal levels and sublevels as per the design.
3. Establishing the ventilation connection to each level and stope blocks through ventilation raises and stope raises.

5.2 YEARLY DEVELOPMENT PLAN:

The yearly proposed excavation plans for the next 2 years for OB X and OB II quarries is shown in drawing no. SCM/MP/06/12 and drawing no. SCM/MP/07/12.

The pit wise, year wise, quantities of overburden/ associated mineral planned to be handled as per approved Scheme of Mining is given in Table 5.2.1 & 5.2.2. Now the proposed change in pit wise, year wise quantities of overburden / associated mineral to be handled is given below in Table 5.2.3 & 5.2.4. The year wise development of underground infrastructure is given below in Table no. 5.2.5.

Table 5.2.1
Yearly Development Plan: OB-X Quarry
(As per approved Scheme of Mining)

YEAR	Opencast/ Underground	OB Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10$ % < 42 %		Total ROM (LCuM)		Total Excavation (in LCuM)
			L CuM	LMT	L CuM	LMT	L CuM	LMT	
2011-12	Opencast	80.00	2.70	7.70	2.59	6.61	5.30	14.32	85.29
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	80.00	2.70	7.70	2.59	6.61	5.30	14.32	85.29
2012-13*	Opencast	65.00	2.28	6.49	2.06	5.25	4.34	11.74	69.34
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	65.00	2.28	6.49	2.06	5.25	4.34	11.74	69.34
Total	Opencast	145.00	4.98	14.19	4.65	11.86	9.64	26.06	154.63
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	145.00	4.98	14.19	4.65	11.86	9.64	26.06	154.63

*figure for 2012-13 is till 11.01.2013

Table 5.2.2
Yearly Development Plan: OB-II Quarry
(As per approved Scheme of Mining)

YEAR	Opencast/ Underground	OB Volume (L CuM)	ROM Processing $Cr_2O_3 \geq 30$%		To Low Grade dump Cr_2O_3 < 30 %		Total ROM (LCuM)		Total Excavation (in LCuM)
			L CuM	LMT	L CuM	LMT	L CuM	LMT	
2011-12	Opencast	12.00	0.38	1.08	0.04	0.11	0.42	1.19	12.42
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	12.00	0.38	1.08	0.04	0.11	0.42	1.19	12.42
2012-13*	Opencast	11.00	0.34	0.96	0.03	0.10	0.37	1.06	11.37
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	11.00	0.34	0.96	0.03	0.10	0.37	1.06	11.37
Total	Opencast	23.00	0.72	2.04	0.07	0.21	0.79	2.25	23.79
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	23.00	0.72	2.04	0.07	0.21	0.79	2.25	23.79

*figure for 2012-13 is till 11.01.2013

Table 5.2.3
Yearly Development Plan: OB-X Quarry
(Proposed Modification)

YEAR	Opencast/ Underground	OB Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10$ % < 42 %		Total ROM (LCuM)		Total Excavation (in LCuM)
			L CuM	LMT	L CuM	LMT	L CuM	LMT	
2011-12	Opencast	50.00	1.04	3.77	2.81	9.00	3.85	12.77	53.85
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	50.00	1.04	3.77	2.81	9.00	3.85	12.77	53.85
2012-13*	Opencast	39.00	1.14	4.15	2.04	6.39	3.18	10.54	42.18
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	39.00	1.14	4.15	2.04	6.39	3.18	10.54	42.18
Total	Opencast	89.00	2.18	7.92	4.85	15.39	7.03	23.31	96.03
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	89.00	2.18	7.92	4.85	15.39	7.03	23.31	96.03

*figure for 2012-13 is till 11.01.2013

Table 5.2.4
Yearly Development Plan: OB-II Quarry
(Proposed Modification)

YEAR	Opencast/ Underground	OB Volume (L CuM)	ROM Processing $Cr_2O_3 \geq 30$ %		To Low Grade dump Cr_2O_3 < 30 %		Total ROM (LCuM)		Total Excavation (in LCuM)
			L CuM	LMT	L CuM	LMT	L CuM	LMT	
2011-12	Opencast	6.50	0.33	1.18	0.01	0.05	0.35	1.23	6.85
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	6.50	0.33	1.18	0.01	0.05	0.35	1.23	6.85
2012-13*	Opencast	5.63	0.22	0.76	0.01	0.04	0.23	0.80	5.85
	Underground	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.05
	Total	5.68	0.22	0.76	0.01	0.04	0.23	0.80	5.90
Total	Opencast	12.13	0.55	1.94	0.02	0.09	0.58	2.03	12.70
	Underground	0.05	0.00	0.00	0.00	0.00	0.00	0.00	0.05
	Total	12.18	0.55	1.94	0.02	0.09	0.58	2.03	12.75

*figure for 2012-13 is till 11.01.2013

The drawing for the yearly proposed mine entry development plan during the Scheme of Mining period as per this modification is enclosed as Drawing no. SCM/MP/16/12. The vertical section of the mine development proposed to be undertaken during the Scheme of Mining period as per this modification is enclosed as Drawing no. SCM/MP/21/12. The year wise mine development proposed during the ensuing scheme period is given below on Table 5.2.5.

Table 5.2.5
Yearly development plan: Underground infrastructures

Underground Development	Total (meters)	Development (meters)		Level mRL
Waste		2011-12	2012-13	
Decline North	2109	0	0	
Decline South	1021	0	300	77
Vertical Shaft	365	0	0	
Vertical & inclined airways	2997	0	0	
Total meterage	6492	0	300	

The year wise, pit wise and bench level wise, quantities of overburden / associated mineral to be handled is given below. The volumetric calculation has been done with help of SURPAC mine planning software. Volume has been derived using net volumes between two DTMs i.e DTM (digital Terrain Models) of the actual position before start the year and plan position after the completion of year.

2011-12: OB X QUARRY

Section	Bench Level		OB Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10 < 42$ %		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		LCuM	LMT	LCuM	LMT	LCuM	LMT	
2100N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	170	180	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	160	170	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	150	160	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	140	150	0.13	0.00	0.00	0.065	0.20	0.06	0.20	0.19
	130	140	1.36	0.00	0.00	0.065	0.20	0.06	0.20	1.43
	120	130	3.01	0.00	0.00	0.028	0.09	0.03	0.09	3.04
	110	120	5.50	0.00	0.00	0.262	0.80	0.26	0.80	5.76
	100	110	5.65	0.00	0.00	0.370	1.13	0.37	1.13	6.02
	90	100	5.59	0.00	0.00	0.415	1.27	0.41	1.27	6.01
	80	90	5.73	0.00	0.00	0.668	2.05	0.67	2.05	6.40
	70	80	5.54	0.04	0.13	0.592	1.81	0.63	1.95	6.17
	60	70	6.00	0.15	0.53	0.213	0.65	0.36	1.18	6.36
	50	60	4.42	0.20	0.74	0.248	0.76	0.45	1.50	4.87
	40	50	3.49	0.17	0.60	0.015	0.05	0.18	0.65	3.67
	30	40	2.72	0.19	0.68	0.000	0.00	0.19	0.68	2.91
	20	30	0.86	0.30	1.08	0.000	0.00	0.30	1.08	1.15
	10	20	0.00	0.00	0.00	0.000	0.00	0.00	0.00	0.00
	Total		52.00	50.00	1.04	3.77	2.94	9.00	3.98	12.77

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.92

2011-12: OB II QUARRY

Section	Bench Level		OB Volume (L CuM)	ROM Processing		To Low gr.dump		Total ROM (LCuM)		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 30$ %		Cr_2O_3 < 30 %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
1100N to 1400N & 3400E to 4000E	165	175	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	155	165	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	145	155	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	135	145	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	125	135	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	115	125	0.12	0.00	0.00	0.00	0.00	0.00	0.00	0.12
	105	115	0.35	0.00	0.00	0.00	0.00	0.00	0.00	0.35
	95	105	0.31	0.00	0.00	0.00	0.00	0.00	0.00	0.31
	85	95	0.39	0.01	0.02	0.00	0.00	0.01	0.02	0.40
	75	85	0.69	0.02	0.07	0.00	0.00	0.02	0.07	0.71
	65	75	1.08	0.04	0.15	0.00	0.00	0.04	0.15	1.12
	55	65	1.31	0.02	0.06	0.00	0.01	0.02	0.07	1.33
	45	55	1.15	0.07	0.26	0.01	0.02	0.08	0.28	1.23
	35	45	0.97	0.09	0.30	0.01	0.02	0.09	0.32	1.06
	25	35	0.14	0.09	0.31	0.00	0.00	0.09	0.31	0.23
	Total		6.50	0.33	1.18	0.01	0.05	0.35	1.23	6.85

Stripping Ratio = Overburden (CuM) / ROM (MT) = 5.30

2012-13 (Till 11.01.2013): OB X QUARRY

Section	Bench Level		OB Volume (L CuM)	ROM to Stackyard		Beneficiable Grade		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 42$ %		$Cr_2O_3 \geq 10 < 42$ %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
2100N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	2.87	0.00	0.00	0.00	0.00	0.00	0.00	2.87
	210	220	3.21	0.00	0.00	0.00	0.00	0.00	0.00	3.21
	200	210	3.49	0.00	0.00	0.00	0.00	0.00	0.00	3.49
	190	200	3.50	0.00	0.00	0.00	0.00	0.00	0.00	3.50
	180	190	3.20	0.00	0.00	0.00	0.00	0.00	0.00	3.20
	170	180	2.64	0.00	0.00	0.00	0.00	0.00	0.00	2.64
	160	170	2.77	0.00	0.00	0.00	0.00	0.00	0.00	2.77
	150	160	3.10	0.00	0.00	0.00	0.00	0.00	0.00	3.10
	140	150	2.70	0.00	0.00	0.05	0.16	0.05	0.16	2.75
	130	140	3.08	0.00	0.00	0.04	0.13	0.04	0.13	3.13
	120	130	2.03	0.00	0.00	0.02	0.06	0.02	0.06	2.05
	110	120	2.30	0.00	0.00	0.00	0.01	0.00	0.01	2.31
	100	110	1.52	0.00	0.00	0.01	0.04	0.01	0.04	1.53
	90	100	2.54	0.00	0.00	0.04	0.12	0.04	0.12	2.58
	80	90	0.00	0.00	0.00	0.06	0.17	0.06	0.17	0.06
	70	80	0.00	0.03	0.10	0.22	0.66	0.25	0.77	0.25
	60	70	0.00	0.10	0.37	0.55	1.68	0.65	2.04	0.65
	50	60	0.00	0.23	0.84	0.59	1.80	0.82	2.63	0.82
	40	50	0.00	0.24	0.87	0.49	1.50	0.73	2.37	0.73
	30	40	0.00	0.18	0.65	0.02	0.06	0.20	0.71	0.20
	20	30	0.00	0.37	1.33	0.00	0.00	0.37	1.33	0.37
	10	20	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.03
	Total		39.00	1.14	4.15	2.09	6.39	3.23	10.54	42.24

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.77

2012-13 (till 11.01.2013): OB II QUARRY

Section	Bench Level		OB Volume (L CuM)	ROM Processing		To Low gr.dump		Total ROM (LCuM)		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 30$ %		$Cr_2O_3 < 30$ %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
1100N to 1400N & 3400E to 4000E	165	175	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	155	165	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	145	155	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	135	145	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	125	135	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	115	125	0.14	0.000	0.000	0.000	0.000	0.000	0.000	0.14
	105	115	0.40	0.000	0.000	0.000	0.000	0.000	0.000	0.40
	95	105	0.35	0.000	0.000	0.000	0.000	0.000	0.000	0.35
	85	95	0.45	0.015	0.054	0.000	0.008	0.015	0.054	0.47
	75	85	0.79	0.041	0.146	0.000	0.022	0.041	0.146	0.84
	65	75	0.97	0.000	0.000	0.000	0.000	0.000	0.000	0.97
	55	65	1.20	0.005	0.018	0.009	0.003	0.005	0.018	1.21
	45	55	0.85	0.012	0.042	0.004	0.006	0.012	0.042	0.86
	35	45	0.45	0.155	0.547	0.010	0.082	0.155	0.547	0.61
	25	35	0.10	0.000	0.000	0.001	0.000	0.000	0.000	0.10
	Total		5.71	0.229	0.807	0.024	0.121	0.229	0.807	5.95

Stripping Ratio = Overburden (CuM) / ROM (MT) = 7.070

5.3 YEARLY PRODUCTION PLAN:

The earlier proposal for despatchable/ marketable chrome and pyroxenite ore production as approved in the Scheme of Mining for the period from 2011-12 to 2012-11.1.2013 is given in Table 5.3.1.

Table5.3.1
Approved Yearly Production Plan
(All figures in MT)

Year	OB X (friable)	OB II (lumpy)	Pyroxenite
2011-12	607530	191852	50000
2012-13*	512193	161745	0
TOTAL	**1119723**	**353597**	**50000**

Now as per this modification to the scheme of mining, the directly despatchable / marketable processed chrome and pyroxenite ore proposed for the next 2 years from 2011-12 to 2012-13 from the different quarries as planned is given in Table 5.3.2 below. Existing mineral rejects in the form of fines as well as pyroxenite boulders is lying outside the quarry shall be crushed for the necessary production in 2011-12 as approved as per Modified Scheme of Mining. The location of the pyroxenite boulders has been shown in the Surface Plan i.e SCM/MP/02/12. The change in production in chrome ore due to both opencast as well as underground mining has also been detailed.

Table 5.3.2
Year wise Finished/ Marketable grade ore Production Plan
(All figures in MT)

Year	**OB X** ($Cr_2O_3 \geq 42\%$)			**OB II** ($Cr_2O_3 \geq 33\%$)			**Pyroxenite** ($MgO \geq 28\%$)		
	Open Cast	Under ground	Total	Open Cast	Under ground	Total	Open Cast	Under ground	Total
2011-12	358020	0	358020	109980	0	109980	25000	0	25000
2012-13*	394434	0	394434	79866	0	79866	0	0	0
TOTAL	752454	0	752454	189846	0	189846	25000	0	25000

* Production figure for 2012-13 is till 11.01.2013

It is proposed that about 1.0 lakh MT of the total hard lumpy ore production from OB II as given above shall be produced by segregation from the old low-grade stacks as well as fresh generation of low-grade ores from OB II quarry. However, since the chrome industry is highly fluctuating the volumes of production may vary depending upon the market conditions and progress of proposed underground developments which is also linked to various statutory clearances. Moreover, as the mine caters to a large number of industries the quantities may go down or increase marginally.

The ROM production required from the respective quarries through both opencast and underground means for the above finished ore production and the yearly generation of beneficiable grade ore incidental to the above production shall be as follows as given in Table 5.3.3.1 & 5.3.3.2:

Table 5.3.3.1
OB-X Quarry Opencast (all figures in MT)

Year	ROM to Stackyard ($Cr_2O_3 \geq 42\%$)		Beneficiable Grade ROM ($Cr_2O_3 \geq 10\% < 42\%$)	Total ROM
	Friable ore Processing Plant	Manual Dressing & Sorting		
2011-12	369326	7537	900000	1276863
2012-13*	406890	8304	638889	1054083
Total	776216	15841	1538889	2330946

* ROM Production figures for 2012-13 is till 11.01.2013

Table 5.3.3.2
OB-II Quarry Opencast (all figures in MT)

Year	ROM to Lumpy Ore Processing Plant ($Cr_2O_3 \geq 30\%$)	ROM to Low Grade Stacks / dumps (Cr2O3<30%)	Total ROM
2011-12	117746	5000	122746
2012-13*	76313	3750	80063
Total	194059	8750	202809

* ROM Production figures of lumpy ore for 2012-13 is till 11.01.2013

The year wise pit wise level mine ROM generation from the both the quarry together is given below in Table 5.3.4.

Table 5.3.4
Year wise level wise total ROM generation

ROM Quantity (Lakh MT)									
Bench Level		**OB-X**			**Bench Level**		**OB-II**		
Bottom RL	**Top RL**	**2011-12**	**2012-13**	**TOTAL**	**Bottom RL**	**Top RL**	**2011-12**	**2012-13**	**TOTAL**
150	160	0.00	0.00	0.00	165	175	0.00	0.00	0.00
140	150	0.20	0.16	0.36	155	165	0.00	0.00	0.00
130	140	0.20	0.13	0.33	145	155	0.00	0.00	0.00
120	130	0.09	0.06	0.14	135	145	0.00	0.00	0.00
110	120	0.80	0.01	0.82	125	135	0.00	0.00	0.00
100	110	1.13	0.04	1.17	115	125	0.00	0.00	0.00
90	100	1.27	0.12	1.38	105	115	0.00	0.00	0.00
80	90	2.05	0.17	2.21	95	105	0.00	0.00	0.00
70	80	1.95	0.77	2.71	85	95	0.02	0.05	0.08
60	70	1.18	2.04	3.23	75	85	0.07	0.15	0.22
50	60	1.50	2.63	4.13	65	75	0.15	0.04	0.19
40	50	0.65	2.37	3.02	55	65	0.07	0.05	0.13
30	40	0.68	0.71	1.39	45	55	0.28	0.10	0.38
20	30	1.08	1.33	2.40	35	45	0.32	0.72	1.04
10	20	0.00	0.00	0.00	25	35	0.31	0.00	0.31
Total		**12.77**	**10.54**	**23.31**	**Total**	**0.00**	**1.23**	**1.12**	**2.34**

* ROM Production figures of OB-X & OB-X quarry for 2012-13 is till 11.01.2013

There would be no incidental production from underground means during above period. There would be no excavation in the pyroxenite section of OB-II quarry during above period. However, existing boulders shall be crushed in the processing plant and existing fines shall be used to produce 25000 MT of pyroxnite ore during 2011-12. The location of the existing pyroxenite boulders is shown in the surface plan as LG dump south of the pyroxinite section.

5.4 DEPLOYMENT OF MACHINERY:

There is no change in the proposed method of opencast mining and deployment of machinery as has been already indicated in the approved mining plan under para 4.6. Since the total volume of handling shall almost remain the same. The existing fleet and the adequacy of the proposed fleet as mentioned in the approved Mining Plan and Scheme of Mining for the major equipments viz. shovels and / backhoes, dumpers and drills in Table 5.4.1shall remain same. There is no change proposed for deployment of machinery for opencast mining.

For underground mine development during the plan period, trackless method of underground mining is being proposed. The proposed equipments for underground operation is as given below in Table-5.4.2.

Table-5.4.2
List of Underground Equipment

Equipment	OB-X	OB-II	Total	Make	Capacity
Twin Boom Drill Rig	1	1	2	Sandvik/ Atlas Copco	76-127mm
Single Boom Drill Rig	1	1	2		
Rock Bolter	1	1	2		33-45mm
Development LHD (5m³)	2	2	4		4-7 cum
Primary Dump Truck(30t)	2	1	3		15 - 20 cum
Shotcreter	1	1	2		
ANFO Loaders	1	1	2		
Service Vehicles	4	3	7		
Total	**13**	**11**	**24**		

6.0 HANDLING OF WASTE / SUB GRADE MATERIAL:

Year wise generation of Associated Minerals / overburden and waste from the different quarries by opencast mining & underground and build up of dumps are given below in table 6.0.1. There would be generation of waste during development of decline and shaft in OB-II quarry in 2012-13 in course of underground mining.

Table 6.0.1
Generation of Overburden & Waste (Lakh Cum)

Year	OB X Quarry (Ni-Limonite)			OB II Quarry (Serpentinite & Quartzite)			OB-X & OB-II		
	Opencast	Underground	Total	Opencast	Underground	Total	Opencast	Underground	Total
2011-12	50.0	0.00	50.0	6.5	0.00	6.50	56.5	0.0	56.5
2012-13*	39.0	0.00	39.0	5.6	0.05	5.68	44.6	0.1	44.7
Total	89	0.0	89	12.1	0.05	12.18	101.1	0.1	101.2

* OB generation figure for 2012-13 is till 11.01.2013

There is no modification being proposed in the definition of waste, mineral rejects and their grade/ quality.

6.1 Generation and Disposal of Waste:

There is no space available within the leasehold area for overburden dumping. The holding capacity of the rest of area available in additional area of 100 ha has been estimated to be 176 lakh cum as on 31.03.2011. The overburden generated during the planned period shall continue to be stored in the allotted additional area of 100ha during the plan period till 11.03.2013. Permission for last phase of 25 ha is at its last stage. Dumping can smoothly continue in 75ha till this 25 ha is available. The dump / stack plan and sections have been indicated in the drawing no. SCM/MP/08/12.

The stripping ratios for the various quarries on the basis of conceptual planning is given in Table 6.1.1 below:

Table 6.1.1

Quarry	OB X Quarry	OB II Quarry
Stripping Ratio, (cum/t)	3.78	6.88

The yearly generation of overburden / associated mineral from the respective quarries proposed from 2011-12 till 11.01.2013 for the year 2012-2013 is given in Table 6.1.2 & Table 6.1.3 below. As shown in the Table 6.1.2 given below, the likely generation of nickeliferous limonite for next two years till the scheme period shall be 89 Lakh Cum with an annual average of 50 Lakh Cum approximately. The yearly generation may vary due to changes in production requirements, which is largely governed by fluctuations in the market demand. However, efforts shall be made to maintain the total quantity for the 2 -year period in order to ensure proper development.

Table 6.1.2
OB-X Quarry

YEAR	(OPENCAST) PLAN			(UNDEGROUND) PLAN		TOTAL OB+ Waste CuM
	TOTAL ROM - MT	STR.RATIO	PROPOSED OB - CuM	TOTAL ROM - MT	PROPOSED Waste - CuM	
2011-12	1276863	3.92	5000000	0	0	5000000
2012-13	1054083	3.70	3900000	0	0	3900000
Total	2330946	3.82	8900000	0	0	8900000

Table 6.1.3
OB-II Quarry

YEAR	(OPENCAST) PLAN			(UNDEGROUND) PLAN		TOTAL OB+ Waste CuM
	TOTAL ROM - MT	STR.RATIO	PROPOSED OB - CuM	TOTAL ROM - MT	PROPOSED OB - CuM	
2011-12	122746	5.30	650000	0	0	650000
2012-13	80063	7.03	562500	0	5000	567500
Total	202809	5.98	1212500	0	5000	1217500

The yearly buildup of dumps from year to year has been shown in the form of plans and sections in drawing no. SCM/MP/08/12. Retreating method of dumping shall be followed wherever feasible depending upon the availability of space and nature of topography. No modification is proposed in the method of dumping and preventive measures to be adopted for reclamation and rehabilitation of dumps.

Chemistry and Mineralogy of existing Waste Dumps:

No modification is being proposed.

6.2 Generation and Stacking of Sub Grade material:

No modification is being proposed.

6.3 Sub Grade material available:

No modification is being proposed.

7.0 USE OF MINERAL :

Chromite is the only source of chromium which is used widely in metallurgical, chemical and refractory industries. The specification of chrome ore for various industries is tabulated as follows:

Sl. No.	Parameters	Beneficiation	Ferrochrome	Charge chrome	Refractory	Chemical
1	Cr_2O_3	>10%<42%	48% (min)	44%(min)	40%(min)	44% (min)
2	Cr : Fe		2.8:1 (min)	1.6 : 1(min)	---	
3	FeO		---	---	20%(max)	20% (max)
4	CaO		---	---	1%(max)	3% (max)
5	MgO		---	---	---	14% (max)
6	Al_2O_3		---	---	14%(max)	14% (max)
7	SiO_2		---	---	< 10%	7% (max)

The year-wise use of mineral expected for next two years is given in Table 7.0.1 below. The forecast given may change as the mine is catering to a number of industries and is dependent on the market conditions which changes very often within very short intervals of time. As shown in the table the requirement of chrome ore is expected to remain close to ~15 lakh MT..

Shortfall in requirement of flux mineral for the company shall be partially met from purchased pyroxenite ore from the Hata, Chaibasa region and the balance as dolomite ore from the existing Gomardih Dolomite quarry operated by the company in Sundergarh district of Orissa as approved in the modified Scheme of Mining.

Table – 7.0.1

Requirement of Chrome ore (All figures in lakh MT)

Purpose	Destination	Type of Ore	2011-12	2012-13
Captive	Bamnipal	Friable	1.06	1.15
		Lumpy	0.31	0.22
	Conversion	Friable	2.71	3.98
		Lumpy	0.79	0.78
	Beneficiation Plant	Friable	9.00	8.52
		Lumpy	0.00	0.00
	Jamshedpur Works and Joda	Friable	0.00	0.34
		Lumpy	0.00	0.07
	Refractory Plants	Friable	0.00	0.06
		Lumpy	0.00	0.00
	Overseas Project, South Africa	Friable	0.00	0.00
		Lumpy	0.00	0.00
Domestic Commitments		Friable	0.00	0.00
		Lumpy	0.00	0.00
Export Commitments		Friable	0.00	0.00
		Lumpy	0.00	0.00
Total		Friable	12.77	14.05
		Lumpy	1.10	1.06
Grand Total (Friable and Lumpy)			**13.87**	**15.12**

The company since long has also been registered as a primary mining company under para 3 (b) of the Memorandum and Articles of Association. Accordingly, the company besides meeting its own captive requirements had been in the business of sale of chrome ore and concentrates since long both for other domestic consumers as well as export. However, the company over the years has increased its captive consumption manifold and continues to do so by progressively adding and modernising both its downstream plant capacities and port handling facilities. The company has also been doing further value addition of its chrome ore produce through other plants under conversion agreements and also provides other raw material requirements such as coke for the same. Thus, while the company aspires to achieve the future end use of the chrome ore and concentrates it produces from this mine as given above under Table 8.1.1, it may under certain very special circumstances - while still remaining within the legal framework and rights, resort to the sale and export of chrome ore and concentrates of certain quantities due to either prevalent force majeure conditions, unforeseen market forces or technical difficulties; in the interest of business continuity/ sustenance and overall benefit to the economy and country. Some of such conditions may be:

- Severe recession in the market leading to the non-viability and closure of the company's captive plants and other plants under conversion agreement – leading to huge build-up of stocks of ore and concentrates;
- Directives from the Government for the sale of ore and concentrates to other consumers in the event of non-availability of the mineral from other domestic sources and for the overall benefit of the country - like defence purposes, or meeting such other economic and livelihood interests of the local community;
- Delays in the commissioning of the new downstream captive plants due to non-availability of land, power, water or other resources - leading to build up of huge stocks of mineral at the mine and stockyards;
- Stoppage of any of the captive downstream plants or plants under conversion agreement, over extended periods of time due to certain force majeure conditions (like strike, lockout, earthquake, flood, transport bottlenecks, blockades, other natural and accidental calamities, shortfall in electrical power, major breakdowns & non-availability of spares, etc.) – leading to build up of huge stocks of ore and concentrates at the mine and stockyards;
- Past long-term commitments of the company with other consuming plants in India & abroad that have been set up with technology that was developed based on specific quality parameters (both physical and chemical) of the ore and concentrates produced and available only from this mine;
- Absolute non-availability of ore at a reasonable price for operating the plants owned by the company in other countries, which may lead to closure of such plant;
- Build-up in stocks of certain specific grades and quality of chrome ore and concentrates which cannot be consumed by the company in its own captive plants due to certain technical limitations in the plant & process capabilities and/ economic non-viability.

7.1 Changes proposed in the use of mineral:

No modification is being proposed.

7.2 Changes in specifications:

No modification is being proposed.

7.3 Efforts made for utilization of sub-grade mineral including fines:

The efforts made in the use of sub-grade mineral have been elaborated under para 8.1 below. Further continuous improvements have been achieved in the Chrome Ore Beneficiation Plant with respect to operational and preventive maintenance procedures. This has been possible by giving a constant thrust on training for up gradation of the skills of all employees as well as greater involvement of the people at the grass roots level in improvement projects. These small incremental improvements over the last 5 years have helped in enhancing the utilization largely. Both the mine and the plant have been certified to standards under ISO-9002 Quality Management Systems, ISO- 18001 Occupational Health & Safety Management System and ISO-14001 Environment Management Systems. This has helped in standardizing all procedures and work instructions at all levels. Regular audits are conducted to ensure that all work is carried out as per the documented procedures and work instructions. The above efforts have all helped in enhancing the availability and utilization of the plant to world class levels. This has eventually resulted in improved production and recoveries over the years as given in Table 7.2.1 below.

Table 7.2.1

YEAR	Feed grade % Cr2O3	Feed quantity lakh MT (dry)	Concentrate grade % Cr_2O_3	Concentrate Production lakh MT (dry)	Tailings grade, % Cr_2O_3	Tailings quantity lakh MT (dry)	Yield % by weight
2003-04	37.05	8.76	55.47	4.34	18.57	4.42	49.54
2004-05	35.78	9.04	53.09	4.28	20.47	4.76	47.36
2005-06	36.91	8.95	51.28	5.07	14.30	3.81	56.67
2006-07	34.77	8.53	50.63	4.66	16.41	3.87	54.65
2007-08	32.51	7.64	47.96	3.79	16.68	3.85	49.60

The COB plant performance for the Scheme period 2008-11 is given in Table 7.2.2 below. The same has also been given in the review chapter.

Table 7.2.2

YEAR	Feed grade % Cr_2O_3	Feed quantity lakh MT (wet)	Concentrate grade % Cr_2O_3	Concentrate Production lakh MT (wet)	Tailings grade, % Cr_2O_3	Tailings quantity lakh MT (wet)	Yield % by weight
2008-09	32.52	6.21	49.98	3.04	15.8	3.17	48.92
2009-10	32.30	7.50	50.69	3.67	14.67	3.83	48.94
2010-11	33.55	8.87	49.97	4.82	14.00	4.05	54.36

The above tonnage figures on dry basis in order to have a like to like comparison with the proposal in the approved Scheme of Mining are given in Table 7.2.3 below.

Table 7.2.3

YEAR	Feed grade % Cr_2O_3 (dry)	Feed quantity lakh MT (dry)	Concentrate grade % Cr_2O_3 (dry)	Concentrate Production lakh MT (dry)	Tailings grade, % Cr_2O_3 (dry)	Tailings quantity lakh MT (dry)	Yield % by weight
2008-09	32.52	5.46	49.98	2.67	15.8	2.79	48.92
2009-10	32.30	6.60	50.69	3.23	14.67	3.37	48.94
2010-11	33.55	7.80	49.97	4.24	14.00	3.56	54.36

The beneficiation process was last modified in the year 2005-06 with the introduction of floatex density separators, additional spiral banks, vacuum belt filter and high frequency screens. This has helped us in reducing the tailings losses from a level of ~ 20% Cr_2O_3 in 2004-05 to a level of ~ 14% Cr_2O_3 in the year 2010-11.

In order to further bring down the tailings losses to the targeted level of < 10% Cr_2O_3, the Company has undertaken pilot scale studies for tailings beneficiation in collaboration with IIMT, Bhubaneswar; DRA and Mintek, South Africa together with our own R&D Dept. at Jamshedpur. Based on such studies the Company has a proposal to set up the tailings beneficiation circuit by 2014-15. The details in this regard have been further incorporated in the Mining Plan for the subsequent years. The tailings shall continue to be stored separately so that the same can be reutilized to recover the chrome values. Copy of the relevant portion of the report is enclosed as Annexure-38.

Thus, no modification in the process as such is envisaged during the current Scheme of Mining period.

The year wise proposed figures from 2011-12 till 11.01.2013 for the year 2012-2013 is given in Table: 7.2.4 below.

Table: 7.2.4

Year	FEED GRADE % Cr2O3	FEED QTY DMT	CONC. GRADE % Cr2O3	CONC. PROD. DMT	TAILING GRADE % Cr2O3	TAILING QTY DMT	YEILD % BY WT
2011-12	33-35	819142	48-51	442336	13 - 15	376806	53-55
2012-13*	33-35	591051	48-51	319167	13 - 15	271884	53-55
Total	32-35	1410193	48-51	761503	13 - 15	648690	53-55

*The figure for 2012-13 is from 01.04.2012 to 11.01.2013

8.0 MINERAL BENEFICIATION:

The capacity of COB Plant was validated by the IBM after the modernization of the beneficiation circuit in 2005-06 to be 7.0 lakh tonnes/ annum. This report on capacity validation studies of the chrome ore beneficiation plant of October, 2005 (Report no. IBM/NGP/R.I. N.O. 1681) states that the plant can produce about 7 lakh tonnes/ year of chrome concentrate (dry basis) assaying about 50-52% Cr_2O_3 with weight % yield of 60-65%, by treating a ROM ore assaying 38-40% Cr_2O_3 and plant availability of 90% i.e. 7800 hours/ year, with a plant feed rate of 170 tph (wet basis) and 150 tph (dry basis).

Presently though, the ROM ore feed to the plant is assaying 33-35% Cr_2O_3. With this lower grade of ROM feed to the plant the feed rate is estimated to be around ~ 125-130 tph (dry basis). Further due to the lower feed grade, the weight % yield being achieved presently is ~ 53-55%.

Thus, considering the above, the plant capacity for the given conditions = 7800 x 127 x 0.54 = 5, 34,924 tonnes/ year, i.e. ~ 5.3 lakh tonnes/ annum of chrome concentrate (dry basis) assaying about 50-52% Cr_2O_3.

The water from the tailings is further proposed to be filtered by the introduction of parallel plate press filters as shown in Fig 8.2 below. 3 such units having a capacity of 27 t/ hr each is proposed to be installed. The tails after dewatering shall then be in cake form and shall then be transported in dumpers to the proposed disposal site. The schematic layout of the manner in which these parallel plate press filters shall be installed within the Chrome Ore Beneficiation Plant is given below in Fig 8.3.

The quantities of friable ore feed to the Mechanised Processing Plant and that for manual dressing & sorting has been given in Table 5.3.2. The quantities of various products expected on the basis of the recoveries are given in Table 8.0.1 below. The other details of feed grade, concentrate grade, recovery etc. expected has been mentioned earlier under Table 7.3.1. The figures in the Table 8.0.1, 8.0.2 & 8.0.3 for the year 2012-13 are till the date 11.01.2013.

Table 8.0.1
Material Balance for OB-X Ore
(all figures in MT)

Year	ROM to Friable Ore Processing Plant (+42%)	ROM to Stackyard for Manual Dressing & Sorting (+42%)	PROD. MANUAL PROCESS		PRODN. MECH. PROCESS			TOTAL FINISHED ORE
			-75 mm +0 mm	MINERAL REJECT	-75 mm + 10 mm	- 25 mm	MINERAL REJECT	
2011-12	324985	81246	77184	4062	97496	211240	16249	385920
2012-13	267992	66998	63648	3350	80397	174195	13400	318240
Total	592977	148244	140832	7412	177893	385435	29649	704160

The expected feed quantities of beneficiable grade ore to the Chrome Ore Beneficiation Plant (COBP) from various sources and expected concentrate production is as given in Table 8.0.2 below. The beneficiable grade ore (BG) is stacked at earmarked location which necessarily acts as blending yard to prepare feed for the COB plant. The location of the BG stacks is marked in the Surface Plan i.e SCM/MP/02/12. The final concentrate production is given in dry basis. However the despatchable concentrate in wet basis may be estimated adding 12% moisture in it.

Table 8.0.2
Beneficiable grade (BG) ore feed for Concentrate production
(All figures in MT)

Year	Plan - Mt								
	BG ROM From Quarry ($Cr_2O_3 \geq 10$ % < 42 %)	Mineral Reject To BG Stacks	Feed From Old BG Stock ($Cr_2O_3 \geq 15$ % < 30 %)	Total Feed To COBP (Wet) ($Cr_2O_3 \geq 33$ % < 35 %)	Mois ture (%)	Total Feed To COBP (Dry)	Prod. Conc. (Dry)	Prod. Conc. (Wet)	Grade Conc. (Cr2O3%)
2011-12	900000	20312	12000	932312	12	820434	402013	450254	47-50
2012-13*	825000	16749	12000	853749	12	751300	368137	412313	47-50
Total	1725000	37061	24000	1786061	12	1571734	770150	862567	47-50

*The figure for 2012-13 is till 11.01.2013

The yearly quantity of feed to Lumpy Ore Processing Plant (LOPP) and the total marketable lumpy ore production including the production through manual segregation from the old low-grade ore stacks based on the recovery is given in Table 8.0.3 below. The ROM generated having more than 10% Cr_2O_3 from the OB-II quarry is directly fed to the LOPP plant to get +10mm product which is booked as marketable grade production. Contact zone material having less than 10% Cr_2O_3 is stores separately in low grade stack. Efforts are made to recover the hard lumpy chromite ore from this low grade stack by manual breaking, dressing & sorting which reflects under the segregation column. The lumpy fines shall be stacked at earmarked location and efforts shall be made to market those or reuse by blending.

Table 8.0.3
Material Balance for OB-II Ore
(All figures in MT)

Year	FEED TO LOPP +10% Cr2O3	PROD.FROM PLANT			SEGRE GATION	TOTAL LUMPY FINISHED ORE
		+10 mm	MINERAL REJCT -10 mm	SLIME LOSS		
1	2=3+4+5	3	4	5	6	7=3+6
2011-12	64094	54480	6409	3205	30000	84480
2012-13*	56541	48060	5654	2827	22500	70560
Total	120635	102540	12063	6032	52500	155040

*The figure for 2012-13 is till 11.01.2013

However there would be change in tailing dam management process as described below:

Tailing dam management:

There is a dirth of space for disposal of tailings in the existing tailing pond. With the reduction of the mining lease area in 1996 during its second renewal, the tailing pond went outside the granted area of 406 ha. A new tailing pond was established within the reduced lease covering an area of 10 ha. The tailing dam is being heightened every year to accommodate the tailing generated from the COB plant which has reached its ultimate capacity now.

Company has therefore has identified and applied for an additional area of 73.685 ha having 65.315 ha of non-forest Government land for future storage of the tailing which is under process. It may take a further period of at least two years to acquire this additional land.

As per approved Modified Scheme of Mining the tailing generated from the COB Plant shall continued to be stored in the pyroxenite section. Embankment shall be made all around the pit to prevent any spillage of water. The water shall be re-circulated back to the plant by continuous pumping. Once we get the applied additional area, we shall re-handle the tailing from the pit and shall store the same in the new area. We shall reopen the pit for pyroxenite ore mining as soon as we re-handle the tailing.

Fig. 8.2: Photograph of the parallel plate press filter

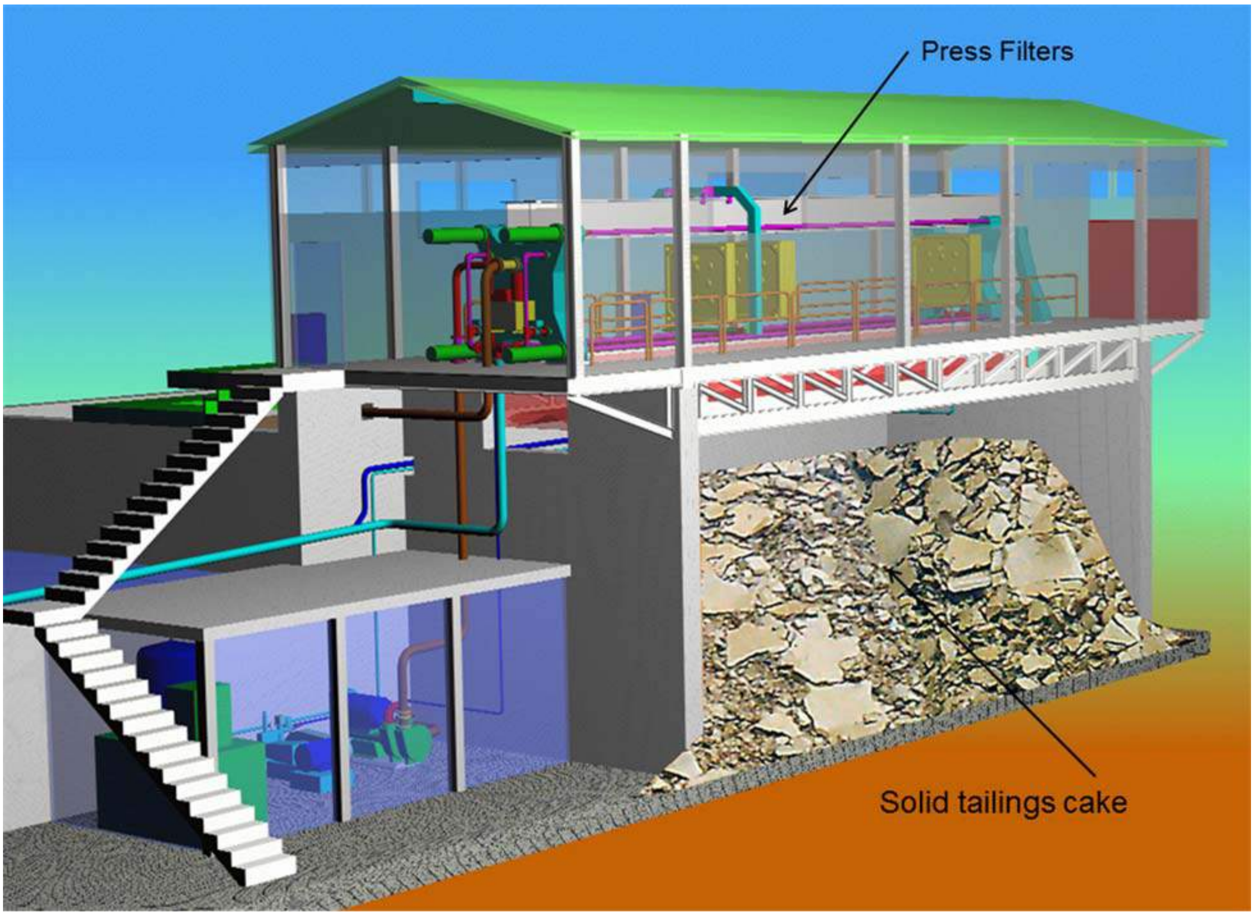


Fig. 8.3: Schematic Layout of the solid tailings disposal system

9.0 ENVIRONMENT MANAGEMENT PLAN:

As discussed in conceptual mining plan of approved Scheme of Mining, afforestation by planting of indigenous species of different varieties shall be done on the slope area of the dump. The year-wise proposal for plantation within the leasehold area and in the additional area of 100 Ha along the slopes of associated mineral stacks shall undergo minor changes due to space available for plantation. The year wise plantation proposed on dumps, vacant land etc with in the Mining Lease area of 406ha and at additional area of 100ha is given below in Table no. 9.1.1 and is as shown in drawing no. SCM/MP/10/12.

Table no. 9.1.1
Year wise Afforestation/ Plantation Programme

Year	Within the Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Total Plantation		
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)
2011-12	4.5	11250	3.50	8750	8.00	20000	16.0
2012-13	1.9	4625	8.55	21375	10.40	26000	20.8
TOTAL	6.4	15875	12.05	30125	18.4	46000	36.8

The coir matting shall be done in the Ni- limonite overburden dump of OB-X quarry where the chances of wash off is more but may not be possible in overburden dump of OB-II quarry due to its very hard nature and sharp edges of overburden material.

Phased reclamation plan:

As mentioned, afforestation can only be taken up after stacks reach a certain level of maturity (i.e. extended up to the final stages along the boundary of the allotted area). The year wise programme for construction of the retailing wall at the toe of the stacks along the boundary of the allotted area as shown in the drawing no. SCM/MP/10/12. is given below in Table 9.1.2

Table 9.1.2
Year wise proposed of retaining Wall in 100Ha.

Year	Retaining Wall length (Mtr.)	Proposed Expenditure (Rs Lakh)
2011-12	1000	15.00

The construction of retaining wall shall be done as required after studying the topography and drainage pattern along the periphery wherever required of the additional area of 100 Ha. The entire proposed retaining wall shall be constructed by the end of 2011-12.

The following additional environmental mitigation measures shall be undertaken as per the modified Scheme of Mining in pyroxenite section for temporary storage of tails during the plan period.

1. Earthen bundh shall be made all around the pyroxenite section to prevent any spillage or overflow of tails from the area.
2. Area shall be barricaded by barbed wire fencing to prevent inadvertent entry of persons and other stray animals.
3. Water from the storage area shall be recycled back and reused in Chrome ore beneficiation plant to ensure zero discharge of any process water.
4. Although water table has not been encountered in the pyroxenite section/ pit, horizontal holes shall be drilled from the OB-II quarry towards and below the pyroxenite section/ pit bottom level to further lower the water table in this part.
5. Monitoring of the ground water quality for its potability shall be undertaken by drawing samples from the existing tube wells within the lease.

The drawing showing the proposal of various environment safeguard measures in lease area of 406 ha, additional area of 100 ha for overburden dumping and additional area of 73.685 ha (65.315 ha of non-forest area+8.37 ha of forest area) for tailing disposal in future is as shown in drawing no. SCM/MP/10/12.

The status of approved management plan and the proposal for next two years is given below in Table 9.1.3.

Table-9.1.3

Salient Items	**Proposal As Per Approved Mining Plan**	**Position At The End Of 3 Years Of Mining Plan Period**	**Proposal For The Next Two Year Plan Period**
Top Soil Storage, Preservation & Utilization	Area already exposed is devoid of top soil. No top soil storage	No fresh area is broken during last five years & hence there is no top soil stored.	Area is devoid of top soil & hence no chance of getting top soil in future from the mining lease.
Land Reclamation & Rehabilitation	Proposal for pit reclamation by afforestation and garland drain at the toe.	No matured pit was formed. However, for OB dumps to prevent soil erosion from the dump slops and increase in slope stability coir mats were spread followed by vegetation, provision of garland drains at the toe of the dump.	The part of OB II quarry is proposed to be reclaimed by back filling. The dead dumps will be progressively rehabilitated by vegetation during 2010-13**.** The area to be so rehabilitated has been shown in drawing no SCM/MP/10/12.
Waste Dump Management	Stabilisation of dead dumps slopes.	Over Burden dumps which were inactive have been vegetated by putting coir mats on slope. Garland drains at the toe with settling pits provided.	Dead dumps will be reclaimed by vegetation and the wash outs will be arrested by garland drains & settling pits. The year-wise no. of saplings and areas of the dumps proposed has been mentioned in the next para and also shown in the drawing no. SCM/MP/10/12.

<table>
<tr><th>Salient Items</th><th colspan="3">Proposal As Per Approved Mining Plan</th><th colspan="3">Position At The End Of 3 Years Of Mining Plan Period</th><th colspan="3">Proposal For The Next Two Year Plan Period</th></tr>
<tr><td rowspan="10">Afforestation Programme With Precaution Proposed</td><td colspan="3">During 2008-2013 86250 saplings were proposed to be planted covering 34.50 ha area inside the lease area (406 ha.) and in additional area of 100 ha.</td><td colspan="3">A total of 291000 nos. of sapling were planted covering 22.7 Ha inside the lease area (406 ha.) and in additional area of 100 ha during 2008-09 to 2010-11.</td><td colspan="3">There is a proposal to plant 44375 saplings during 2011-13 inside the lease area (406 ha.) and in additional area of 100 ha covering 17.75 ha. This includes rehabilitation of the dead dumps. The proposed filling of pits with sweet earth, cow-dung, etc. and post plantation activities like hoeing, weeding, watering will continue.</td></tr>
<tr><td colspan="3">Proposal break-up</td><td colspan="3">Plantation break-up</td><td colspan="3">Proposal break-up</td></tr>
<tr><td>Year</td><td>Nos</td><td>Area (ha)</td><td>Year</td><td>Nos</td><td>Area (ha)</td><td>Year</td><td>Nos</td><td>Area (ha)</td></tr>
<tr><td>2008-09</td><td>11750</td><td>4.70</td><td>2008-09</td><td>122000</td><td>4.7</td><td></td><td></td><td></td></tr>
<tr><td>2009-10</td><td>21750</td><td>8.70</td><td>2009-10</td><td>84000</td><td>9.5</td><td></td><td></td><td></td></tr>
<tr><td>2010-11</td><td>20000</td><td>8.00</td><td>2010-11</td><td>85000</td><td>8.5</td><td></td><td></td><td></td></tr>
<tr><td>2011-12</td><td>18250</td><td>7.30</td><td></td><td></td><td></td><td>2011-12</td><td>20000</td><td>8.00</td></tr>
<tr><td>2012-13</td><td>14500</td><td>5.80</td><td></td><td></td><td></td><td>2012-13</td><td>26000</td><td>10.40</td></tr>
<tr><td colspan="3"></td><td colspan="3">With all the precautions 90% survival rate has been achieved in the three years.</td><td colspan="3">Adequate plant protection and management measures shall be undertaken for maintaining the plants.</td></tr>
<tr><td colspan="9"></td></tr>
<tr><td rowspan="3">Quality Of Air</td><td colspan="3" rowspan="3">No adverse impact on Ambient Air Quality was expected</td><td colspan="3">The monitoring data during 2008-2011 indicates that the ambient air quality is well within the prescribed limit. The monitoring has been done by our consultant. Data is enclosed in Annexure-9</td><td colspan="3">Ambient air quality will be monitored at a frequency of twice a week by conducting 24 hour sampling. The location of monitoring stations are shown in drawing. No. SCM/MP/09/12.</td></tr>
<tr><td colspan="3">Water is being sprinkled on roads for dust suppression.</td><td colspan="3">All these measures will continue to be maintained.</td></tr>
<tr><td colspan="3">Avenue plantation, wet drilling, wet ore processing, construction and maintenance of pucca roads has reduced the air borne particulate matter.</td><td colspan="3"></td></tr>
</table>

Salient Items	Proposal As Per Approved Mining Plan	Position At The End Of 3 Years Of Mining Plan Period	Proposal For The Next Two Year Plan Period
Treatment Of Mine Water And Recirculation	Treatment of mine water for hexavalent chromium and removal of sludge in horizontal roughing filter.	The monitoring data during 2008-11 shows that Cr+6 content in mine discharge water (ETP outlet) is nil in most of the times and has never gone beyond the permissible limit. The TSS in the outlet water was in the range of 15-25 mg/lit, which is well within permissible limit of 100 mg/l. All other water quality parameters were also well within the limits. The monitoring has been done by our consultant. Data is enclosed in Annexure-9.	Water will be treated in effluent treatment plants before discharging out of the lease hold area to ensure that all the water quality parameters are within the permissible limits. Water quality at lease end will continue to be monitored at a frequency of once a month for all water quality parameters as prescribed by the Orissa Pollution Control Board.
Quality Of Make - Up Water Including Surface & Ground Water	No water requirement as mining does not requires water except for dust suppression & colony & make -up water for COB P lant	Mine discharge is utilized for dust suppression and make-up water for Chrome Ore Beneficiation Plant. The process water is re-circulated in the plant. Since there will be no discharge out of lease from these sources the impact on surface/ ground water is not envisaged.	In future also mining operation will not require water except for dust suppression, township, and ore processing. In case of any discharge from any of the sources the same will be duly treated to ensure that it meets all the desired water quality parameters.

Salient Items	Proposal As Per Approved Mining Plan	Position At The End Of 3 Years Of Mining Plan Period	Proposal For The Next Two Year Plan Period
Noise Level And Vibration	No problem of nuisance of noise & vibration was expected	All the precautions such as delay blasting between hole to hole and row to row were taken to reduce vibration during blasting. Detailed noise survey of work zone and ambient air was carried out once in six months. The machines with noise level more than 90 dB were corrected as far as possible and the people working in noisy areas were provided with ear protection appliances. Green belt developed all around has been acting as a baffle to the noise transmission to the camp area. Strict preventive maintenance schedules for equipments operating in the mines & Chrome Ore Beneficiation plant are adhered to which helps in controlling the noise generation	In future, no change in the noise level is envisaged. Noise survey in the work zone will be done once in six months. Ambient noise monitoring will additionally be done at a frequency of once in six months.

Prepared by:

(GK Guin)

RQP
Regd.No: RQP/BBS/044/2003/A

Dated: 9th May' 2012

PART – III

PROGREIVE MINE CLOSURE PLAN
(FROM 2011-12 TO 11th JAN'2013)

10.0 INTRODUCTION :

a) Name of Lessee :

No modification is being proposed**.**

b) Location:

No modification is being proposed. The Latitude and Longitude of area is as Table 2.4.1 of Part-II.

c) Lease Area Breakup:

Same as Chapter- 2.4 of Part-II

d) Validity of the Lease :

Same as Chapter- 2.5 of Part-II

e) Present Land Use Pattern :

The location and extent of lease area, the type of lease area (forest, non-forest etc) and the present land use pattern is given in the in plan SCM/MP/12/12. The activity wise breakup of the land is given below in Table 10.1.1.

Table 10.1.1
Present Land Use

Sl No	Activity	Present Land use (Ha.)
1	Area to be excavated	199.77
2	Storage of Top Soil	0.00
3	Over Burden Dump	37.47
4	Mineral Storage	37.08
5	Infrastructure (Workshop, magazine etc)	3.27
6	Roads (Present L= 7812m, W= 25m)	19.05
7	Railways	0.00
8	Greenbelt	38.53
9	Tailing Pond	15.95**
10	Effluent Treatment Plant	0.32
11	Mineral Separation Plant	18.38
12	Township Area*	36.18
13	Others (to be specified)	0.00
	Total	406.00

* The township area includes the roads with in the township
** includes area for temporary storage of tailing in pyroxenite section

f) Method of Mining and Mineral Processing:

Same as Chapter-5 & Chapter -8 of Part-II respectively

10.1 REASON OF CLOSURE :

No modification is being proposed.

10.2 STATUTORY OBLIGATION :

No modification is being proposed.

10.3 CLOSURE PLAN PREPARATION :

a) Name of the applicant & address :

The applicant and the present nominated owner of the mine is Mr. H. M Nerurkar. The copy of photo ID of nominated owner along with passport details towards address proof is enclosed as Annexure-14. The details of address are as follows.

Mr. H. M Nerurkar
Managing Director
Tata Steel Ltd.

At. / P.O.	: Jamshedpur
Dist	: East Singhbhum
Jharkhand	: 831001
Phone	: 0657- 2423298
Fax	: 0657- 2431818
Email	: mdoffice@tatasteel.com

b) Status of applicant :Nominated Owner

c) The present members of the Board of Directors of TATA STEEL are:

Same as Chapter-1.4 of Part-I.

d) Name of the RQP preparing the Progressive Mine Closure Plan Address:

Mr. G K Guin
Head (MPP)
TATA STEEL Ltd. Sukinda

Dist	: Jajpur
State	: Orissa
Pin Code	: 755028
Phone	: 09238100835
Fax	: 06726268734

Registration No.: RQP/BBS/044/2003/A, valid till 03.02.2013 (The copy of the certificate is enclosed as Annexure-16). The RQP, Sri. G. K. Guin has been duly authorized by the applicant. The letter of consent is enclosed as Annexure -24. The Key Person has been assisted by the officers and staff of Planning Cell. The underground part of the above modification has been prepared by Haridrumat Behera, RQP, having Registration No. RQP/BBS/093/2010-A valid up to 19.04.2020 (Copy of RQP certificate enclosed as Annexure-17). Services from the departments of Geological Services, Land & Lease, Safety, Environment and the field personnel of Sukinda Chromite Mine were also availed in relevant areas of their expertise.

e) Name of executing agency:

No modification is being proposed.

11.0 MINE DESCRIPTION :-

11.1 Geology:

No modification is being proposed.

11.2 Reserve:

Same as Chapter-3 of Part-II.

11.3 Method of Mining:

Same as Chapter-5 of Part-II.

11.4 Mineral Beneficiation:

Same as Chapter-8 of Part-II

12.0 REVIEW OF IMPLEMENTATION OF MINING PLAN / SCHEME OF MINING INCLUDING FIVE YEARS PROGRESSIVE MINE CLOSURE PLAN UP TO THE

Afforestation:

The year wise proposal of afforestation for last three years and compliance thereof is as given below in Table 12.1.1.

Table 12.1.1
Year wise Afforestation

Year	Plan		Actual	
	Nos	Area (ha)	Nos	Area (ha)
2008-09	11750	4.70	122000	4.70
2009-10	21750	8.70	84000	9.50
2010-11	20000	8.00	85000	8.50
Total	53500	21.4	291000	22.7

Total 2150 m of retaining wall using earth, overburden material and boulders were made against the plan of 1700m in the year 2008-09.

Air & Water Quality Management:

As planned, ambient air quality is being monitored at a frequency of twice a week by conducting 24 hour sampling and all the parameters are found to be within prescribed limits. Similarly mine water is being treated in effluent treatment plants before discharging out of the lease hold area to ensure that all the water quality parameters are within the permissible limits Water quality at lease end will continue to be monitored at a frequency of once a month for all water quality parameters as prescribed by the Orissa Pollution Control Board. The year wise ambient air and water quality recorded during the period 2008-11 is enclosed as Annexure-9.

FINAL MINE CLOSURE PLAN:

No modification is being proposed.

13.0 CLOSURE PLAN :

13.1 Mined out Land:

The temporary change in land use is addressed under para 17 & Table: 17.1.

13.2 Water Quality Management:

Over and above the proposal as per the approved Scheme of Mining & Progressive Mine Closure Plan, monitoring of the ground water quality for its potability shall be undertaken by drawing samples from the existing tube wells within the lease.

13.3 Air Quality Management:

No modification is being proposed.

13.4 Waste Management:

a) Generation of Waste & Rejects in next planned period:

Same as Chapter 6.1 of Part-II.

b) Land chosen for disposal of waste with proper justification:

No modification is being proposed.

c) Build of Dumps:

Same as Chapter 6.1 of Part-II.

13.5 Top Soil Management:

No modification is being proposed.

13.6 Tailing Dam Management

A tailing dam exists within the lease area to store the tailings generated from the Chrome Ore Beneficiation Plant. There is a dirth of space for disposal of tailings in the existing tailing pond. With the reduction of the mining lease area in 1996 during its second renewal, the tailing pond went outside the granted area of 406 ha. A new tailing pond was established within the reduced lease covering an area of 10 ha. The tailing dam is being heightened every year to accommodate the tailing generated from the COB plant.

IIT, Kharagpur was engaged to assess the holding capacity of the tailing pond and the stability of the dam. In their report, IIT, Kharagpur has recommended for the tailing dam heightening to be done to a maximum height of 215 mRL. Presently the dam has already reached a height of 212 mRL. The present tailing dam has reached its ultimate capacity. Company has therefore has identified and applied for an additional area of 73.685ha having 65.315 ha of non-forest Government land for future storage of the tailing which is under process. It may take a further period of at least two years to acquire this additional land.

During such period the mining shall be temporarily discontinue for pyroxenite section of OB-II quarry in order to store the tailing generated from the COB Plant in the pit as per the approved modified Scheme of Mining. The water shall be re-circulated back to the plant by continuous pumping. Once we get the applied additional area, we shall re-handle the tailing from the pit and shall store the same in the new area. We shall reopen the pit for pyroxenite ore mining as soon as we re-handle the tailing.

Physical Stability:

No change is proposed in the management of existing tailing dam which has exhausted. Coir matting shall be done in its embankment for better stability and to prevent any erosion of slope. The tailing which shall be stored the pyroxenite section of OB-II Quarry shall be stored within the earthen embankment all around the pit. Water shall be regularly pumped to minimize any hydrostatic pressure.

Chemical Stability:

The existing tailing dam shall be left to dry naturally. However backup pumping arrangements shall be kept to dewater the monsoon water. Similarly water from the tailing stored in the temporarily discontinued pyroxenite section of OB-II quarry shall be tried to keep devoid of any excess water by continuous pumping. All the water pumped shall be reused in the Chrome Ore Beneficiation Plant. Thus, no water from the pit shall be allowed to be discharged out of the lease.

13.7 Infrastructure

No modification is being proposed.

13.8 Disposal of Mining Machinery

No modification is being proposed.

13.9 Safety & Security

No modification is being proposed.

13.10 Disaster Management and Risk Assessment

No modification is being proposed.

Fire/ Explosion:

No modification is being proposed.

Accidents to Man & Machine:

No modification is being proposed.

13.11 Care and Maintenance during Temporary Discontinuance

No modification is being proposed.

14.0 ECONOMIC REPERCUSSIONS OF CLOSURE OF MINE & MANPOWER RETRENCHMENT:

No modification is being proposed.

15.0 TIME SCHEDULING FOR ABANDONMENT:

Time scheduling for final abandonment is not envisaged under the scope of Progressive Mine Closure Plan, will be duly submitted during Final Mine Closure Plan. Over and above the proposals as per the approved Scheme of Mining for the current plan period, the earthen bundh and fencing around the pyroxenite section has already provided and the water recirculation system has been established as per the proposal given in approved modified scheme of mining.

16.0 ABANDONMENT COST:

The abandonment cost as per the approved Scheme of Mining for the current period had been estimated to be around Rs 5.4 crores.

The cost of recommissioning of the pyroxenite section/ pit shall involve the re-handling and transportation cost of the tails which is being proposed to be temporarily stored in this area as per this modification. It is estimated that the maximum quantity of tails that shall be stored during the period of temporary discontinuance of pyroxenite mining and this current plan period till 2013 shall be around 10 lakh tonnes. This is equivalent to 6.25 Lakh Cum. The current rate for evacuation and transportation is Rs 54.70 per cum and Rs 12.80 per Cum.km. Considering a 10% escalation during the forthcoming 2 year period and transportation distance of 6km, the cost of recommissioning of the pyroxenite section/ pit shall be around Rs 9.40 crores.

The amount of reclamation and rehabilitation job done along with the cost incurred during the scheme period from 2008-09 to 2010-11 is given in Table 16.1.1.

The amount of reclamation and rehabilitation job planned for rest of the planned period till 2013 along with the cost to be incurred in given below in Table 16.1.2 & 16.1.3.

Table 16.1.2
Year wise Afforestation Programme

Year	Within the Mining Lease Area of 406 ha.			Additional Area of 100 ha.			Total Plantation		
	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)
2011-12	4.5	11250	9.00	3.50	8750	7.00	8.00	20000	16.0
2012-13	1.9	4625	3.70	8.55	21375	17.10	10.40	26000	20.8
TOTAL	6.4	15875	12.7	12.05	30125	24.1	18.4	46000	36.8

Table 16.1.3
Year wise Toe Wall Construction Programme

Year	Within the Mining Lease Area of 406 ha.		Additional Area of 100 ha.	
	Length (m)	Cost (Rs Lakh)	Length (m)	Cost (Rs Lakh)
2011-12	0	0	1000	15.0
2012-13	0	0	0	0.0
TOTAL	0	0	1000	15.0

Table 16.1.1

Item	Details	Area (Hact)			Quantity			Expenditure (Rs Lakh)			Remarks
		Proposed*	Achievement	Cumulative	Proposed	Achievement	Cumulative	Proposed	Actual	Cumulative	
1	2	3	4	5	6	7	8	9	10	11	12
(A) RECLAMATION & REHABILITATION OF MINED OUT LAND/ AREA	i) Backfilling	Not proposed	Nil	4.05	Nil	Nil	4.81 Lakh Cum	Not specified	Nil	577	Backfilling done in earlier area identified (4.05 ha.) for the same in Western part of OB-II quarry.
	ii) Afforestation in backfilled area	Not proposed	Nil	Nil	Not proposed	Nil	Nil	Not specified	Nil	Nil	
	iii) Other (Please specify) e.g. afforestation on exhausted benches	Not proposed	Nil	Nil	Not proposed	Nil	Nil	Not specified	Nil	Nil	There are no such exhausted benches available. Part of OB-II quarry which is available can't be taken up in near future as proposals for joint mining with M/s Jindal Stainless is under review at that area along the common lease boundary.
	iv) Pisciculture	Not proposed	Nil	Nil	Not proposed	Nil	Nil	Not specified	Nil	Nil	
	v) Converting into water reservoir	Not proposed	Nil	Nil	-do-	Nil	Nil	Not specified	Nil	Nil	
	vi) Picnic spot	Not proposed	Nil	Nil	-do-	Nil	Nil	Not specified	Nil	Nil	
(B) STABILISATION & REHABILITATION OF DUMPS (within lease)	i) Terracing	Not specified	Nil	Nil	3 lifts of 25m each	Complied to proposal	Complied to proposal	Not specified	Nil	Nil	New active dumps shall be made of lifts of 10m height each.
	ii) Pitching	Not specified	Nil	Nil	Not proposed	Nil	Nil	Not specified	Nil	Nil	25000 Sqm dump area coir matted from 2008-09 till 2010-11. Expenditure included in cost of afforestation.
	iii) Construction of parapet wall/ retaining wall at toe of dumps	Not specified	Not applicable	Not applicable	Not proposed	1200m	3258m	Not specified	12.50	81.21	Damaged retaining walls were reconstructed.
	iv) Construction of check dams along the slope of vallies etc.	Not specified	Not applicable	Not applicable	Not proposed	Heightening of existing Check dam done	Heightening of existing Check dam done	Not specified	25.00	71.00	Based on need the check dam was heightened by 10m
	v) Construction of settling ponds (Garland drains etc.).	Not specified	Not applicable	Not applicable	Not specified	225 m	2283m	Not specified	3.50	52.71	Old garland drains were reconstructed.
	vi) Desiltation of settling ponds channels	Not specified	Not applicable	Not applicable	Not specified	10 Nos.	10 Nos.	Not specified	7.00	24.62	
	vii) Afforestation on dumps	Nil* 8.00 ha**	2.0 ha.* 6.50ha**	8.84 ha.* 13.86ha**	Nil* 20000 Nos**	25000 Nos.* 60000 Nos.**	138250 Nos.* 152750 Nos.**	20.00	38.35	99.32	* Within Mining Lease Area ** Within Additional area of 100ha.Species: Sirisa, Karanja, Jammun, Akacia, Mahanim, Nim, Chhatiana etc.

	viii) Others (Dump Stability Study)	Not specified	Nil	Nil	Not specified	Nil	Nil	Not specified	6.00	6.00	Dump Stability Study on progress by IIT, Kharagpur
© REHABILITATION OF BARREN AREA WITHIN LEASE	i) Afforestation (green belt building)	Not specified	Gap filling	Gap filling	Not specified	1300 Nos.* 19180 Nos.**	3300 Nos.* 61863 Nos.**	Not specified	6.40	11.41	*Within the Colony Area. ** Within Buffer Zone in villages by TSRDS
	ii) Others (Avenue Plantation)	Not specified	Gap filling	Gap filling	Not specified						
(D) ENVIRONMENTAL MONITORING (Core Zone)	i) Ambient air quality	4 Locations	4 Locations	4 Locations	Twice a week	Twice a week	Twice a week	Not specified	58.50	143.26	Air and Water quality parameters are well within the permissible limits. The quarterly report for air and water quality is being sent to Regional Controller of Mines, IBM, Bhubaneswar Region. The day wise monthly report for Cr+6 and pH for ETP Inlet and Outlet is also being sent to Regional Controller of Mines, IBM, Bhubaneswar Region on or before 2nd of every month for the preceding month.
	ii) Water quality	Inlet and outlet of 3 ETPs	Inlet and outlet of 3 ETPs	Inlet and outlet of 3 ETPs	Analysis done monthly once by third party. Daily analysis of water for Cr+6 and pH at inlet and outlet at our own Lab. Qualitative analysis on hourly basis at all ETPs.	Analysis done monthly once by third party. Daily analysis of water for Cr+6 and pH at inlet and outlet at our own Lab. Qualitative analysis on hourly basis at all ETPs.	Analysis done monthly once by third party. Daily analysis of water for Cr+6 and pH at inlet and outlet at our own Lab. Qualitative analysis on hourly basis at all ETPs.				
	iii) Noise level Survey	11 Locations	11 Locations	11 Locations	Quarterly	Quarterly	Quarterly				Noise level is well within the permissible limits
	iv) Ground vibration	Not applicable	Not applicable	Not applicable	Not specified	CMRI is doing the blast vibration study	CMRI is doing the blast vibration study				Controlled blasting techniques like presplit blasting, use of NONEL and SME (Site Mixed Emulsion) are being practiced.
	v) Others (Ground Water)	Not applicable	Not applicable	Not applicable	Not specified	2 Locations	3 Locations				Ground Water Quality is well within the permissible limits
(E) ENVIRONMENTAL MONITORING (Buffer Zone)	i) Ambient Air quality	10 Villages	10 Villages	10 Villages	Quarterly	Quarterly	Quarterly				Ambient air quality in 10 villages in Buffer zone is within the permissible limit.
	ii) Water quality	Not applicable	Not applicable	Not applicable	Not specified	Quarterly	Quarterly				Water samples taken from tube wells/ open wells from nearby village's shows no trace of Cr+6.

Note:
Proposal and Achievement is for the year 2010-11.
Cumulative figures are from 2008-09 to 2010-11 under on-going approved Scheme of Mining period

17.0 FINANCIAL ASSURANCE:

The amount of financial assurance to be deposited has been calculated based on the area proposed to be used for mining and allied activities at the rate of Rs. 25000 / hectare. The area to be utilized has been evaluated on the basis of the conceptual mining plan/ land use pattern at the end of the scheme period, which is enclosed as drawing no SCM/MP/13B/12. The change in area at present vrs. that at the end of scheme period has also been shown in the drawing no. SCM/MP/13C/12. For this, the areas put to use for mining and allied activities and the area to be considered for financial assurance has been calculated as in the table 17.1.1 given below.

Table:17.1.1
Land use Pattern & Area considered for Financial Assurance

Sl No.	Type of Land Use	Area in Hect.				
		Present Land Use (31.03.2012)	Additional Area requirement during the plan period	As at the End of Plan Period (2012-13)	The area considered as fully reclaimed and rehabilitated	Net area considered for calculation of financial Assurance
		A	B	C=A+B	D	E=C-D
1	Area to be Excavated	199.77	1.73	201.50	0.00	201.50
2	Storage of Top Soil	0.00	0.00	0.00	0.00	0.00
3	Over Burden Dump	37.47	0.00	37.47	0.00	37.47
4	Mineral Storage	37.08	-6.38	30.70	0.00	30.70
5	Infrastructure (Workshop, magazine, U/G installations etc)	3.27	4.65	7.92	0.00	7.92
6	Roads	19.05	0.00	19.05	0.00	19.05
7	Railways	0.00	0.00	0.00	0.00	0.00
8	Greenbelt	38.53	0.00	38.53	29.69	8.84
9	Tailing Pond	15.95**	0.00	15.95	0.00	15.95
10	Effluent Treatment Plant	0.32	0.00	0.32	0.00	0.32
11	Mineral Separation Plant	18.38	0.00	18.38	0.00	18.38
12	Township Area*	36.18	0.00	36.18	0.00	36.18
13	Others (to be specified)	0.00	0.00	0.00	0.00	0.00
	Total	406.00	0.00	406.00	29.69	376.31

*includes roads within the colony
**includes pyroxenite section for temporary storage of tailing

The copy of financial assurance of Rs 94, 07,750/- (Ninety four lakh seven thousand seven hundred fifty only), already submitted in form of a Bank Guarantee No. 0006BG00009013 dated 19.04.2012 and validity till 31.03.2018 drawn from ICICI Bank in favour of the Regional Controller of Mines, Bhubaneswar is enclosed as Annexure-25.

18.0 CERTIFICATE:

Certificate duly signed by the nominated owner, to the effect that said closure plan complies to all statutory rules, regulation, order made by the Central and State Govt. Where ever necessary and whenever any specific permission is required, the concerned authorities will be approached has been enclosed in this document. Certificate further indemnifies that regulations related to miner's health as per DGMS shall be implemented. Certificate to this effect from the applicant is attached Annexure-26.

19.0 PLANS, SECTIONS ETC.:

No modification is being proposed. The list of plans & sections enclosed along with this Progressive Mine Closure Plan is as given below:

1. SCM/MP/12/12 : Present Land Use Plan
2. SCM/MP/13B/12 : Financial Assurance Plan
3. SCM/MP/14/12 : Progressive Mine Closure Plan

Prepared by:

(G.K Guin)
RQP
Regd.No: RQP/BBS/044/2003/A

Dated: 9th May' 2012

MINING PLAN
(From 12th Jan'2013 onwards)

SUKINDA CHROMITE MINE
(Lease Area: 406 ha.)

INTRODUCTION

Scope of this Mining Plan:-

The Tata Steel Limited (earlier Tata Iron & Steel Co. Ltd) secured a mining lease in the year 1952 over an area of 1813.457 hectares in Sukinda Valley of Jajpur district in the state of Orissa from the then Maharaja of Sukinda Garh. The lease was ratified in 1953.

The first renewal of the lease was for 20 years with effect from 12.01.1953.The first renewal was for a period of 20 years from 12.01.1973 to 11.01.1993 over an area of 1261.657ha. The second renewal was for a period of 20 years from 12.01.1993 to 11.01.2013 over an area of 406.00 ha.

This Mining Plan is being prepared and submitted under Rule 22 & 24A of the Mineral Concession Rules, 1960 for an area of 406 ha as a part-fulfillment of lease renewal application. The proposals in this Mining Plan have been made for five years form 12th Jan' 2013 onwards.

The company has also in the meanwhile duly applied for prior Environmental Clearance to the Ministry of Environment & Forests (MoEF) for undertaking this change of technology to underground mining, increased production, increase in project area and renewal of the lease. As per the Terms of Reference for this Environmental Clearance, the MoEF has also demanded for an approved Mining Plan for underground mining.

At present this mine is one of the main sources of supply of chromite ore to meet the requirement of Ferro Alloys Plant at Bamnipal and Cuttack, conversion plants at different locations in the country as well as pyroxenite ore for the company's plant at Jamshedpur.

Further to above the mines shall continue to be the main source of chromite ore to meet the increased requirement of Ferro Alloys plants after expansion of our plant at Bamnipal, Cuttack, addition of new facility at Gopalpur and Nayagarh & for beneficiation plant with in the leasehold area to upgrade the low grade chromite ore. The company is also looking forward to increase the value addition of chromite ore through ferro chrome making by having more no of conversion agreements. With the new steel making facility coming up at Kalinga Nagar, Chattisgarh and brown field expansion of existing plant at Jamshedpur, the requirement of pyroxenite is also going to be twice of present rate.

1. GENERAL

a) Name of the Applicant with address :

The applicant and the present nominated owner of the mine is Mr. H. M Nerurkar. The copy of photo ID of nominated owner along with passport details towards address proof is enclosed as Annexure-14. The details of address are as follows.

Mr. H. M Nerurkar
Managing Director
Tata Steel Ltd.
At. / P.O. : Jamshedpur
Dist : East Singhbhum
Jharkhand : 831001

Phone : 0657- 2423298
Fax : 0657- 2431818
Email : mdoffice@tatasteel.com

b) Status of the applicant :

The Tata Steel Limited is a public limited company having its Steel Works at Jamshedpur in the state of Jharkhand. The company also operates a Ferro Alloys Plant at Joda in Keonjhar district of Orissa state. Besides this, the company operates number of metal and coal mines in many parts of the country to cater its works requirements in respect of various raw materials. The applicant is the Nominated Owner of the company. The present members of the Board of Directors of TATA STEEL are:

Name	Pan number	Voter ID number	Citizenship	Passport number / DL number
Ratan N. Tata	AAAPT0002F	MT 04 019 060249	Indian	Z 2177343
B. Muthuraman	AAHPB7489R	JVN 1545873	Indian	H 5574492
Nusli N. Wadia	AAAPW0990M	Not allotted	Indian	F 6352000
S.M. Palia	AABPP2138D	Not allotted	Indian	A 6129072
Suresh Krishna	AABPK3154E	Not allotted	Indian	Z 1759392
Ishaat Hussain	AAVPH 6348F	MT 04 019 057708	Indian	Z 2026139
Jamshed J. Irani	AAFPI1888M	BR 50292 333139	Indian	H 6855092
Subodh Bhargava	AAIPB9290R	MT 08 038 363390	Indian	Z 2007543
Jacobus Schraven	----	Not Applicable	Dutch	NT07K92F5
Andrew M. Robb	----	Not Applicable	British	800240979
Karl-Ulrich Kohler	----	Not Applicable	German	507726111
H.M. Nerurkar	ABGPN0776M	JVN 4233342	Indian	G 9227924

The resolution of Board of Directors for nominated owner, Memorandum of Association & Article of Association issued under Companies Act etc. is enclosed as Annexure-15 & Annexure-12 respectively.

c) Mineral(s) which are occurring in the area and which the applicant intends to mine :

Chromite & Pyroxenite Ore

d) Period for which the mining lease is granted/ renewed /proposed to be applied :

The company prospected the area under a PL dated 14.4.1952 and subsequently got a ML from the then Raja of Sukinda for a period of 20 years with effect from 22.10.1952. With the enactment of Orissa Estate Abolition Act (Act 1 of 1952), the area got vested with the State Govt. and the same Mining Lease was ratified by the State Govt. for a period of 20 years with effect from 12.1.1953.

Due to remoteness and inaccessibility of the place, non-availability of skilled manpower in the area and lack of market for the friable variety of chrome ore, the mining originally started in a small scale to supply refractory grade lumpy ore to our refractory plant at Jamshedpur & Belpahar during December 1960.

The 2nd renewal over a reduced area of 406 ha. has been granted by Central Government vide letter no. 5(22)/95-MIV, dated 17th Aug'1995 (Annexure-1) for a period of 20 years with effect from 12.01.93. The steel company has also submitted the renewal application for the 3rd renewal of the lease over an area of 406 ha. for the period of 20 years from 12.01.2013 to 11.01.2033. The mining lease application has been marked with State Serial No. 364, dated 04.01.2012. Copy of Form-D towards receipt of Mining Lease renewal application issued by Dy. Director Mines, Jajpur Road Circle, Jajpur Road vide Memo No. 48, dated 04.01.2012 is enclosed as Annexure-27. This Mining Plan is being prepared and submitted under Rule 22 & 24A of the Mineral Concession Rules, 1960 for mining lease area of 406 ha as a part-fulfillment of lease renewal application.

e) Name and address of the RQP preparing the Mining Plan :

Mining Plan for Sukinda Chromite Mine is prepared by G K Guin, RQP & Head (Mine and Production Planning) for the Tata Steel Ltd. The certificate of RQP bears the Registration no. RQP/BBS /044 /2003 /A and is valid till 03.02.2013. The copy of RQP certificate is enclosed as Annexure-16.

The address of the RQP is given below:

Mr. G K Guin
Head (Mine and Production Planning)
Tata Steel Ltd., Sukinda

Dist	: Jajpur
State	: Orissa
Pin Code	: 755028
Phone	: 09238100835
Fax	: 06726268734

The RQP has been assisted by the officers and staffs of Planning Cell. Sri Haridrumat Behera RQP and Head (Underground) has assisted in chapters for the underground mining part of the document. His certificate bears the number RQP/BBS/093/2010-A and valid up to 19.04.2020 (Annexure-17). Services from the departments of Geological Services, Land & Lease, Safety and Environment and the field personnel were also availed in relevant areas of their expertise.

The RQP, Sri G. K. Guin has been duly authorized by the applicant. The letter of consent is enclosed as Annexure-24.

f) Name and address of the prospecting agency :

The company's own Natural Resources Division (earlier Geological Department) has prospected the lease area of Sukinda Chromite Mine on various occasions during the lease-hold period. The prospecting of lease hold was last conducted by the department during the investigation period 2008-11.

g) Reference Number And Date Of Consent Letter From The State Govt:

Since the tenure of the lease period of twenty (20) years expires on 11.01.2013, this mining plan is prepared statutorily under Rule 22 & 24A of MCR, 1960 (amendment 1987) in respect of renewal of lease for a period of twenty (20) years up 11.01.2033. The company has also submitted the renewal application for the 3rd renewal of the lease over an area of 406 ha. for the period of 20 years from 12.01.2013 to 11.01.2033. The mining lease application has been marked with State Serial No. 364, dated 04.01.2012. Copy of Form-D towards receipt of Mining Lease renewal application issued by Dy. Director Mines, Jajpur Road Circle, Jajpur Road vide Memo No. 48, dated 04.01.2012 is enclosed as Annexure-27.

2. LOCATION AND ACCESIBILITY:

i. Details of the Area :

Sukinda Chromite Mine lies in the district of Jajpur, Orissa and is connected with Jajpur Keonjhar Road Railway Station on the Howrah-Chennai trunk line of East Coast Railways by 52 KM all weather road. J K Road Railway Station is 336 KM away from Tatanagar Railway Station via Kharagpur and is 337 KM from Howrah, 100 KM from Bhubaneswar-state capital and 156 KM from Paradip, the nearest major port on the Eastern Coast.

The all-weather road connecting Sukinda Chromite Mine with JK Road Railway station meets NH-200 connecting JK Road with Bhuban at Mangalpur at a distance of 21 KM and Express Highway no 1 connecting Daitari with Paradip at Duburi at a distance of 32 KM from the mine.

Sukinda Chromite Mine comprises of 406 hectares of land in villages Kalarangiatta, Kaliapani, Mahulkhal & Forest Block No. 27 in the Revenue district of Jajpur, Sub-division Jajpur and Thana Kaliapani.

Village : Kalarangiatta, Kusumundia, Kaliapani etc
Tehsil : Sukinda
P.S. : Kaliapani
District : Jajpur
State : Orissa
Lease Area : 406 ha.

The Mining Lease over an area of 406ha is valid till 11.01.2013. Co-terminus with the mining lease we have been allotted 100ha of additional area for overburden dumping contiguous to southern lease boundary. This 100 ha was essentially forest land for which forest clearance has been obtained. We have further applied for 73.685ha of area south of the additional area of 100ha for tailing disposal. The above area consists of 8.370ha of forest land which shall act as part of the corridor to reach the area of tailing disposal spread over an area of 65.315ha which is a non-forest Govt. land.

Type of Lease area:

The land classification of Mining leasehold area over 406ha, additional area of 100ha allotted for overburden dumping and applied area of 73.685ha for tailing storage is given below in Table 2.1.1.

Table: 2.1.1

Area Type	Total Area (Ha.)	Forest Block No-27 Area (Ha)	Khesra Forest (Ha.)	DLC Area (Ha.)	Total Forest Area (Ha.)	Non-Forest Area (Ha.)	Remarks
ML Area	406.000	73.612	0.085	0.000	73.697	332.303	Mining Lease Area
Additional area	100.000	95.450	4.550	0.000	100.000	0.000	Allotted for dumping of Overburden
Applied Area	73.685	1.940	6.430	0.000	8.370	65.315	Applied for disposal of Tailing

The detail of land (schedule) is enclosed as Annexure-28.

ii. Toposheet No. With Latitude & Longitude And Key Plan:

The Key Plan of the lease area of 406 ha, the allotted area of 100ha contiguous with the southern lease boundary of the existing lease and the applied area of 73.685ha is shown in Drawing No. SCM/MP/01/12. The area falls under the Survey of India Topo Sheet no 73-G/12, 73-G/16, 73-H/9, 73-H/13. The approved Lease Plan is enclosed as drawing no. SCM/ MP/01A/12. The Surface Plan of the leasehold area is shown in drawing no. SCM/MP/02/12 updated as on 01.04.2012. The extent of Latitude and Longitude of the above area is mention below in Table. 2.2.1

Table: 2.2.1

Details of Latitude & Longitude

Sl No.	Area Type	Area (ha)	Latitude From	Latitude To	Longitude From	Longitude To	Remarks
1	ML Area	406.00	$21^0$00’39.60”	$21^0$02’5.81”	$85^0$44’27.10”	$85^0$46’22.37”	Mining Lease Area
2	Additional Area	100.00	$21^0$00’27.41”	$21^0$01’1.68”	$85^0$45’22.09”	$85^0$46’51.42”	Allotted for dumping of Overburden
3	Applied Area	73.685	$20^0$59’34.88”	$21^0$00’51.00”	$85^0$46’46.84”	$85^0$47’32.69”	Applied for disposal of Tailing

PART- A

3. GEOLOGY & EXPLORATION :

a) Topography, general/ regional geology & local/ mine geology:

(i) Topography:

Sukinda valley, best known for hosting the treasure of chrome wealth in the country is located in the eastern state of Orissa, India. This valley is bounded by Tomka-Daitari Range in North and Mahagiri Range in South having a general slope of 18-20° towards South-West with isolated mountains & ridges. The area has a tropical to sub-tropical climate with rainfall during monsoon months of June to September. The annual rainfall in the area is around 90 – 194 cm and the temperature varies between 8-9° C in winter to 46-47° in summer.

(ii) Regional Geology:

The Sukinda ultramafic complex, bounded by latitudes 20°53’ and 21°05’ and longitudes 85°40’ and 85°53’, forms a part of the metamorphosed Pre-Cambrians of the Peninsular India consisting of sporadically occurring dismembered chromiferous ultramafic bodies (Banerjee, 1972). Chromite deposits are mainly associated with ultramafic rocks and based on genetic occurrences, categorized into two types – stratiform and podiform. Majority of chrome deposits, located in Sukinda chromite valley, mainly occur as bands, lenses and pockets in the serpentinized dunite-peridotite, partially or fully limonitized/lateritized at the central part of the valley. They are well differentiated layered igneous complexes and belong to the well known stratiform complexes of the world (Mohanty, 2006). Mineralization of grey ores normally vary from massive in the central part to banded, spotted and laminated in the contact zones with the ultra basic rock. The brown ores have been formed due to alteration of chromite ores during lateritization / limonitization which also caused obliteration of all primary structures. Chromite ore also shows distinct variation in texture and chemical composition. Chromite bands are offset by a number of dykes, faults and shear zones.

The ultramafic intrusive, trending NE-SW, consisting of Dunite, Peridotite, Orthopyroxenite and chromite seams, has intruded low grade metamorphic rocks of the Archean Iron Ore Group (IOG) with faulted contact at both the margins, in the north with Banded Iron Formation of the IOG of Tomka-Daitari Ranges and in the south with Quartzite of Mahagiri Ranges. The intrusion has been subjected to a variable degree of alteration to serpentine-talc-chlorite-magnetite±magnesite±sulfide with cumulate igneous texture commonly retained. In extreme cases, particularly in the central part of the valley, extensive lateralization/limonitization process caused formation of saprolite, limonite and in-situ as well as transported laterites and thereby causing obliteration of all primary structures.

The chromite bearing ultramafics of Sukinda area have intruded into the Precambrian metamorphites in the form of lopolith, covering 2 to 5 km width and extending from Kansa in the east to Maruabil and beyond in the west in ENE-WNW direction, at the junction of quartzite and enstatite-peridotite rocks in 40-45 km long, shear zone. The ultramafic body consists essentially of dunite-peridotite within IOG rocks with chromite bands and subordinate amount of pyroxenite devoid of chromite mineralization. They are stratiform type deposits where bands and layers are indicative of gravitational settling.

Prevalent rock types in the area include Quartzite, Serpentinized Dunite-Peridotite, Pyroxenite, Dolerite and Laterite/Limonite. The Quartzite consists of quartz grains of almost quardrant to tabular habit. The dunite-peridotite suite is highly serpentinized and has given rise to serpentinite, talc-serpentinite having a grey to grayish green colour. Pyroxenite, relatively fresh and less altered, is greenish grey in colour and composed of coarse grained orthopyroxene (enstatite). Dolerite is dark grey (fresh) to green (weathered) in colour and consists of small to medium grained plagioclase laths and pyroxene.

The regional stratigraphy of Sukinda-Nuasahi belt (Mondal, 2009) is given below:

		Generalised Sequence	Zircon Ages	Other Methods
	Kolhan Group	**KG** (Time Equivalent to SMB)		2100-2200 Ma[3]
-------------	Unconformity------------------	----------------	-----------------	-----------------
Singhbhum Mobile Belt (SMB)	**Dolerite Dyke Swarms**	**NDS**		950-2500 Ma[3, 6]
	Dhanjori-Simlipal-Dalma-Jagannathpur-Malangtoli and Singhbhum Group: Igneous and Sedimentary Sequences	**Late Archaean to Proterozoic Mobile Belts (SMB)**		2072 Ma[5] (~3.09-2.25 Ga)[7]
-------------	Unconformity----------------	-----------------	------------------	-----------------
Archaean Granite-Greenstone Terrain (AGGT) (Older Metamorphic Group, Iron Ore Group, Older Metamorphic Tonalite Gneiss, Singhbhum Granite, Bonai Granite, Nilgiri Granite, Mayurbhanj Granite	**Mayurbhanj Granite**	**SBG-B**	3.1 Ga[1]	
	Singhbhum Granite Type B	**SBG-B**		3.1 Ga[3]
	Iron Ore Group Igneous and Sedimentary Sequences: IOG Igneous Suites (Ultramafic-mafic-plutonic suite e.g., Nuasahi-Sukinda-Jojohatu (NSJ) ultramafic suite; Nuasahi-Nilgiri-Gorumahishani-Badampahar (NNGB) gabbro-anorthosite-diorite-mafic suite; Ultramafic-mafic suite e.g., komatiites and high-Mg basalts in Gorumahishani-Badampahar, Tomka-Daitari and Jamda-Koira belts; Felsic volcanics); IOG Sedimentary Sequences	**Iron Ore Group (IOG)**	3506.8 $\pm$ 2.3 Ma[9] age of zircon from dacitic lava within the Iron OreGroup; Tomka-Daitari Basin 3121 $\pm$ 3 Ma[4] ageof zircon from gabbroic suite, Nuasahi breccia zone 3285 $\pm$ 7 Ma[8] age of zirconfrom the pegmatitic biotite granodiorite overlain by IOG conglomerate	3205 $\pm$ 280 Ma[4] Sm-Nd isochron age of gabbroic suite from the Nuasahi massif
	Singhbhum Granite Type A	**SBG-A**	3328 $\pm$ 7 Ma[1]	3.3 Ga[3]
	Older Metamorphic Tonalite Gneiss	**OMTG**	Age clustering at 3.4 and 3.2 Ga[1]	3288 + 35 Ma[2] Sm-Nd isochron age with OMG
	Older Metamorphic Group	**OMG**	Age clustering at 3.55, 3.4 and 3.2 Ga[1]	3305 + 60 Ma[2] Sm-Nd isochron age

[1]Mishra et al. (1999); [7]Mishra and Johnson (2005); [2]Sharma et al. (1994); [3]Saha et al. (1988); [4]Auge et al. (2003); [5]Roy et al. (2002); [6]Roy et al. (2005); [8]Nelson et al. (2008); [9]Mukhopadhyay et al. (2008)

(iii) Local Geology:

Three chrome ore bands exist within the lease hold area, which are of varying thickness having NE-SW trend. These ore bodies are sub-vertical in nature. In general, Northern and Middle bands are brown, ferruginous & physically friable in nature whereas the Southern band is grey, hard lumpy & siliceous in nature. The updated Geological Plan in a scale of 1 : 4000 of the leasehold area is enclosed as drawing no. SCM/MP/03/12. Part Geological Plans in a scale of 1 : 2000 has also been enclosed for greater clarity.

Prevalent rock types in the lease area include Quartzite, Serpentinized Dunite-Peridotite, Pyroxenite, Dolerite, Laterite & Limonite.

The Quartzite consists of quartz grains of almost quardrant to tabular habit. The dunite-peridotite suite is highly serpentinized and has given rise to serpentinite, talc-serpentinite having a grey to greyish green colour. Pyroxenite, relatively fresh and less altered, is greenish grey in colour and composed of coarse grained orthopyroxene (enstatite). Dolerite is dark grey (fresh) to green (weathered) in colour and consists of small to medium grained plagioclase laths and pyroxene. All these rocks have been partly/fully weathered to form Limonite with secondary silica enrichment in the form of chert along fractures and pockets. The various rock types that were identified are as follows:

HUB : Hard Ultrabasics, category does not recognize metamorphic influences, unweathered grey rock
SUB : Soft Ultrabasics, weathered HUB, rock
LIM : Limonite, Completely weathered SUB, soil-like
CHT : Chert, locally overprinted on limonite to varying degrees, soil/ soft rock
DYKE : Dolerite
CRM : Chromite ore, does not distinguish weathering state

The material can be distributed into three broad groups as follows:

STRONG	:	Chrome (lumpy), Dyke ('hard') and HUB
INTERMEDIATE	:	Chrome (mixed), SUB and Dyke ('soft')
WEAK	:	Chrome (friable) and the limonite-chert (soil)

The country rock in the vicinity of the ore bodies at greater depths considered to be taken up for underground mining consists mainly of HUB. In the North and Middle ore bands the material near surface, and in close proximity to the ore at depth, has been preferentially weathered. A photograph to illustrate the progressive weathering from the deep seated HUB to SUB to LIM soil nearer the surface is given below in Fig. 3.1.

The HUB is partly or fully altered to serpentinite or talcose schist. The HUB in the vicinity of the North and Middle Ore Bands is generally much less altered than the HUB in the South Band. This is probably related to the presence of the shear zones in the north and south walls of the South Band. A photograph to illustrate the alteration of the HUB is given below in Fig. 3.2.

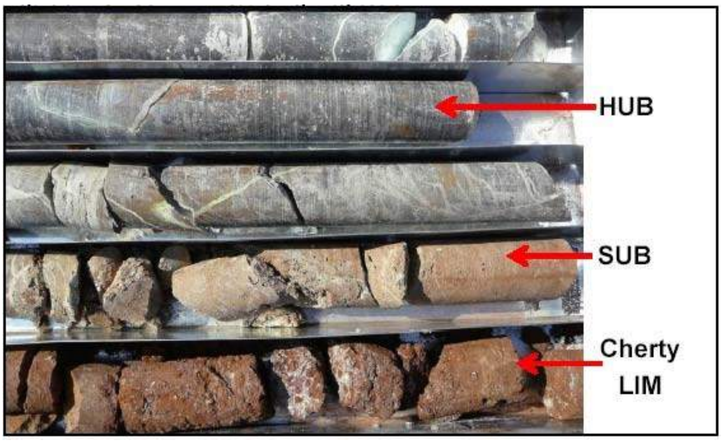


Fig. 3.1: HUB-SUB-LIM Weathering Progression

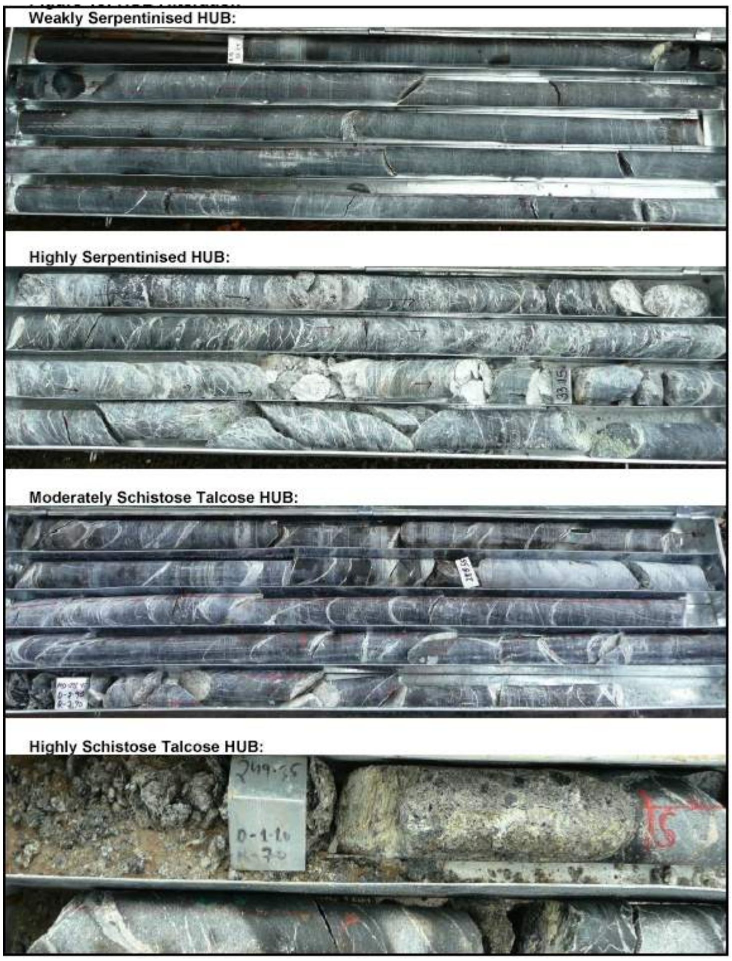


Figure 3.2: HUB Alteration

Dykes:

Dolerite dyke intrusions crosscut the three ore bands and offset them. These dykes vary in character from strong, weathered and structureless, to weak, highly weathered and structured. The dyke widths vary between 15 - 75m and this area is barren. The compartments defined by the dykes vary between 250 - 600m in strike length. These dykes offset the chromite bands in a step-wise fashion on to the south by generally 20 - 30m, upto 130m in the eastern end of the Middle Band. A photograph to illustrate the different dolerite types encountered in the area is given below in Fig.3.3.

Chromite Ore types:

The chromite bands within the leasehold show the characteristics of typical stratiform nature. Pinch and swell structures are also common. Bands are dipping at an angle vertical to sub vertical. The description of these bands is given as follows:

Northern friable ore band:

Northern friable chrome ore band is the longest friable band in the lease area. Its strike length from west to east is about 2km. This band strikes in ENE-WSW direction with steep dip of 85-90 degrees towards N-NNW. It shows swelling and pinching nature along both the strike and dip direction. It has been dissected by dolerite dykes. The width of ore body varies from 4m to 20m. Chromite of this band, presently being mined by open cast method is generally brownish in colour, fine to coarse grained, and friable/powdery in nature. Chromite consists of 38-58% Cr_2O_3, 9.5-24.5% Fe and 0.88-9.08% SiO2 as principal constituents. Gangue minerals are goethite, limonite (ferrous oxide) weathered serpentinous and talcosic materials, cherts/ silica (in form of veins) etc.

Middle friable ore band:

Middle friable ore band is situated between northern & orthopyroxenite band, parallel to the former. It is running almost in the same direction as northern band, keeping a lateral distance of 200-250m from the same. This band contains chrome ore of high iron content. It is friable, hard sometimes chilled and brownish to reddish brown in colour. The grade varies mainly between medium to inferior grade. Chromite consists of 25-44% Cr_2O_3, 9-28.5%Fe, 1.2-13% SiO2 as principal constituents. It is observed that eastern part consists of inferior grade chrome ore. Swelling and pinching is quite common with the ore body width varying between 3m to 8m.

Southern Lumpy ore band:

Southern Lumpy ore band is located at the northern foot hill of the Mahagiri hill range. The ore in this band is hard, lumpy, siliceous and grey colored in its physical appearance. It has a strike length of 0.85km and width of ore body varies from 0.75m to 8m. Chromite consists of 23.42-47.94% Cr_2O_3, 10.29-11% Fe, & 8.88-24.61% SiO2 as principal constituents. OB II quarry is part of this band. The ore body is almost vertical dipping. This band produces mostly, lumpy ore having relatively higher silica content.

Physical Characteristics of the Northern & Middle chrome ore band:

With the exploratory drilling to greater depths, in the Northern & Middle chrome ore bands, it is observed that there is a distinct change in the physical characteristics of the chrome ore with increasing depth.

The chromite ore in the Northern & Middle chrome ore bands can be categorized into three groups, based on weathering state. The materials are described further as “friable”, “mixed” or “lumpy”, and represent progressively less weathered states.

The FRIABLE material is a very soft rock (upto 5 MPa intact rock strength) or soil material. It occurs as reddish or black, silty, slightly clayey sand, with minor amounts of cohesion. Such friable ore disintegrates with a single, light hammer blow. The upper chrome orebody in the Northern & Middle Bands are friable and the distribution with depth is associated with the distinctive weathering cone that extends down to ~ 150 - 300m below ground surface. This material may act as a conduit for ground and surface water infiltration. Thus, developing any type of underground mine excavation within this material will be extremely hazardous and should be avoided, if at all possible.

The MIXED material is the transitional horizon between the soft-friable and hard lumpy ore. Core intersections exhibit both characteristics in alternating bands. Underground mining within the mixed should be possible, provided that the support is installed within the mining cycle.

The LUMPY is slightly weathered to unweathered rock. The lumpy is penetrated by faults and dykes, and some talcose alterations are occasionally present.

A photograph to illustrate the three types of chromite ore as mentioned above is given below in Fig. 3.4.

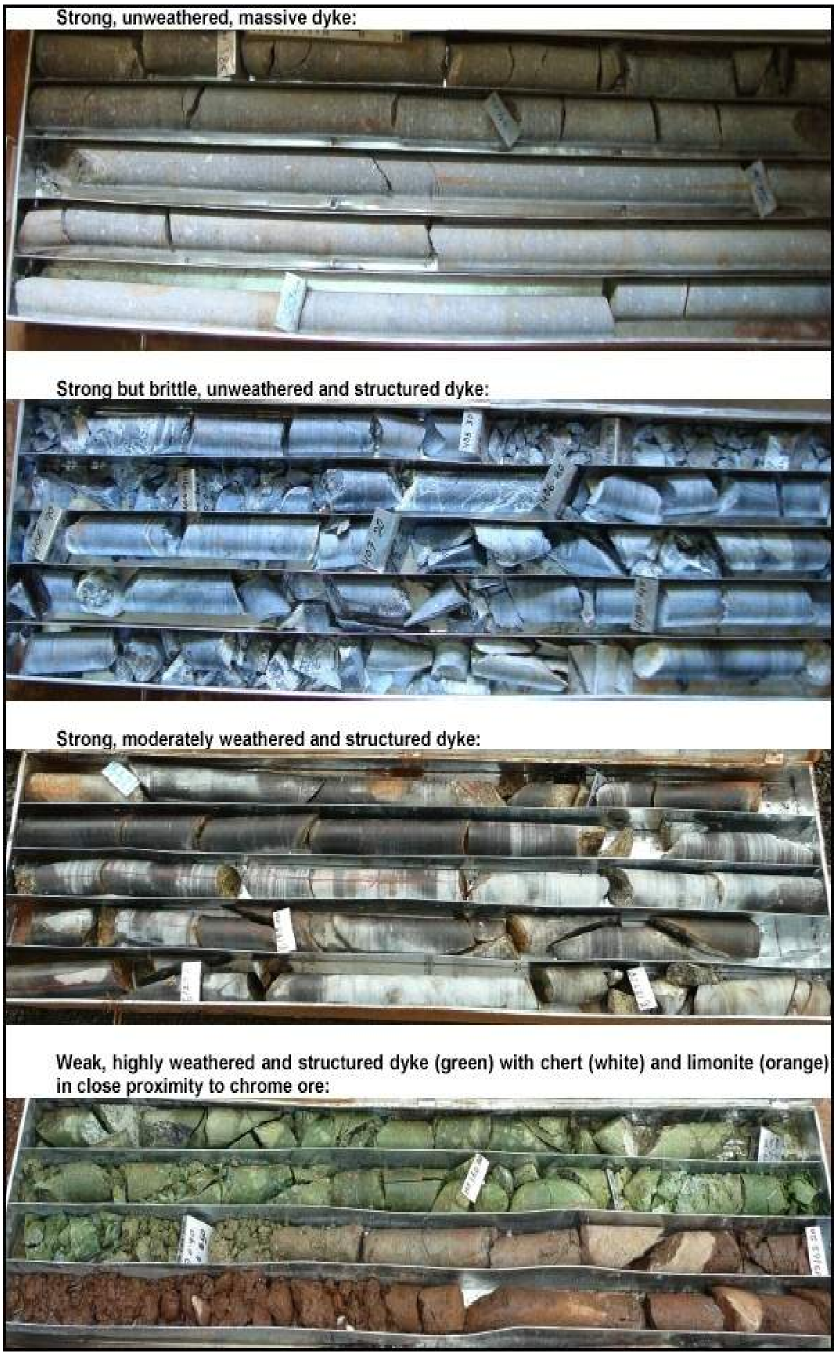


Figure 3.3: Dolerite Dyke Material Condition

Fig. 3.4: Chromite ore types of Northern & Middle band

Control of mineralization:

The Sukinda Ultramafic body is layered complex composed of alternate bands of chromite, dunite, peridotite & other orthopyroxenite repeated in a rhythmic manner. The chromites are associated with the peridotites of the rhythmic sequence. The control of chromite mineralization is 'Lithologic'.

The occurrence of Nickel & Cobalt is due to lateritization of ultramafic rocks. In course of lateritization, the NI-Co bearing ultramafics have been changed to the following formations within the leasehold:

Ni-Co poor weathered and partly lateritised ultramafics
Ni-Co rich laterite
Top oxidized lateritic – morrum layer

The Ni-Co mineralization, therefore, is topographically controlled. The wall rocks of the Northern & Middle bands are totally lateritized to Ni-Co bearing laterites due to extensive weathering. The present grade of Nickel in the overburden laterite/limonite, varies between 0.39 to 0.49% Ni. Away from the orebody, this Ni-Co bearing laterite grades to:

Ni-Co poor weathered horizon; and finally to
Hard weathered basement

In contrast, the wall rocks of the southern band are weathered but not lateritized. The contact of wall rock with the southern Mahagiri Quartzite is a thrust, which has resulted in a number of landslides all along the foothills of Mahagiri.

b) Details of Exploration:

Previous Exploration:

Apart from the fact that the Company brought the occurrence of chrome ore in Sukinda valley to light in 1949, it was also first to carry out exploration in the area. The details of exploration undertaken in the erst-while leasehold of 1261.476 hectares and category-wise reserves established in million tonnes are as given below in Table 3.1.1.

Table 3.1.1

Phase	Meters	Proved	Probable	Possible	Total	As on
1952-55	1027	0.43	1	-	1.43	31.03.55
1964-67	7200	3.21	0.16	-	3.37	31.03.67
1970-84	7845	12.9	-	12.5	25.4	31.03.84
1985-89	12136	14.7	3.15	15.05	32.9	31.03.89
1989-95	18384	11.76	2.17	15.36	29.39	31.03.95

A total of 829 boreholes were drilled till 31.03.95. The year wise borehole meterage drilled during the period is given in the following Table 3.1.2.

Table 3.1.2

Year	No of Bore Hole	Meter Drilled
Upto 1989	611	28,208
1989-90	49	2,876
1990-91	61	4,826
1991-92	54	5,001
1992-93	19	1,684
1993-94	18	2,028
1994-95	17	1,969
Total	829	46,592

With the reduction in the area of the leasehold to 406 hectares, the data pertaining to this reduced area is as given in Table 3.1.3.

Table 3.1.3

Year	No of Bore Hole	Meter Drilled
Upto 1989	270	14,589
1990	25	1,889
1991	36	3,778
1992	12	1,056
1993	6	698
1994	15	1,978
1995	21	2,124
1996	29	3,898
Total	414	30,010

In course of chromite mining, occurrence of Nickel & associated Cobalt was discovered by the Company and was reported to the Govt. as provided under Rule 27(1)(a) of the Mineral Concession Rules, 1960. In pursuance of an application made by the Company, the State Govt. was pleased to grant P.L., detailed geological investigations were conducted by the Company and a Probable Reserve of 19.75 million tonnes of Nickel bearing mineral was established. For this, 60 boreholes covering 5884 meters were drilled.

The Company also has investigated the erstwhile leasehold area for occurrence of Pyroxenite & established a reserve potential of 103 million MT. Incidentally this has been included as a second mineral in the Company's the then lease, with effect from 05.12.1991.

After the lease area reduction to existing 406 ha the major part of the core drilling done during the period 1998-03 was essentially for establishing the geological structure of the host rock, hydro-geological investigations, rock strength parameters etc. for the study on open pit optimization. The following Table 3.1.4 gives an idea of the exploration done. As against the proposal of drilling 6250m during the period 1998-2003, the actual meterage drilled was 6877.66m.

Table 3.1.4

Year	No of Bore Hole	Meter Drilled
1998-99	4	692.95
1999-00	3	282.19
2000-01	36	3,274.11
2001-02	9	2,502.88
2002-03	1	125.53
TOTAL	53	6877.66

Some of the exploratory drilling undertaken 2002-08 was for the purpose of gap filling. The amount of exploration done during 2003-08 is as given in Table 3.1.5.

Table 3.1.5

Year	No of Bore Hole	Meter Drilled
2003-04	9	664
2004-05	3	1317
2005-06	7	1216
2006-07	7	1299
2007-08	2	446
TOTAL	28	4942

Present Exploration:

As the reserves of chrome ore was already established by detailed proving up to a depth of 150 m, considering the soft strata conditions encountered in the friable ore bodies, it was felt necessary to prove the northern friable ore body beyond 150 m depth. However, considering the restrictions imposed by the lease boundaries for lateral development of the opencast operations, boreholes have been proposed for 250m depth to prove the presence of ore and strata condition at greater depth for possible underground mining in future.

Table 3.1.6 below gives the exploration figures as proposed in the approved scheme of mining and the actual exploration done. As against the proposal of drilling 6116 m during the period 2008-12, the actual meterage drilled during the period 2008-12 was 22758 m.

Table 3.1.6

Year	Proposed Bore-holes (No.'s)		Actual Bore-holes drilled	
	Total No	Meterage	Total No	Meterage
2008-09	12	2222.40	12	2222.40
2009-10	13	3893.20	13	3866.60
2010-11	0	0	34	10513.25
2011-12	0	0	18	6155.60
TOTAL	25	6115.60	77	22757.85

Additional drilling of 34 boreholes (7 Exploratory, 9 Geotechnical, 15 Hydro geological & 3 In-situ Stress Measurement boreholes) comprising 10513 m total meterage was done in 2010-11 to serve as further inputs for feasibility of Underground Mining and underground resource estimation. Apart from that, further 18 exploration boreholes, 6155 m were done during the year 2011-12 for upgradation of resource from inferred to indicated category

The logging details of these boreholes have been given as Annexure-8.

c) Geological Sections:

The updated Geological Plan & Geological Section of the leasehold area showing the present boreholes is enclosed as drawing no. SCM/MP/03/12, SCM/MP/04A/12, SCM/MP/04B/12, & SCM/MP/04C/12.

d) Future Exploration Programme:

The morphology of the Chromite ore body as existent in Sukinda valley has been thoroughly studied during the long exploration history of Sukinda. Sukinda Chromite deposit is a stratiform, stratabound and Tabular deposit of regular habit. This observation has also been made by GSI in their Detailed Information Dossier on Chromite Ore in India, January, 2008 (Chapter2, pg-9) where they have clearly mentioned that **"The Chromite deposit of Sukinda belt is mainly stratiform type"**. Chromite body exists in a continuous stretch intermittently intercepted by dykes. Within the leasehold area, these intercepting dykes (four in numbers) occur naturally and cut across the orebody at 4 places in the Friable bands and penetration through them has not disturbed the neighbouring ore morphology nor it is expected that sudden dykes will appear throughout the stretch which is evident from the present exploration. The nature of the dyke in the friable ore bands is well established. There is no apprehension of sudden occurrences of dyke as seen during exploration. The drilling results also support the correlation.

The entire leasehold area has been well explored although more emphasis has been given for detailed exploration for three established chromite bands. Orebody being tabular / vertical detailed exploration at depth has been done. To rule out occurrences of other orebodies, exploration has been done in the surrounding quarry areas laterally. This apart, Hydrogeological, Geotechnical and Infrastructure holes have been drilled. We have not detected presence of any other mineral / orebody through these holes made. Geophysical study on “Locating Probable Groundwater zones around Lease hold area of Tata Steel” conducted by Faculties from IIT, Kharagpur does not indicate presence of any other orebody except the three being currently discussed. GSI has also found and taken only these three bands into consideration while conducting their study in this belt / lease-hold area from time to time through Geological mapping and other spectral data.

As may be observed from the updated surface geological plan, all the ore bands have been explored by drilling mostly inclined boreholes to cover the total strike length from extremities in the west to the east of the lease hold. As is evident from the updated longitudinal geological section, the depth of proving for the opencast mining upto the ultimate pits limits has been more than adequate with a closely spaced grid interval of 50 – 100m. Most of the mineable reserves for opencast workings are in the proved category in all the three ore bodies. In view of the dwindling resources of chrome ore for opencast mining, the company has since long taken up further core drilling program to assess the resources at greater depths. Feasibility studies for underground mining have been completed based on the resources so established.

So far till 01.04.2012 the deepest intersections in the northern, middle and southern ore bands have been (-) 375, (-) 235 & (-) 175 mRL respectively. The average depth upto which the resources have already been explored and brought to the indicated (G2) category has been (-) 150, (-) 190 & (-) 125 mRL respectively.

The future exploration has been conceptualized to further prove the depth persistency of all the three orebodies and progressively bring the inferred or (G3) resources to the (G2) category by way of drilling more number of boreholes from the surface. The remaining gaps in exploration of the orebodies have been identified. Accordingly the future proposals of drilling deeper boreholes from the surface to fill these gaps and also further explore the depth persistency during the next 5 years have been proposed, to be in consonance with MOM directives for conditions under 27(3) of MCR, 1960. Considering the near vertical nature of the orebodies, drilling to still greater depths from the surface would require the drilling to be done at fairly steep angles due to lack of space laterally. Moreover, there is also a likelihood of missing the orebody while attempting to drill very long holes at steep angles. Accordingly, the maximum depths of such surface boreholes have been conceptualized. The deepest intersections proposed for the northern, middle and southern ore bands are (-) 450, (-) 450 & (-) 400 mRL respectively. Further, in order to bring the indicated (G2) category of resources at the deeper horizons to the measured (G1) category of resources, the exploratory drilling at lateral intervals of 50-100m and 30-50m vertical depth has be proposed to be undertaken from the proposed underground haulage drives/ drifts. The targeted intersection points for the proposed boreholes have been indicated in the longitudinal geological section for all three orebodies.

Besides the above, boreholes for infrastructure development of the major underground infrastructures like declines and vertical shafts have also been planned to be done at the earliest.

The details of the proposed drilling for the next 5 years of the Mining Plan period have been given in Table 3.1.7. The location of the proposed boreholes has been indicated in drawing No. SCM/ MP/ 03/ 12. The details of proposed surface exploratory boreholes mentioned above are given in Table 3.1.7 and those for underground boreholes to be drilled are given in Table 3.1.8. The expected length and direction of all these boreholes have been planned by plotting the same in the transverse geological sections. The yearwise and purpose wise distribution of all these boreholes have been shown in different colour in the Geological Plans & Sections enclosed. The planned points of intersection of these boreholes with the orebody have been shown in the Longitudinal Vertical Section for all three orebands.

Table 3.1.7
Yearwise Exploration Proposal for Surface Exploratory Boreholes

Year	Type	Ore Body	BH No	Northing	Easting	RL	Planned Level Of Intersection, mRL	Dip	Azimuth	Appx. Length, M
2013-14	Exploration	Northern Band	PBH33	2717	2990	124	-300	68	180	500
			PBH34	2333	2000	119	-350	59	0	575
			PBH35	2800	2100	114	-350	61	180	575
			PBH36	2709	3700	134	-350	66	180	600
			PBH37	2704	3900	137	-350	73	180	575
		Middle Band	PBH38	2464	3900	72	-300	60	180	475
			PBH39	2407	3494	63	-300	66	180	450
			PBH40	2443	3110	48	-300	67	180	450
			PBH41	2408	2900	61	-300	73	180	500
			PBH42	2441	3300	53	-350	58	180	525
		Southern Band	PBH43	1464	3250	121	-200	49	180	450
			PBH44	1477	3300	121.6	-200	73	180	475
			PBH45	1350	3450	56.11	-225	57	180	400
			PBH46	1540	3500	131.5	-200	55	180	525
			PBH47	1517	3700	130.4	-200	59	180	400
			PBH48	1513	4006	147.3	-200	63	180	500
			PBH49	1428	3150	121	-300	66	180	500
			PBH50	1516	3750	131	-300	65	180	600
			PBH51	1513	3897	136	-350	66	180	600
			PBH52	1534	3595	133	-300	68	180	600
	Total									**10275**
2014-15	Exploration	Northern Band	PBH53	2728	2805	123	-400	70	180	625
			PBH54	2926	1900	112	-450	61	180	700
			PBH55	2746	2600	121	-450	66	180	650
			PBH56	2716	3700	136	-450	70	180	650
			PBH57	2718	4000	136	-450	77	180	675
			PBH58	2757	3200	138	-400	72	180	800
		Middle Band	PBH59	2542	1800	70	-400	68	180	475
			PBH60	2442	2503	57	-400	66	180	550
			PBH61	2453	2700	43	-450	65	180	475
			PBH62	2411	3702	64	-450	70	180	625
			PBH63	2482	2000	63	-400	73	180	500
		Southern Band	PBH64	1534	3500	131	-400	69	180	700
			PBH65	1508	4000	147	-400	72	180	650
			PBH66	1433	3050	121	-400	75	180	700
	Total									**8775**

Table 3.1.8

Yearwise Exploration Proposal for Underground Exploratory Boreholes

Ore Body	Level of intersection	FY 16			FY 17			FY 18		
		No of BH	**Meterage**	**Easting**	**No of BH**	**Meterage**	**Easting**	**No of BH**	**Meterage**	**Remark**
SB	-35	24	1840	3350-3700	48	4166	2950-3300 & 3750-4100	-	-	-
	-65									
	-95									
MB	-55	5	452	1900 & 2000						
	-75									
	-95									
	-55	-	-	-	33	2548	1700-2200	-	-	-
	-85									
	-115									
	-145	-	-	-	-	-	-	52	4244	1700-2950
	-175									
NB	-95	3	262	1975 & 2050	27	2233	1700-2200	-	-	-
	-125									
	-155									
	Total	**32**	**2554**		**108**	**8947**		**52**	**4244**	

e) RESERVE:

i) CATEGORY-WISE RESERVES INDICATED IN THE APPROVED SCHEME OF MINING :

The mineable reserve including all categories was updated during the second review of approved Scheme of Mining and was estimated to be 9.752 million tonnes as on 31.03.08 as shown in Table no 3.5.1.

Table 3.5.1
Summary of Reserves as on 01.04.2008
(in Million Tonnes)

Orebody	Grade ($\%Cr_2O_3$)	Reserve (31.03.2008)		Total
		111	122	111+122
Northern Band	+ 10 - 40	0.052	0.799	
	+ 40 - 52	0.003	2.033	
	>52	2.667	0.342	
	Total	2.722	3.174	5.896
Mid Band	+ 10 - 40	0.150		
	+ 40 - 52	0.006	2.936	
	>52	0.004		
	Total	0.160	2.936	3.096
Southern Band	+ 10 - 40	0.049	0.347	
	+ 40 - 52	0.364	0.000	
	>52	0.000	0.000	
	Total	0.413	0.347	0.760
Total (NB+MB+SB)		3.295	6.457	9.752

ii) DEPLETION OF RESERVES :

The year wise depletion of reserves during the period 01.04.08 to 31.03.12 was as given in Table 3.5.2

Table 3.5.2
ROM Ore Production
(all figures in million tonnes)

Orebody	Grade ($\%Cr_2O_3$)	Depletion of Reserves (2008-12)									
		2008-09		2009-10		2010-11		2011-12		Total	
		111	122	111	122	111	122	111	122	111	122
Northern Band	+ 10 - 40	0.006		0.045		0.042		0.008		0.101	0.000
	+ 40 - 52	0.003	0.154	0.000	0.165	0.000	0.194	0.000	0.293	0.003	0.806
	>52	0.145		0.089		0.080		0.051		0.365	0.000
	Total	0.154	0.154	0.133	0.165	0.121	0.194	0.059	0.293	0.468	0.806
Mid Band	+ 10 - 40	0.095		0.055		0.000		0.000		0.150	
	+ 40 - 52	0.006	0.517	0.000	0.679	0.000	0.853	0.000	0.901	0.006	2.950
	>52	0.004		0.000		0.000		0.000		0.004	
	Total	0.105	0.517	0.055	0.679	0.000	0.853	0.000	0.901	0.160	2.950
Southern Band	+ 10 - 40	0.074		0.070		0.041		0.082		0.266	0.000
	+ 40 - 52	0.025		0.021		0.015		0.025		0.087	0.000
	>52									0.000	0.000
	Total	0.099	0.000	0.090	0.000	0.056	0.000	0.107	0.000	0.353	0.000
Total (NB+MB+SB)		0.359	0.671	0.278	0.844	0.177	1.047	0.167	1.194	0.981	3.756

Thus 4.376 million tonnes of chrome ore ROM has been mined out during the period of 2008-12.

iii) ADDITIONAL RESERVES ESTABLISED :

As mentioned in the approved Mining Plan the major part of the core drilling done during the period 1998-02 was essentially for establishing the geological structure of the host rock, hydro-geological investigations, rock strength parameters etc. for the study on open pit optimization. Some of the exploratory drilling undertaken during 2002-10 was for the purpose of gap filling and prove the resources at greater depths. The geological sections & ore body model were accordingly updated. The updated geological sections have been enclosed as drawing no SCM/MP/04/12.

The eastern side of the middle chrome ore band, which was earlier planned to be mined up to 140mRL, is now proposed to be mined up to 20mRL by re-handling of the existing dump.

iv)RESOURCE ASSESSMENT OF CHROME ORE

Basis of Resource Estimation

All exploration data were compiled systematically and stored in excel database. Surpac 6.1.3 software was used for modeling and resource estimation. Different steps followed in the modeling of the ore bodies are described below.

Geological Database Creation:

The data consists of coordinates of borehole locations along with their orientation, depth-wise record of lithology and analytical results of samples from boreholes. These were entered in Excel Worksheet files. Proper care was taken to ensure error free exploration database. The basic data was collected and stored in excel CSV format as per the structure below:

Borehole Collar file:

The final collars of boreholes were properly surveyed and those information were stored in the borehole collar file having the following fields: Borehole ID, Northing, Easting, mRL, Total Depth of borehole and Path of borehole.

Borehole Survey file:

Due to near-vertical nature of the ore body, the boreholes were planned inclined to intersect the ore body at desired levels. The borehole deviation data was checked in the field and later on stored systematically in the database. As the general strike of the orebody is 59°, the azimuth of the boreholes was set to be either 149° (for boreholes looking South) and 329° (for boreholes looking North). As the local mine grid is rotated from the true north at an angle of 31°, the actual azimuths of boreholes were later on converted to grid azimuth by addition of 31° to the actual survey data. The survey file contains the following fields: Borehole ID, Depth (at which survey is taken), Inclination, Azimuth and Remarks.

Borehole Geology file:

All the drilled core was systematically logged and the data was stored in digital format. Geology file contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Primary lithology (PCODE), Secondary lithology (SCODE), Remarks.

Borehole Assay file:

Assay file contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Core Recovery, Code (Ore/Waste), Cr_2O_3%, Fe%, SiO_2%, Al_2O_3%, CaO%, MgO%, Mn%, TiO_2%, Ni%, Co% & Remarks.

Zone file:

Zones were defined with Threshold value of 10% Cr_2O_3 as per IBM guideline. It contains the following fields: Borehole ID, Sample ID, Depth From, Depth To, Width, Cr_2O_3%, Fe%, SiO2%, Code (Ore/Waste) & Remarks.

Determination Of Specific Gravity:

Core samples of 8 to 30 cm length, averaging 10cm length, free of visible moisture were selected. The samples were dried, weighed in air and the mass in air (Ma) recorded to the nearest 0.1 g. The sample was then suspended in water and the mass in water (Mw) was recorded. Calculation of the density was done by the following formula:

Density = Ma ⁄ (Ma – Mw)

Nearly 2000 specific gravity measurements of chromite ore & host rocks in the hanging wall & foot wall were taken. Correlation of Cr_2O_3 grade with specific gravity was established for each ore band and correlation equation was developed, which, in turn, was used in block model estimate for specific gravity.

Northern band yielded the following correlation equation:
$y = 0.038x + 1.857$.
Middle band yielded the following correlation equation:
$y = 0.050x + 1.221$.

Southern band yielded the following correlation equation:
$y = 0.028x + 2.578$

Where y = Specific gravity
x = % Cr_2O_3

Solid Body Modeling:

Integrating topographical survey data, geological information and chemical analysis, systematic database was created using SURPAC (6.1.3 version) software. This database was the foundation for 3D modeling of ore body & rock mass, estimation of the grade and calculation of the resources / reserves. The surface topography was sectioned along cross-section lines at 50 m interval (1650E to 4150E) covering area of interest to generate topographical profiles for cross-sections preparation. String files of borehole trace, lithology, assay and ore zones in boreholes along 52 transverse sections from 1650E to 4150E at 50 m interval were extracted from geological database for correlation. The transverse sections thus generated were correlated for continuity of ore body and other lithologies. The chromite body and the other lithologies were correlated upto the present topography as per the topographical profiles along each section and the boundary strings for each lithology were digitised.

This was followed by litho-wise stitching of the ore/lithologies to have a model of each ore band and individual lithologies. All ore/litho solid were combined to get the 3D litho-model of the entire leasehold. The snapshot of the orebody model showing all three orebodies is given below in Fig 3.5.

Determination of hard-soft Boundary in the Northern & Middle ore bands:

The character of the weathered material in the Northern & Middle chromite ore bands goes from soil-like near the surface to weak friable ore. The unweathered ore is fairly hard and lumpy and there is a transition zone where the ore goes from friable to lumpy. The soft powdery ore above the lumpy ore is un-mineable by conventional underground methods of mining.

As no methodology has been suggested till date for mining the friable ore from underground, for Northern and Middle Band, one more exercise was carried out to determine the boundary between Friable & Lumpy Ore. For determining the same, all available drilled core in the ore zone was critically re-logged to see whether the ore is of friable or lumpy in nature. The collected data was plotted in the Longitudinal Vertical Section. The boundary between friable and lumpy portion of the individual ore body were delineated from the LVS which was followed by creation of the Friable-Lumpy Boundary DTM. The depth of the weathering for the North and Middle Bands varies from about 0 mRL in the west to -250 mRL in the east for the North Band, while the same for the Middle Band is less the -200 mRL.

The schematic diagrams of the longitudinal sections showing the above weathering profile in the Northern and Middle chromite ore bands is given below in Fig.'s 3.6 to 3.9.

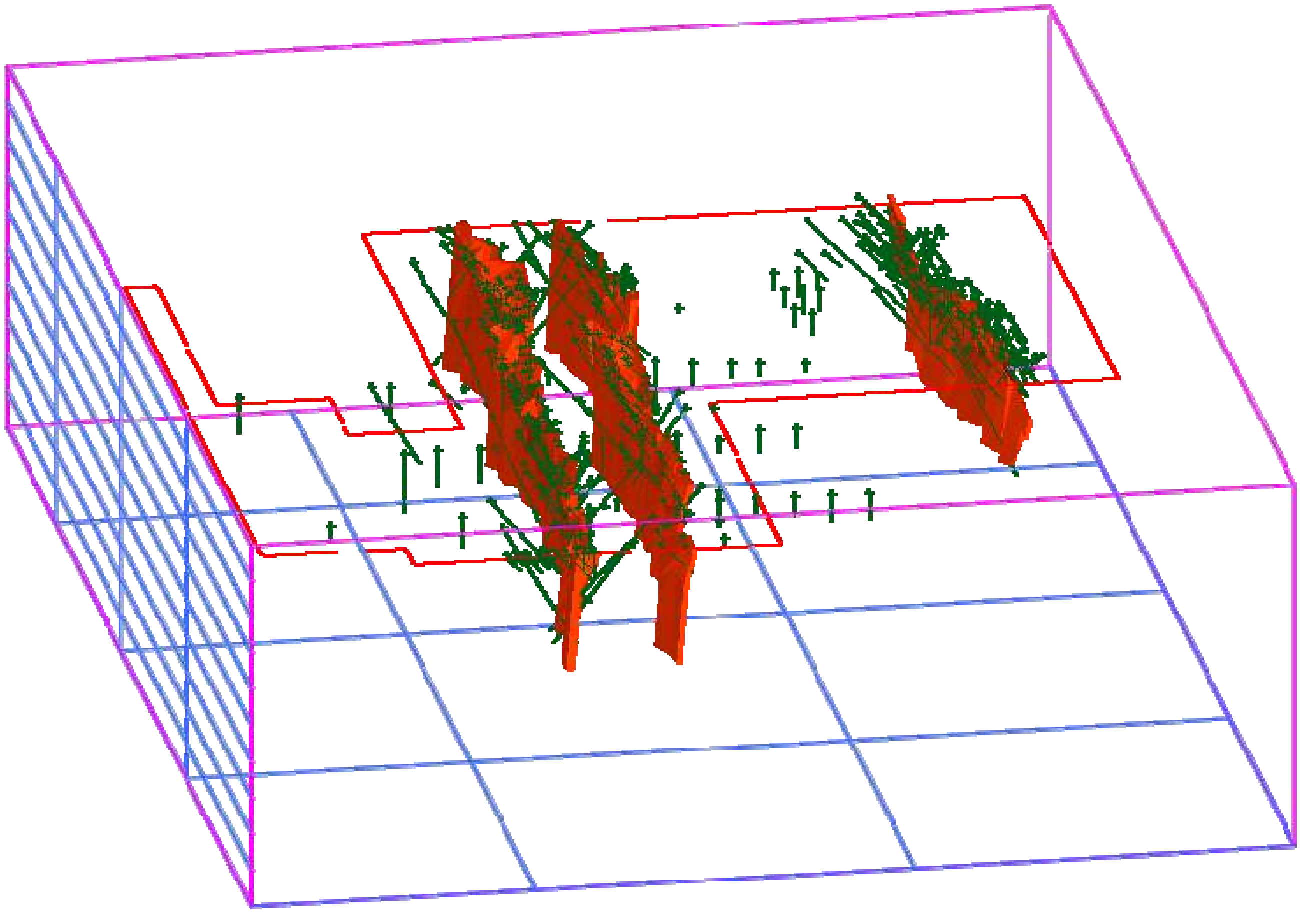


Figure 3.5: Snapshot of the ore body model showing all three chrome ore bands

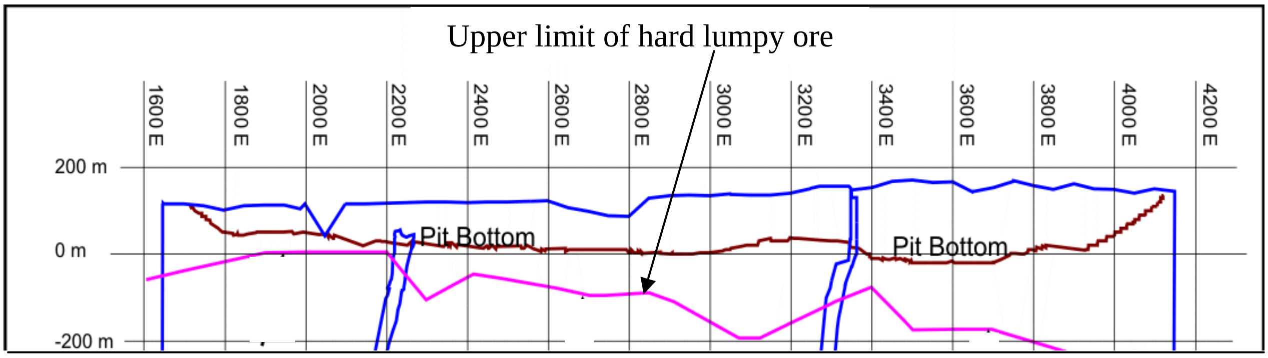


Fig. 3.6: Longitudinal Section of North Band showing Weathering Profile

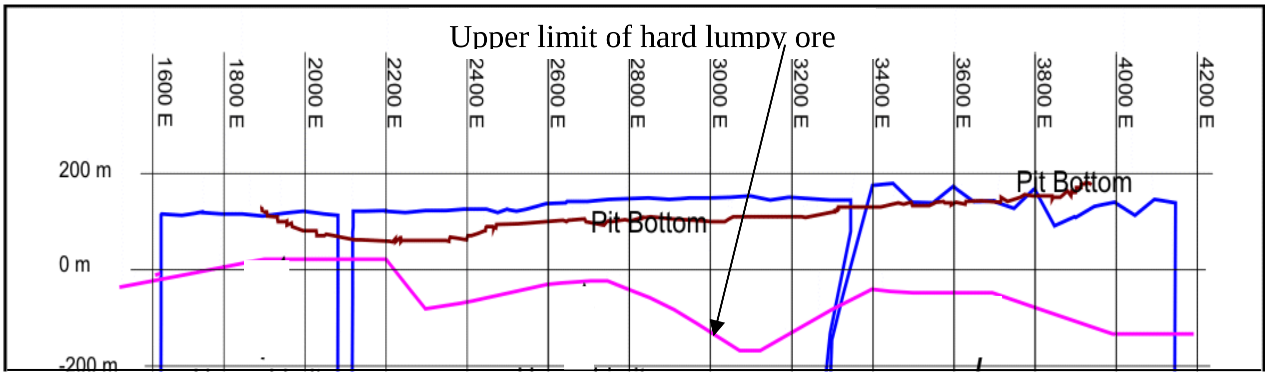


Fig. 3.7: Longitudinal Section of Middle Band showing Weathering Profile

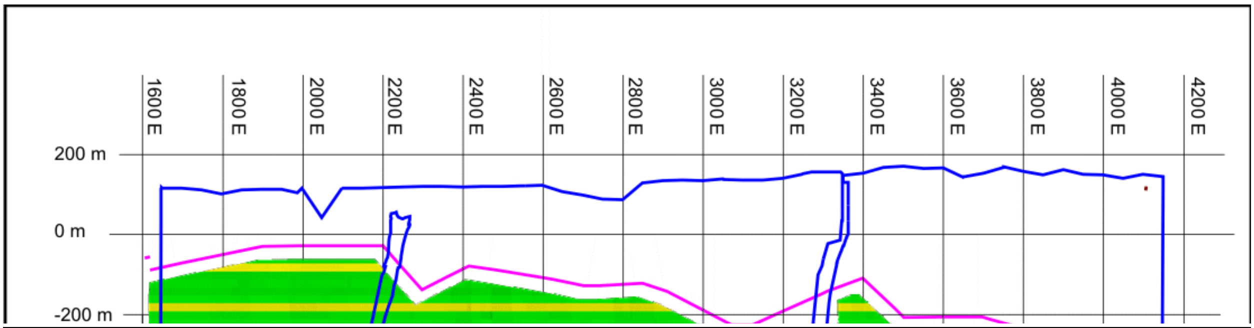


Fig. 3.8: Longitudinal Section of North Band Hard Lumpy Ore Zone (in green)

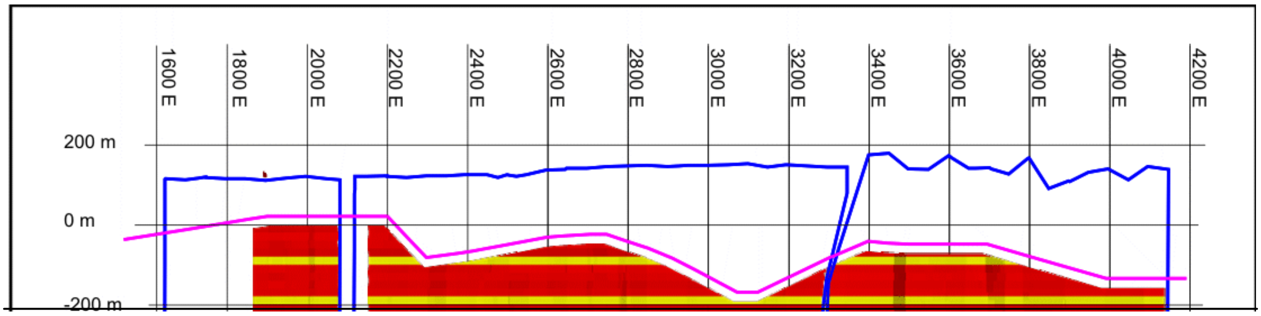


Fig. 3.9: Longitudinal Section of Middle Band Hard Lumpy Ore Zone (in red)

Preparation Of Block Model:

For block model estimation, composite samples were made at 2 m intervals inside the wireframe model and basic statistical values were determined. Geostatistical analysis was carried out to understand the distribution type, variance, standard deviation, etc, followed by variogram modeling. For each variable (i.e. Cr_2O_3, Fe and SiO_2), variogram along three directions i.e. along strike, along dip of the ore body and across strike of the ore body was generated. Based on the experimental curve generated by the data points, a model curve was fitted to smoothen the experimental curve and range, nugget (C0) and sill (C1) were calculated in each direction of the variogram.

These variogram parameters obtained were used in the estimation by Ordinary Kriging. While defining the search ellipse parameter for estimation, the search ellipse was defined considering the disposition of the ore body. Because the ore body is striking in E-W direction and dipping approximately 80^0 towards north, bearing of the search ellipse was considered 90^0 and dip at 80^0, so that the search ellipse move in down dip direction while estimation.

v) BASIS OF RESOURCE CATEGORISATION AS PER UNFC:

For delineation of the resource boundaries field guidelines of UNFC were followed. The following criteria were taken under consideration

For Measured Resource: 100 m (laterally) x 50 m (vertically) grid spacing
For Indicated Resource: 200-300m (laterally) x 100–150 m (vertically) grid spacing
For Inferred Resource: 300 m grid spacing

The depth extension parameters for resource estimation have been considered to be 25m below the positive intersection depth for the Measured Resources. For Indicated resources, where there is lack of borehole intersection, depth influence of 25 m below the measured resource boundary has been considered. Similarly for Inferred resources, a depth influence of 25 m below the indicated resource boundary has been considered. For all the categories, 10% Cr_2O_3 cut-off grade has been considered for the resource estimation.

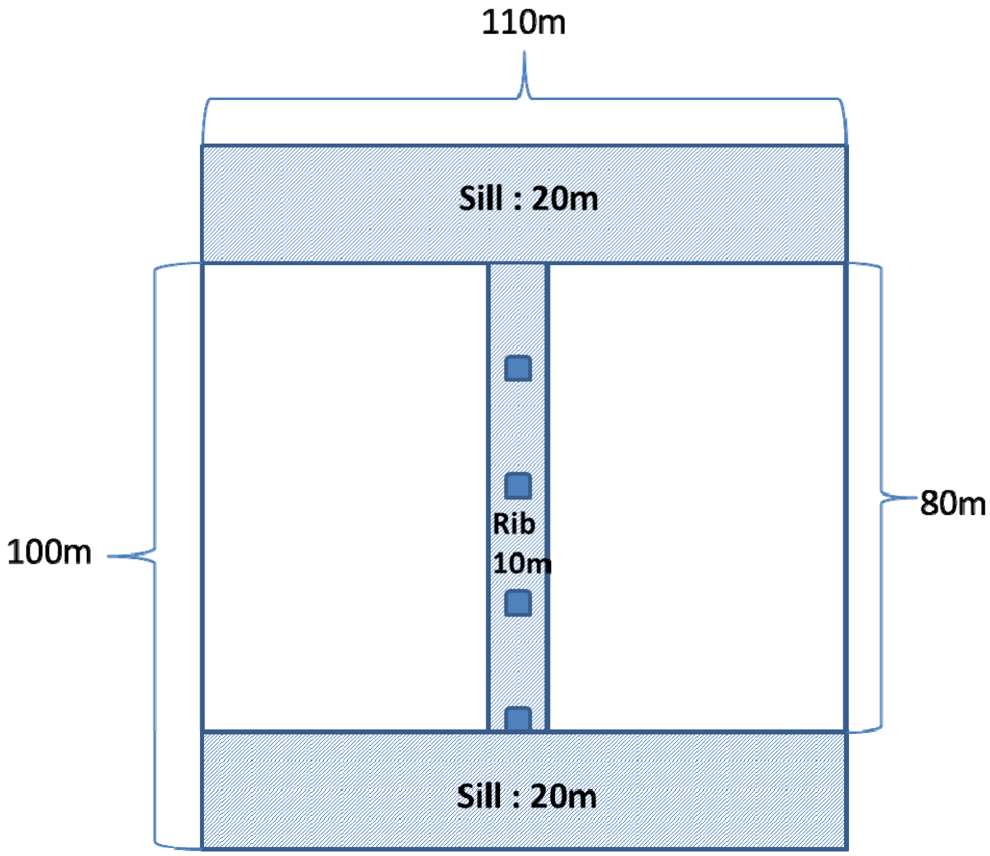
Thereafter, the resultant mineable reserves and presently non-mineable resources has been derived out of resources after due consideration of the feasibility studies and with consideration of modifying factors (UPL, Statutory barriers ,pillars, geotechnical reasons, environmental/forest clearances etc.) for both underground and opencast.

Crown Pillar:

The development within the orebody is proposed to commence below the crown pillars which has been shown in the longitudinal geological section for all 3 orebodies (Drawing no.'s SCM/MP/21/12). As per geotechnical investigation report (enclosed as Annexure-22, Ref. pp-

53) it is stated that a crown pillar will be unstable in the soft friable chrome ore material and pillars can therefore not be developed in this material. Accordingly, the crown pillar has been designed to be below the weathered horizon of the Northern and Middle ore bands and within the hard and competent ore. Presently the thickness of the crown pillar for all three ore bodies has been proposed to be 60m in line with the requirements of the statute.

Sill & Rib Pillar:



Total Area of the Block = 110m * 100m =11000m^2
The Area blocked in Sill & Rib Pillar = (110m * 20m) + (80m * 10m) = 3000m^2
Hence the % of area blocked in Sill & Rib Pillar = (3000/110000) *100% =72.87%

As a conservative approach, the % of resource blocked in Sill & Rib pillars has been taken as 30%

The conceptual longitudinal vertical section to elaborate the basis of categorization as per UNFC for the Northern & Middle chromite ore bands are given below in Fig 3.10 and for Southern band is given below in Fig 3.11.

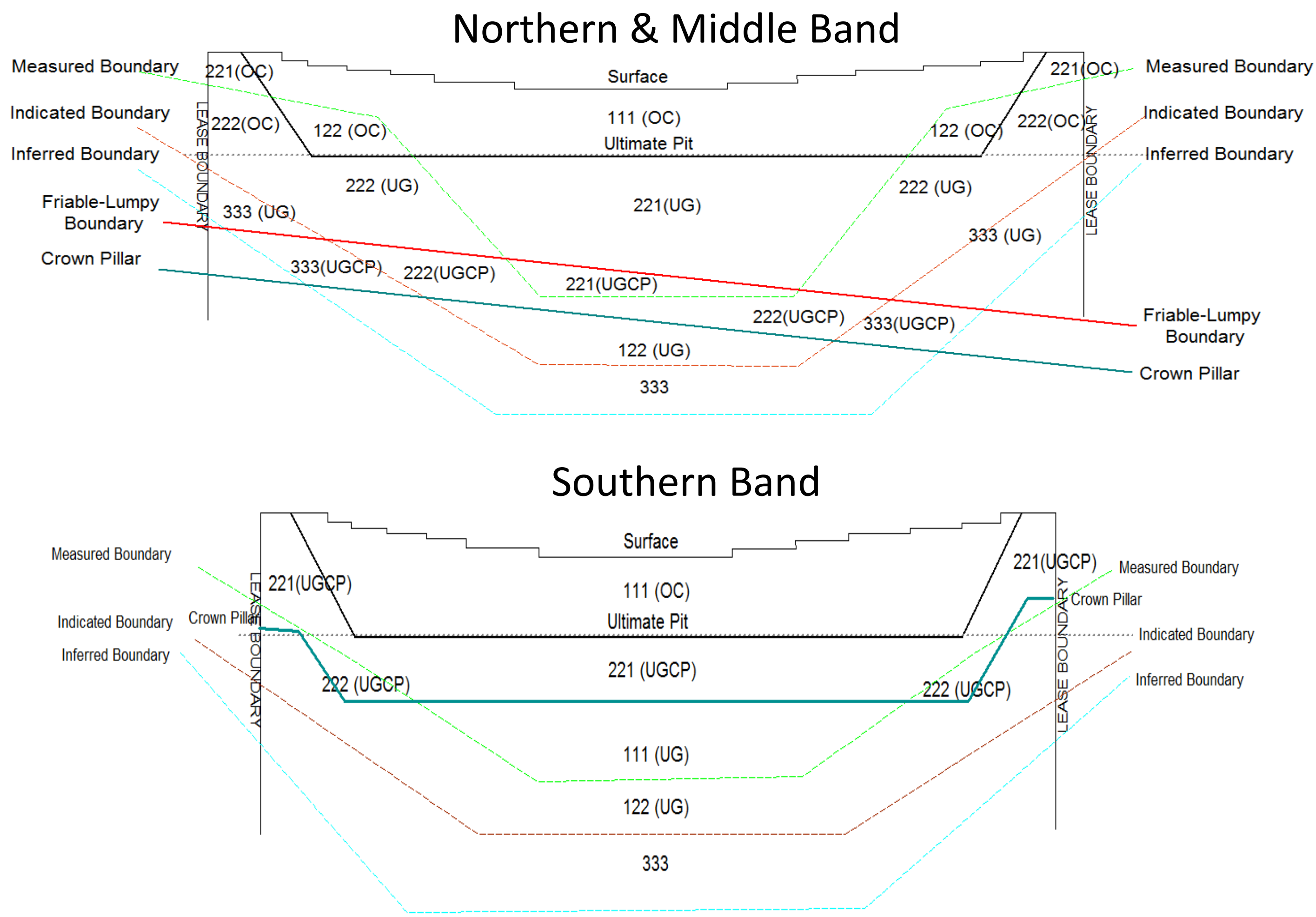


Fig. 3.1.1: Conceptual Longitudinal Vertical Section

Based on above facts and information/data generated during the course of exploration and proposed change in mining practices, additional resource for chrome ore was established as given in Table 3.3.1. The justification of the UNFC codes has been given in the Feasibility Report which is enclosed as Annexure-33.

Table 3.3.1
Addition of Resources (2008-12)
(in Million Tonnes)

Ore Body	Category	UNFC Category	Grade Range, %Cr_2O_3	Addition of Resources (Million tonnes)
Northern Band	Proved Mineral Reserve - Open Cast	111	10-40	0.332
			40 - 52	1.588
			>52	-0.520
	Probable Mineral Reserve - Open Cast	122	10-40	-0.596
			40 - 52	-0.364
			>52	-0.063
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	211	10-40	0.314
			40 - 52	0.710
			>52	0.567
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	222	10-40	0.169
			40 - 52	0.372
			>52	0.038
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	333	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	221	10-40	0.171
			40 - 52	1.422
			>52	5.397
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	222	10-40	1.723
			40 - 52	6.295
			>52	4.162
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	333	10-40	0.290
			40 - 52	1.857
			>52	2.798
	Prefeasibility Mineral Resource blocked in crown pillar	221	10-40	0.000
			40 - 52	0.067
			>52	0.174
	Prefeasibility Mineral Resource blocked in crown pillar	222	10-40	0.234
			40 - 52	2.229
			>52	1.728
	Inferred Mineral Resource blocked in crown pillar	333	10-40	0.134
			40 - 52	1.296
			>52	1.241

	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.406
			40 - 52	0.903
			>52	0.367
	Proved Mineral Reserve - Underground	**111**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	0.947
			40 - 52	2.106
			>52	0.857
	Inferred Mineral Resource - Underground	**333**	10-40	0.540
			40 - 52	2.684
			>52	1.631
	TOTAL RESERVE + RESOURCES			**44.209**
Middle Band	Proved Mineral Reserve - Open Cast	**111**	10-40	2.027
			40 - 52	1.221
			>52	0.078
	Probable Mineral Reserve - Open Cast	**122**	10-40	1.617
			40 - 52	0.808
			>52	0.013
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**211**	10-40	0.299
			40 - 52	0.023
			>52	0.000
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**222**	10-40	0.753
			40 - 52	0.088
			>52	0.000
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	**333**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**221**	10-40	0.003
			40 - 52	0.126
			>52	0.033
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**222**	10-40	2.027
			40 - 52	4.998
			>52	0.586
	Underground Inferred Mineral Resource: Presently non-mineable	**333**	10-40	0.054
			40 - 52	0.208

	because of friable nature of the formation		>52	0.001
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	1.703
			40 - 52	2.754
			>52	0.225
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.122
			40 - 52	0.083
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.686
			40 - 52	0.890
			>52	0.025
	Proved Mineral Reserve - Underground	**111**	10-40	0.000
			40 - 52	0.000
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	1.601
			40 - 52	2.076
			>52	0.059
	Inferred Mineral Resource - Underground	**333**	10-40	0.640
			40 - 52	1.177
			>52	0.020
	TOTAL RESERVE + RESOURCES			**27.024**
Southern Band	Proved Mineral Reserve - Open Cast	**111**	10-40	0.482
			40 - 52	-0.190
			>52	0.000
	Probable Mineral Reserve - Open Cast	**122**	10-40	-0.347
			40 - 52	0.000
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	1.328
			40 - 52	0.364
			>52	0.000
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	0.482
			40 - 52	0.010
			>52	0.000
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.000
			40 - 52	0.000

			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.070
			40 - 52	0.005
			>52	0.000
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	1.145
			40 - 52	0.077
			>52	0.000
	Proved Mineral Reserve - Underground	**111**	10-40	0.163
			40 - 52	0.011
			>52	0.000
	Probable Mineral Reserve - Underground	**122**	10-40	2.671
			40 - 52	0.180
			>52	0.000
	Inferred Mineral Resource - Underground	**333**	10-40	1.530
			40 - 52	0.090
			>52	0.000
	TOTAL RESERVE + RESOURCES			**8.070**

3.1 CATEGORY-WISE UPDATED RESERVE :

The category – wise updated reserves for chrome ore as on 31.3.2008 was 9.752 million tonnes as given in Table- 3.1.1. This was the recoverable reserve considering the resource within the conceptualized ultimate open pit limits only. The band-wise, category-wise mineral resource as on 31.03.2012 for chrome ore, considering the depletion during 2008-12 and subsequent addition of resource was calculated and given in Table - 3.4.1 (a), (b) & (c) below.

The cut-off grade for chrome ore is considered to be 10% Cr_2O_3. Chemical analysis of ore/associated mineral/sub grade/waste from Odisha Govt. lab at Jajpur Road & M/s Shiva Analyticals (India) Ltd., Bangalore (NABL accredited lab) is enclosed as Annexure-32. The company's chemical analysis lab at the mine has also been recognized as a standard reference laboratory by the Directorate of Geology, Govt. of Odisha.

The criteria G1, G2 & G3 level as per UNFC has been synthesized on the basis exploration carried out over entire lease area on spatial and level-wise till date is as follows

Detailed Exploration (G1): 100 m (laterally) x 50 m (vertically) grid spacing
General exploration (G2): 200-300m (laterally) x 100–150 m (vertically) grid spacing
Prospecting (G3): 300 m grid spacing

Accordingly corresponding resource categories have been estimated. The depth extension parameters for resource estimation have been considered to be 25m below the positive intersection depth for the Measured Resources. For Indicated resources, where there is lack of borehole intersection, depth influence of 25 m below the measured resource boundary has been considered. Similarly for Inferred resources, a depth influence of 25 m below the indicated resource boundary has been considered. For all the categories, 10% Cr2O3 cut-off grade has been considered for the resource estimation. The resources under G1, G2 & G3 axis have been further categorized to reserves and remaining resources is given in Table 3.4.1 (a) below.

Table 3.4.1 (a)

Resource category	Northern Band	Middle Band	Southern Band	All total
Measured Mineral Resources (331)	12.476	3.810	2.293	18.579
Indicated Mineral Resources (332)	23.882	20.895	4.565	49.342
Inferred Mineral Resources (333)	12.472	2.305	1.619	16.396

Thereafter, the resultant mineable reserves and presently non-mineable resources has been derived out of resources after due consideration of the feasibility studies and with consideration of modifying factors (UPL, Statutory barriers ,pillars, geotechnical reasons, environmental/forest clearances etc.) for both underground and opencast. The distribution of mineable reserves and remaining resources (non-mineable part) for opencast and underground working as derived from Measured and Indicated resources are as follows:

Table 3.4.1 (b)

RESOURCES IDENTIFIED *(By degree of exploration)*				RESERVES AND REMAINING RESOURCES AFTER CONDUCTING FEASIBLITY STUDIES for O/C *(with consideration of modifying factors)*					
Resource category	Northern Band	Middle Band	Southern Band	Description	UNFC STD	Remarks	Northern Band	Middle Band	Southern Band
Measured Mineral Resources (331)	12.476	3.810	2.293	Above Ultimate Pit(O/C)	111	Open Cast	3.654	3.326	0.351
					211	Lease Boundary constraint	1.592	0.323	0.000
				Below Ultimate Pit (U/G)	121	Underground	0.000	0.000	0.174
					221	Soft Friable Ore	6.991	0.161	0.000
						Crown Pillar	0.241	0.000	1.693
						Sill & Rib Pillar	0.000	0.000	0.075
Indicated Mineral Resources (332)	23.882	20.895	4.565	Above Ultimate Pit(O/C)	122	Open Cast	1.345	2.424	0.000
					222	Lease Boundary constraint	0.579	0.841	0.000
				Below Ultimate Pit(U/G)	122	Underground	3.911	3.736	2.852
					222	Soft Friable Ore	12.181	7.611	0.000
						Crown Pillar	4.190	4.682	0.491
						Sill & Rib Pillar	1.676	1.601	1.222

Table 3.4.1 (c)
Resource of Chrome Ore as on 31.03.2012, (in Million Tonnes)

Ore Body	Category	UNFC Category	Grade Range, %Cr_2O_3	Updated Resources (As on 31.03.2012)	
				Tonnage	**Total**
Northern Band	Proved Mineral Reserve - Open Cast	**111**	10-40	0.283	3.653
			40 - 52	1.588	
			>52	1.782	
	Probable Mineral Reserve - Open Cast	**122**	10-40	0.203	1.345
			40 - 52	0.863	
			>52	0.279	
	Open cast Feasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**211**	10-40	0.314	1.591
			40 - 52	0.710	
			>52	0.567	
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**222**	10-40	0.169	0.579
			40 - 52	0.372	
			>52	0.038	
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	**333**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**221**	10-40	0.171	6.990
			40 - 52	1.422	
			>52	5.397	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**222**	10-40	1.723	12.180
			40 - 52	6.295	
			>52	4.162	
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	**333**	10-40	0.290	4.945
			40 - 52	1.857	
			>52	2.798	
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	0.000	0.241
			40 - 52	0.067	
			>52	0.174	
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	0.234	4.191
			40 - 52	2.229	
			>52	1.728	
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.134	2.671
			40 - 52	1.296	
			>52	1.241	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000	0.000
			40 - 52	0.000	

			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.406	1.676
			40 - 52	0.903	
			>52	0.367	
	Proved Mineral Reserve - Underground	**111**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	0.947	3.910
			40 - 52	2.106	
			>52	0.857	
	Inferred Mineral Resource - Underground	**333**	10-40	0.540	4.855
			40 - 52	2.684	
			>52	1.631	
	TOTAL RESERVE + RESOURCES				**48.83**
Middle Band	Proved Mineral Reserve - Open Cast	**111**	10-40	2.027	3.326
			40 - 52	1.221	
			>52	0.078	
	Probable Mineral Reserve - Open Cast	**122**	10-40	1.604	2.425
			40 - 52	0.808	
			>52	0.013	
	Open cast Feasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**211**	10-40	0.299	0.322
			40 - 52	0.023	
			>52	0.000	
	Open cast Prefeasibility Mineral Resource: Presently non-mineable because of lease boundary constraint	**222**	10-40	0.753	0.841
			40 - 52	0.088	
			>52	0.000	
	Open cast Inferred Mineral Resource: Presently non-mineable because of lease boundary constraint	**333**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**221**	10-40	0.003	0.162
			40 - 52	0.126	
			>52	0.033	
	Underground Prefeasibility Mineral Resource: Presently non-mineable because of friable nature of the formation	**222**	10-40	2.027	7.611
			40 - 52	4.998	
			>52	0.586	
	Underground Inferred Mineral Resource: Presently non-mineable because of friable nature of the formation	**333**	10-40	0.054	0.263
			40 - 52	0.208	
			>52	0.001	
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	

	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	1.703	4.682
			40 - 52	2.754	
			>52	0.225	
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.122	0.205
			40 - 52	0.083	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**222**	10-40	0.686	1.601
			40 - 52	0.890	
			>52	0.025	
	Proved Mineral Reserve - Underground	**111**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	1.601	3.736
			40 - 52	2.076	
			>52	0.059	
	Inferred Mineral Resource - Underground	**333**	10-40	0.640	1.837
			40 - 52	1.177	
			>52	0.020	
	TOTAL RESERVE + RESOURCES				**27.01**
Southern Band	Proved Mineral Reserve - Open Cast	**111**	10-40	0.264	0.351
			40 - 52	0.087	
			>52	0.000	
	Probable Mineral Reserve - Open Cast	**122**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in crown pillar	**221**	10-40	1.328	1.692
			40 - 52	0.364	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in crown pillar	**222**	10-40	0.482	0.492
			40 - 52	0.010	
			>52	0.000	
	Inferred Mineral Resource blocked in crown pillar	**333**	10-40	0.000	0.000
			40 - 52	0.000	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in stope (rib/sill) pillar	**221**	10-40	0.070	0.075
			40 - 52	0.005	
			>52	0.000	
	Prefeasibility Mineral Resource blocked in	**222**	10-40	1.145	1.222

	stope (rib/sill) pillar		40 - 52	0.077	
			>52	0.000	
	Proved Mineral Reserve - Underground	**111**	10-40	0.163	0.174
			40 - 52	0.011	
			>52	0.000	
	Probable Mineral Reserve - Underground	**122**	10-40	2.671	2.851
			40 - 52	0.180	
			>52	0.000	
	Inferred Mineral Resource - Underground	**333**	10-40	1.530	1.620
			40 - 52	0.090	
			>52	0.000	
	TOTAL RESERVE + RESOURCES				**8.477**

The level wise reserve and resource for all three ore bodies is given in Table 3.4.1.1 to 3.4.1.3.

Table 3.4.1.1

NORTHERN BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Mining Method	From (mRL)	To (mRL)	Grade (% Cr_2O_3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable		Total	Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 211	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total		Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	**Sub Total**		
					A		B	**C=(A+B)**	D	E	F	G	**H=(D+E+F+G)**	I	J	K	L	**M=(I+J+K+L)**	N	O	P	Q	**R=(N+O+P+Q)**	**S=(H+M+R)**	**T=(C+S)**
Open Cast	145	135	+10 - 40						0.000																
			+40 - 52						0.000				**0.000**					**0.000**					**0.000**	0.000	**0.000**
			+52						0.000																
	135	125	+10 - 40						0.000																
			+40 - 52						0.009				**0.009**					**0.000**					**0.000**	0.009	**0.009**
			+52						0.000																
	125	115	+10 - 40						0.001																
			+40 - 52						0.027				**0.028**					**0.000**					**0.000**	0.028	**0.028**
			+52						0.000																
	115	105	+10 - 40						0.020																
			+40 - 52						0.041				**0.060**					**0.000**					**0.000**	0.060	**0.060**
			+52						0.000																
	105	95	+10 - 40	0.003					0.026																
			+40 - 52	0.003	**0.009**			**0.009**	0.074				**0.113**					**0.000**					**0.000**	0.113	**0.122**
			+52	0.002					0.012																
	95	85	+10 - 40	0.001					0.039																
			+40 - 52	0.007	**0.022**			**0.022**	0.072				**0.163**					**0.000**					**0.000**	0.163	**0.185**
			+52	0.014					0.051																
	85	75	+10 - 40	0.000					0.040																
			+40 - 52	0.008	**0.021**			**0.021**	0.090				**0.194**					**0.000**					**0.000**	0.194	**0.216**
			+52	0.014					0.065																
	75	65	+10 - 40	0.012		0.006			0.051																
			+40 -	0.037	**0.072**	0.006	**0.012**	**0.084**	0.085				**0.213**					**0.000**					**0.000**	0.213	**0.297**
			+52	0.022		0.000			0.077																
	65	55	+10 -	0.049		0.013			0.050					0.004											
			+40 -	0.062	**0.157**	0.033	**0.052**	**0.209**	0.082				**0.220**	0.006				**0.010**					**0.000**	0.230	**0.439**
			+52	0.046		0.006			0.087					0.000											
	55	45	+10 -	0.074		0.020			0.027					0.035											
			+40 -	0.215	**0.510**	0.105	**0.162**	**0.672**	0.072				**0.181**	0.026				**0.061**					**0.000**	0.242	**0.915**
			+52	0.221		0.037			0.083					0.000											
	45	35	+10 -	0.057		0.019			0.022					0.039											
			+40 -	0.359	**0.631**	0.182	**0.300**	**0.931**	0.070				**0.158**	0.057				**0.098**					**0.000**	0.255	**1.186**
			+52	0.215		0.099			0.065					0.001											

			+10 -	0.043		0.051			0.020					0.046											
	35	25	+40 -	0.378	**0.820**	0.251	**0.389**	**1.210**	0.055				**0.144**	0.093				**0.142**					**0.000**	0.286	**1.495**
			+52	0.400		0.087			0.069					0.003											
			+10 -	0.043		0.095			0.018					0.045											
	25	15	+40 -	0.497	**1.375**	0.282	**0.425**	**1.800**	0.033				**0.108**	0.189				**0.268**					**0.000**	0.376	**2.176**
			+52	0.834		0.048			0.057					0.034											
			+10 -	0.000		0.000			0.000					0.000											
	15	10	+40 -	0.021	**0.036**	0.003	**0.004**	**0.041**	0.000				**0.000**	0.000				**0.000**					**0.000**	0.000	**0.041**
			+52	0.015		0.001			0.000					0.000											
Sub Total			**49.13**		**3.654**		**1.345**	**4.999**	**1.592**	**0.000**	**0.000**	**0.000**	**1.592**	**0.579**	**0.000**	**0.000**	**0.000**	**0.579**	**0.000**	**0.000**	**0.000**	**0.000**	**0.000**	**2.171**	**7.170**
			+10 -							0.087					0.224					0.011					
	15	0	+40 -							0.659			**1.861**		0.753			**1.381**		0.011			**0.025**	3.267	**3.267**
			+52							1.115					0.404					0.002					
			+10 -							0.084	0.000				0.651	0.165				0.147	0.000	0.003			
	0	-60	+40 -							0.755	0.009		**4.355**		2.685	0.617		**6.604**		0.555	0.008	0.014	**1.259**	12.218	**12.218**
			+52							3.471	0.035				1.862	0.624				0.512	0.020	0.001			
			+10 -			0.048				0.000	0.000				0.033	0.007	0.020			0.069	0.000	0.000			
	-60	-71	+40 -			0.038	**0.146**	**0.146**		0.005	0.016		**0.372**		0.368	0.093	0.016	**1.057**		0.165	0.022	0.006	**0.491**	1.921	**2.067**
			+52			0.060				0.320	0.032				0.448	0.047	0.026			0.226	0.003	0.001			
			+10 -			0.119				0.000	0.000				0.133	0.012	0.051			0.060	0.015	0.021			
Underground	-71	-95	+40 -			0.172	**0.397**	**0.397**		0.000	0.031		**0.510**		0.678	0.185	0.074	**2.234**		0.231	0.019	0.022	**0.978**	3.722	**4.119**
			+52			0.106				0.425	0.053				0.848	0.209	0.045			0.560	0.047	0.002			
			+10 -			0.617				0.000	0.000				0.683	0.046	0.264			0.002	0.009	0.195			
	-95	-195	+40 -			1.464	**2.383**	**2.383**		0.003	0.011		**0.133**		1.811	1.335	0.627	**6.338**		0.828	0.668	0.382	**4.627**	11.098	**13.482**
			+52			0.302				0.065	0.053				0.601	0.840	0.130			1.414	0.773	0.359			
			+10 -			0.163									0.000	0.003	0.070			0.001	0.111	0.322			
	-195	-295	+40 -			0.432	**0.984**	**0.984**					**0.000**		0.000	0.000	0.185	**0.433**		0.066	0.579	1.605	**4.225**	4.658	**5.642**
			+52			0.389									0.000	0.007	0.167			0.083	0.399	1.058			
			+10 -																			0.000			
	-295	-381	+40 -										**0.000**					**0.000**				0.655	**0.866**	0.866	**0.866**
			+52																			0.211			
Sub Total			**45.49**				**3.911**	**3.911**	0.000	6.991	0.241	0.000	**7.231**	0.000	12.181	4.190	1.676	**18.04**	0.000	4.945	2.672	4.855	**12.47**	**37.750**	**41.661**
GRAND TOTAL								**8.909**	1.592	6.991	0.241	0.000	**8.823**	0.579	12.181	4.190	1.676	**18.62**	0.000	4.945	2.672	4.855	**12.47**	**39.921**	**48.830**

* Presently Non Mineable

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Table No.. 3.4.1.2

MIDDLE BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Middle Band	From (mRL)	To (mRL)	Grade (%Cr2O3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable			Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 211	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	**Sub Total**	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	**Sub Total**		
					A		B	**C=(A +B)**	D	E	F	G	**H=(D +E+F +G)**	I	J	K	L	**M=(I +J+K +L)**	N	O	P	Q	**R=(N +O+ P+Q)**	**S=(H +M+ R)**	**T= (C+S)**
Open Cast			+10 - 40	0.000																					
	170	160	+40 - 52	0.000	**0.000**			**0.000**					**0.000**					**0.000**					**0.000**	**0.000**	**0.000**
			+52	0.000																					
			+10 - 40	0.003																					
	160	150	+40 - 52	0.000	**0.004**			**0.004**					**0.000**					**0.000**					**0.000**	**0.000**	**0.004**
			+52	0.000																					
			+10 - 40	0.021					0.000																
	150	140	+40 - 52	0.007	**0.028**			**0.028**	0.000				**0.000**					**0.000**					**0.000**	**0.000**	**0.028**
			+52	0.000					0.000																
			+10 - 40	0.084					0.012					0.000											
	140	130	+40 - 52	0.034	**0.118**			**0.118**	0.000				**0.012**	0.000				**0.000**					**0.000**	**0.013**	**0.130**
			+52	0.000					0.000					0.000											
			+10 - 40	0.148		0.001			0.006					0.013											
	130	120	+40 - 52	0.053	**0.201**	0.000	**0.001**	**0.201**	0.000				**0.006**	0.000				**0.013**					**0.000**	**0.019**	**0.220**
			+52	0.000		0.000			0.000					0.000											
			+10 - 40	0.176		0.007			0.020					0.025											
	120	110	+40 - 52	0.066	**0.242**	0.000	**0.007**	**0.249**	0.002				**0.022**	0.000				**0.025**					**0.000**	**0.048**	**0.297**
			+52	0.000		0.000			0.000					0.000											
			+10 - 40	0.195		0.019			0.043					0.031											
	110	100	+40 - 52	0.096	**0.291**	0.000	**0.019**	**0.310**	0.001				**0.043**	0.000				**0.031**					**0.000**	**0.075**	**0.384**
			+52	0.000		0.000			0.000					0.000											
			+10 - 40	0.189		0.044			0.047					0.039											
	100	90	+40 - 52	0.097	**0.286**	0.000	**0.044**	**0.330**	0.002				**0.049**	0.000				**0.039**					**0.000**	**0.088**	**0.418**
			+52	0.000		0.000			0.000					0.000											
			+10 - 40	0.165		0.073			0.048					0.049											
	90	80	+40 - 52	0.116	**0.282**	0.000	**0.074**	**0.356**	0.002				**0.050**	0.000				**0.049**					**0.000**	**0.099**	**0.455**
			+52	0.002		0.000			0.000					0.000											
			+10 - 40	0.228		0.107			0.040					0.051											
	80	70	+40 - 52	0.116	**0.355**	0.022	**0.131**	**0.486**	0.011				**0.051**	0.004				**0.055**					**0.000**	**0.106**	**0.592**
			+52	0.011		0.002			0.000					0.000											

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			+10 - 40	0.251		0.161			0.044					0.054											
	70	60	+40 - 52	0.156	**0.420**	0.050	**0.211**	**0.631**	0.002				**0.046**	0.011				**0.065**					**0.000**	**0.112**	**0.742**
			+52	0.013		0.000			0.000					0.000											
			+10 - 40	0.226		0.226			0.033					0.069											
	60	50	+40 - 52	0.152	**0.386**	0.113	**0.339**	**0.725**	0.001				**0.034**	0.014				**0.083**					**0.000**	**0.117**	**0.842**
			+52	0.008		0.000			0.000					0.000											
			+10 - 40	0.211		0.284			0.006					0.109											
	50	40	+40 - 52	0.145	**0.368**	0.153	**0.437**	**0.805**	0.000				**0.007**	0.009				**0.118**					**0.000**	**0.125**	**0.930**
			+52	0.012		0.000			0.000					0.000											
			+10 - 40	0.109		0.329			0.000					0.137											
	40	30	+40 - 52	0.106	**0.230**	0.210	**0.539**	**0.769**	0.000				**0.000**	0.016				**0.153**					**0.000**	**0.153**	**0.922**
			+52	0.015		0.000			0.000					0.000											
			+10 - 40	0.020		0.340			0.000					0.168											
	30	20	+40 - 52	0.074	**0.112**	0.246	**0.596**	**0.708**	0.001				**0.001**	0.033				**0.201**					**0.000**	**0.202**	**0.910**
			+52	0.017		0.010			0.000					0.000											
			+10 - 40	0.000		0.013			0.000					0.006											
	20	15	+40 - 52	0.002	**0.003**	0.012	**0.027**	**0.030**	0.000				**0.000**	0.002				**0.008**					**0.000**	0.008	**0.038**
			+52	0.001		0.001			0.000					0.000											
Sub Total			**37.59**		**3.326**		**2.424**	**5.750**	0.323	0.000	0.000	0.000	**0.323**	0.841	0.000	0.000	0.000	**0.841**	0.000	0.000	0.000	0.000	**0.000**	**1.163**	**6.914**
			+10 - 40							0.003					0.613	0.151									
	20	0	+40 - 52				**0.000**	**0.000**		0.096			**0.120**		0.751	0.050		**1.665**					**0.000**	**1.785**	**1.785**
			+52							0.021					0.100	0.000									
			+10 - 40			0.026				0.000					0.939	0.446	0.011			0.000	0.007	0.000			
	0	-51	+40 - 52			0.028	**0.056**	**0.056**		0.030			**0.041**		2.451	0.355	0.012	**4.636**		0.042	0.000	0.002	**0.050**	**4.728**	**4.784**
			+52			0.002				0.012					0.328	0.092	0.001			0.000	0.000	0.000			
			+10 - 40			0.152									0.386	0.514	0.065			0.036	0.012	0.010			
Underground	-51	-95	+40 - 52			0.239	**0.402**	**0.402**					**0.000**		1.042	1.083	0.102	**3.328**		0.125	0.002	0.020	**0.206**	**3.534**	**3.937**
			+52			0.011									0.080	0.051	0.005			0.001	0.000	0.000			
			+10 - 40			0.968									0.089	0.590	0.415			0.018	0.103	0.193			
	-95	-195	+40 - 52			1.393	**2.377**	**2.377**					**0.000**		0.753	1.048	0.597	**3.658**		0.041	0.068	0.441	**0.865**	**4.522**	**6.899**
			+52			0.016									0.077	0.083	0.007			0.000	0.000	0.000			
			+10 - 40			0.455										0.002	0.195				0.000	0.437			
	-195	-285	+40 - 52			0.416	**0.901**	**0.901**					**0.000**			0.218	0.178	**0.606**			0.013	0.714	**1.184**	**1.791**	**2.691**
			+52			0.030										0.000	0.013				0.000	0.020			
Sub Total			**40.96**				**3.736**	**3.736**	0.000	0.161	0.000	0.000	**0.161**	0.000	7.611	4.682	1.601	**13.894**	0.000	0.264	0.205	1.837	**2.305**	**16.360**	**20.096**
GRAND TOTAL								**9.486**	0.323	0.161	0.000	0.000	**0.484**	0.841	7.611	4.682	1.601	**14.735**	0.000	0.264	0.205	1.837	**2.305**	**17.524**	**27.010**

* Presently Non Mineable Resource

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Table No. 3.4.1.3

SOUTHERN BAND LEVEL WISE/GRADE WISE RESOURCE (AS ON 31.03.2012)

Southern Band	From (mRL)	To (mRL)	Grade (%Cr2O3)	Reserves					Remaining Resources																Total Resources
				Proved		Probable			Prefeasibility										Inferred					Total	
				STD 111		STD 122			STD 221	STD 221				STD 222					STD 333						
				Million Tonne	Total	Million Tonne	Total	Sub Total	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	Sub Total	Lease Boundary constraint	Soft Friable	Crown pillar	Sill/Rib pillar	Sub Total	Lease Boundary constraint	Soft Friable	Crown pillar	Inferred Resource	Sub Total		
					A		B	C=(A+B)	D	E	F	G	H=(D+E+F+G)	I	J	K	L	M=(I+J+K+L)	N	O	P	Q	R=(N+O+P+Q)	S=(H+M+R)	T=(C+S)
Open Cast	115	105	+10 - 40	0.000																					
			+40 - 52	0.000	**0.000**			**0.000**																0.000	**0.000**
			+52	0.000																					
	105	95	+10 - 40	0.003																					
			+40 - 52	0.002	**0.005**			**0.005**																0.000	**0.005**
			+52	0.000																					
	95	85	+10 - 40	0.005																					
			+40 - 52	0.010	**0.015**			**0.015**																0.000	**0.015**
			+52	0.000																					
	85	75	+10 - 40	0.004																					
			+40 - 52	0.016	**0.020**			**0.020**																0.000	**0.020**
			+52	0.000																					
	75	65	+10 - 40	0.028																					
			+40 - 52	0.006	**0.034**			**0.034**																0.000	**0.034**
			+52	0.000																					
	65	55	+10 - 40	0.018																					
			+40 - 52	0.001	**0.019**			**0.019**																0.000	**0.019**
			+52	0.000																					
	55	45	+10 - 40	0.036																					
			+40 - 52	0.005	**0.041**			**0.041**																0.000	**0.041**
			+52	0.000																					
	45	35	+10 - 40	0.128																					
			+40 - 52	0.036	**0.164**			**0.164**																0.000	**0.164**
			+52	0.000																					
	35	30	+10 - 40	0.043																					
			+40 - 52	0.010	**0.053**			**0.053**																0.000	**0.053**
			+52	0.000																					
Sub Total			34.85		**0.351**			**0.351**					**0.000**					**0.000**					**0.000**	**0.000**	**0.351**
ergr oun	175	105	+10 - 40								0.242														
			+40 - 52		**0.000**			**0.000**			0.027		**0.269**											0.269	**0.269**
			+52								0.000														

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		+10 - 40	0.025		0.002					0.091	0.011					0.001								
105	89	+40 - 52	0.000	**0.025**	0.000	**0.002**	**0.027**			0.034	0.000	**0.135**				0.000	**0.001**						0.136	**0.163**
		+52	0.000		0.000					0.000	0.000					0.000								
		+10 - 40	0.107		0.063					0.235	0.046					0.027								
89	35	+40 - 52	0.007	**0.113**	0.000	**0.063**	**0.176**			0.127	0.003	**0.411**				0.000	**0.027**						0.438	**0.614**
		+52	0.000		0.000					0.000	0.000					0.000								
		+10 - 40	0.006		0.013					0.511	0.003				0.068	0.005					0.001			
35	0	+40 - 52	0.004	**0.011**	0.000	**0.013**	**0.023**			0.157	0.002	**0.672**			0.002	0.000	**0.075**				0.000	**0.001**	0.749	**0.772**
		+52	0.000		0.000					0.000	0.000				0.000	0.000					0.000			
		+10 - 40	0.012		0.115					0.249	0.005				0.414	0.049					0.028			
0	-31	+40 - 52	0.000	**0.012**	0.007	**0.122**	**0.135**			0.020	0.000	**0.274**			0.007	0.003	**0.474**				0.000	**0.028**	0.777	**0.911**
		+52	0.000		0.000					0.000	0.000				0.000	0.000					0.000			
		+10 - 40	0.013		1.613						0.006					0.691					0.225			
-31	-95	+40 - 52	0.000	**0.013**	0.151	**1.765**	**1.778**				0.000	**0.006**				0.065	**0.756**				0.000	**0.225**	0.987	**2.765**
		+52	0.000		0.000						0.000					0.000					0.000			
		+10 - 40			0.866											0.371					1.268			
-95	-195	+40 - 52		**0.000**	0.022	**0.887**	**0.887**					**0.000**				0.009	**0.380**				0.089	**1.357**	1.737	**2.624**
		+52			0.000											0.000					0.000			
		+10 - 40																			0.007			
-195	-206	+40 - 52		**0.000**		**0.000**	**0.000**					**0.000**					**0.000**				0.001	**0.008**	0.008	**0.008**
		+52																			0.000			
Sub Total		32.5		**0.174**		**2.852**	**3.026**	0.000	0.000	1.693	0.075	1.767	0.000	0.000	0.491	1.222	1.714	0.000	0.000	0.000	1.619	1.619	5.100	**8.126**
GRAND TOTAL							**3.377**	0.000	0.000	1.693	0.075	**1.767**	0.000	0.000	0.491	1.222	**1.714**	0.000	0.000	0.000	1.619	**1.619**	**5.100**	**8.477**

* **Presently Non mineable**

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SUMMARY OF RESOURCES:

The summary of the mineral reserve & remaining resources that can be exploited by both opencast as well as underground means is given below in Table 3.4.2.

Table 3.4.2 (a)
Summary Of Chrome Ore Reserves & Resources As on 01.04.2012 (In Million tonnes)

Ore Body	Method of Mining	Reserves						Remaining Resources					Total Resources	
		Proved		Probable		Total Reserves		Prefeasibility		Inferred	Total Remaining Resources			
		STD 111		STD 122				STD 211 &221	STD 222	STD 333				
		A		B		C=(A+B)		D	E	F	G=(D+E+F)		H= (C+G)	
Northern Band	Open cast	3.654	3.654	1.345	5.256	4.999	8.909	1.592	0.579	0.000	2.171	39.921	7.170	48.830
	Under ground	0.000		3.911		3.911		7.231	18.047	12.472	37.750		41.661	
Middle Band	Open cast	3.326	3.326	2.424	6.160	5.750	9.486	0.323	0.841	0.000	1.163	17.524	6.914	27.010
	Under ground	0.000		3.736		3.736		0.161	13.894	2.305	16.360		20.096	
Southern Band	Open cast	0.351	0.525	0.000	2.852	0.351	3.377	0.000	0.000	0.000	0.000	5.100	0.351	8.477
	Under ground	0.174		2.852		3.026		1.767	1.714	1.619	5.100		8.126	
GRAND TOTAL	Open cast	**7.331**	**7.505**	**3.769**	**14.267**	**11.100**	**21.773**	**1.914**	**1.420**	**0.000**	**3.334**	**62.545**	**14.435**	**84.317**
	Under ground	**0.174**		**10.498**		**10.672**		**9.160**	**33.655**	**16.396**	**59.211**		**69.883**	

The above reserves and resources as per UNFC standards are given in Table 3.4.2 (b) below.

Table No. 3.4.2 (b)
Total Chromite Resources as Per UNFC

	Category/Code	Total	Description	Quantity in Mill.tonnes (+10% Cr_2O_3)
Reserves (a)	Proved **(111)**	7.331	Open Cast	7.331
			Underground	0
	Probable **(121)**	0.174	Open Cast	0
			Underground	0.174
	Probable **(122)**	14.267	Open Cast	3.769
			Underground	10.498
Remaining Resources (b)	Feasibility Resources **(211)**	1.914	Open Cast	1.914
	Pre-feasibility Resources **(221)**	9.160	Underground	9.160
	Pre-feasibility Resources **(222)**	35.075	Open Cast +Underground	35.075
	Measured Resources **(331)**	--		
	Indicated Resources **(332)**	--		
	Inferred Resources **(333)**	16.396	Underground	16.396
Total Resources (a+b)		**84.317**		84.317

Pyroxenite Reserve:

The proved mineral resource of pyroxenite was 37.98 lakh tonnes as on 31.03.08. During the year 2008-09, 2009-10 total 5.20 lakh tonnes of pyroxenite ROM was mined. So the updated pyroxenite reserve as on 01.04.2010 was 32.78 Lakh tonnes.

Based on the geological mapping data & the boreholes drilled for pyroxenite, the Pyroxenite body was remodeled using Surpac software. A solid body model of the Pyroxenite was created. Considering the proximity of the tailing dam the ultimate pit was redesigned upto 73mRL. From the block model, the different UNFC categories of Pyroxenite resource was calculated using a fixed specific gravity of 2.5. The details of Pyroxenite resource is tabulated below in Table 3.4.3:

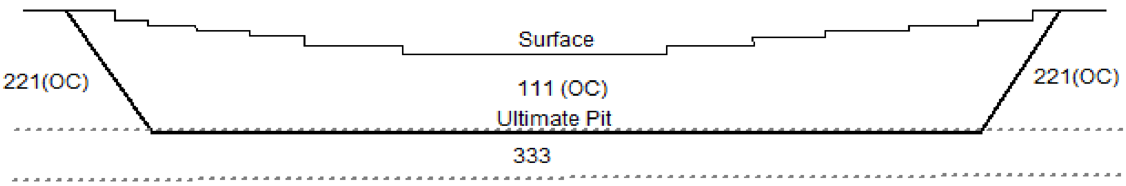


Table 3.4.3
Updated Reserve & Resources for Pyroxenite
(as on 31.03.2012)

Reserves					Remaining Resources					**Total Resources**
Proved		Probable		**Total (Million Tonnes)**	Prefeasibility		Inferred		**Total**	
STD 111		STD 122			STD 221		STD 333			
Million Tonne	Total	Million Tonne	Total		Resource blocked in boundary	**Sub Total**	Inferred Resource	**Sub Total**		
	A		B	**C=(A+B)**	D	**E**	F	**G**	**H = E+G**	**I = C+H**
3.080	**3.080**			3.080	5.74	5.74	3	3	8.74	**11.820**

Other Resources:

The stock of resources estimated within mineralized stacks of LG friable ore, lumpy ore together with mineral rejects in form of fines for lumpy ore, pyroxenite ore, as well as tailings stored in the tailing pond and other areas as on 01.04.2012 is given below in Table 3.4.4.

Table: 3.4.4

Sl No.	Type of Material	Tonnage (Lakh MT)	Location
1	LG Ore (Friable)	3.00	LG Dump (2),(6)
2	LG Ore (Lumpy)	14.00	LG Dump (4),(4A)
3	Mineral Rejects (Lumpy Fines)	1.20	MR (8), MR (Lumpy fines)
4	Mineral Rejects (Pyroxenite Fines)	0.75	MR (Pyrox fines) in 1(A) dump
5	Tailings	75.00	In the Tailing pond
6	Total	93.95	

TATA

4. MINING :

a) EXISTING/ PROPOSED METHOD FOR DEVELOPING/ WORKING THE DEPOSIT :

EXISTING METHOD (OPENCAST MINING):

There are two separate quarries, which have been developed within the leasehold to produce lumpy chromite, friable chromite and pyroxenite ore. These quarries are OB-X & OB-II Quarry.

Out of these, OB II is located in the foot hill of Mahagiri range which has been developed for mining the lumpy ore and OB X for mining friable ore bands, which occurs between small hillocks north of the Mahagiri range. The pyroxenite section falls as a part of the OB II quarry.

Opencast mining is being carried out in all the quarries with shovel-dumper combination. Since the orebody is near vertical, the ore is normally mined from the pit bottom/ lowest floor level along the strike. At times the ore may be encountered in the second lowest bench due to its meandering nature along the strike direction. While the high & medium grade ore is processed & marketed directly, the low & below low grade ores/ beneficiable grade ores are stacked separately for further beneficiation through the Chrome Ore Beneficiation Plant. The major part of the mid band resources is of beneficiable grade and for planning purposes has been considered to serve as a feed to the beneficiation plant. This is evident from the slice plan showing the grade distribution of the resources which has been enclosed as drawing no. SCM/MP/04C/12. The overall average grade of the mineable reserves for the middle band is 37.49% Cr_2O_3. The mixed powdery ore in the contact zones and ore which is diluted due to the flow of slimes/ soil from the top regions of the quarry during the rainy season also are sent to the beneficiation plant. Accordingly the pit-wise & year-wise excavation plans have been developed through an iterative process after confirming the likely grade of ROM to be produced which is derived from the SURPAC Mine Planning software and resource model. However, channel and blast hole sampling and analysis is done at regular interval of time with the deepening of the pit, all along and across the strike of the exposed ore body to assist in guiding the flow of different grades of ore encountered to the different location for further processing.

The salable grade ore from the OB-X quarry having $\geq$42% Cr_2O_3 is either processed at the lumpy ore processing plant or by way of manual dressing and sorting at the stackyard. The powdery and soft form of the ore is sent to the stackyard for manual dressing and sorting. The relatively harder ore along with boulder are further crushed and screened in the LOPP. Although the proportion of friable ore to the LOPP and stackyard for manual dressing and sorting is planned on the basis of past experience, this can always change depending upon the extent of weathering in different regions of the orebody. The hard lumpy and siliceous ore having $\geq$ 30% Cr_2O_3 from the OB-II quarry is fed directly to the LOPP. The ore having < 30% Cr_2O_3 and along the contact zones is stacked at low grade ore dumps. There is little colur difference between the hard lumpy ore and the adjacent serpentinite overburden and sometimes it is extremely difficult to delineate the ore from the waste due to a coating of slimes at the pit bottom. Such low grade ores and mixed ore are further manually dressed and sorted in order to segregate the fractions having better quality. The flow of different grades of ore produced from the OB-X and OB-II quarries have been further elaborated under chapter 10.

The height of the benches varies between 8 m and 10 m and the width varies between 12 m and 16 m. Tata Hitachi hydraulic Excavators (3.5 Cum. bucket capacity) and 35 T BEML Haulpak dumpers are used for removing overburden in the mine. The ore benches are being worked in slices by 0.9-2 cum capacity shovels and transported to the stack yard for dressing, sorting and stacking.

The requirement of drilling and blasting at Sukinda Chromite Mine is mostly confined to the OB II quarry where the overburden consists of hard quartzite and hard Ultrabasic rock (serpentinites) and also for ore raising in both the quarries.

The pits viz. OB-II & OB-X quarries for the lumpy and friable ores respectively are now be designed as per the revised permission received with respect to the overall pit slopes.

Mechanized opencast mining using shovel-dumper combination is being adopted for mining. Before mining the ore, the exposed ore body is demarcated into various marketable grade-zones by drawing channel samples across the strike direction at regular intervals. On this basis, the crude ore is raised or removed in slices of 3 to 4m each. The process of mining involves drilling, blasting, shovelling and trucking and the details regarding the same has been dealt with in subsequent paras of this document.

PROPOSED METHOD (OPENCAST MINING & UNDERGROUND MINING):

Both opencast as well as underground mining shall continue for a period till the opencast mineable ore reserves are exhausted and the underground mining attains its rated capacity. The opencast mining shall continue in the manner described above. The proposed underground mining shall be done as enumerated below.

Methods of Underground Mining:

The selection of the mining method was based on extensive geotechnical studies and numerical modeling done by SRK, South Africa and CIMFR, Nagpur. The recommendations of both the studies were:

- The friable and soft chrome ore with weak soil like hanging and footwalls in the Northern & Middle friable ore bands up to the weathering horizons should not be mined from underground as it will be extremely unsafe;
- Below the weathering zone in the Northern & Middle friable ore bands both Cut and Fill method and Sub-level Open Stoping with post filling is possible depending upon the hardness of the orebody, and; similarly
- Both Sub-Level Open Stoping with post filling and Cut and Fill methods shall be applicable in the Southern lumpy ore band depending upon the extent of disturbance/ fracturing and soundness of the orebody and its wall rocks.

Sub-level Open Stoping-

The stope block shall be mined out from the “bottom-up”, as a panel between sub-levels is mined and filled, the fill shall then used as a platform for the above panel is be mined. Before the panel is mined the in-stope development, on the top and bottom, to the limits of the block to be completed. A raise or slot shall be developed up from the bottom ore drive to the top ore drive. This slot shall provide a free-breaking face for the ring blasting and ventilation.

The sub-level open stoping and its many derivatives, is relatively flexible in that it can accommodate varying widths down to a minimum of 2 m. The maximum mineable width is dependent on the rock mass characteristics of the ore body and the surrounding host rock.

Cut & Fill stoping-

The cut and fill mining method shall be based on longitudinal drifts mined on strike. The drifts shall be 5 m wide x 5 m high and the length dependent of distance apart of the access crosscuts.

Access to the ore body shall be from ramps and the ore shall be mined in horizontal cuts and carried out in a “bottom-up” sequence. The drift shall be filled and mining to take place on top of the fill material. Drift and fill mining shall be generally used in rock conditions that do not allow the opening up of the stope to the full width of the ore body.

The filling of the stopes shall be done with the rejects generated from the underground development drives as well as with the indigenously available overburden and mill tailings as a paste fill. A separate plant shall be set up in the surface to pump the fill material for the stowing of the voids in the underground stopes.

Process flow of Proposed Underground Mining:

Broadly the mining operation can be divided into two categories, i.e. development and stoping. The process flow chart for proposed underground mining is given in Fig. 4.1 below.

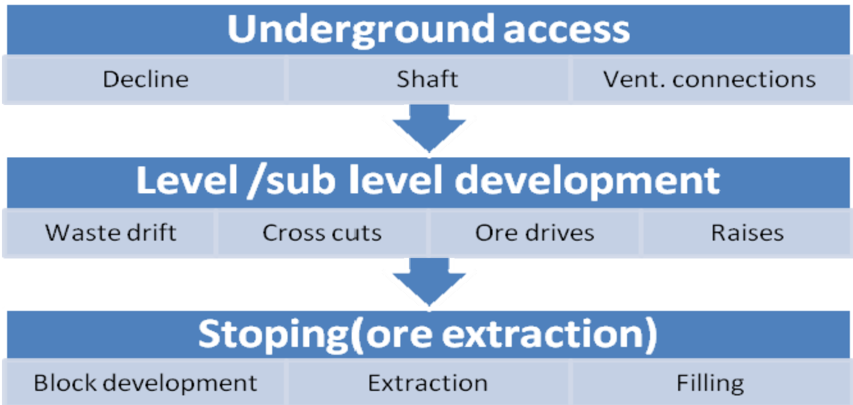


Fig. 4.1: Underground process flow

Development shall essentially consist of the following steps:

1. Construction of the entry headings to the deposit by declines for trackless equipments and shaft from the surface level
2. Construction of the horizontal levels and sublevels as per the design
3. Establishing the ventilation connection to each level and stope blocks through ventilation raises and stope raises.

The development of openings generally follow the following cyclic pattern which shall consists of the following operations like a) Electro hydraulic jumbo drilling b) Charging and blasting c) Roof dressing, supporting and stabilisation d) Removal of the blasted rock by trackless equipments like LHD and LPDTs and preparation for next round of drilling. The process flow chart for the proposed stoping operations is given below in Fig. 4.2:

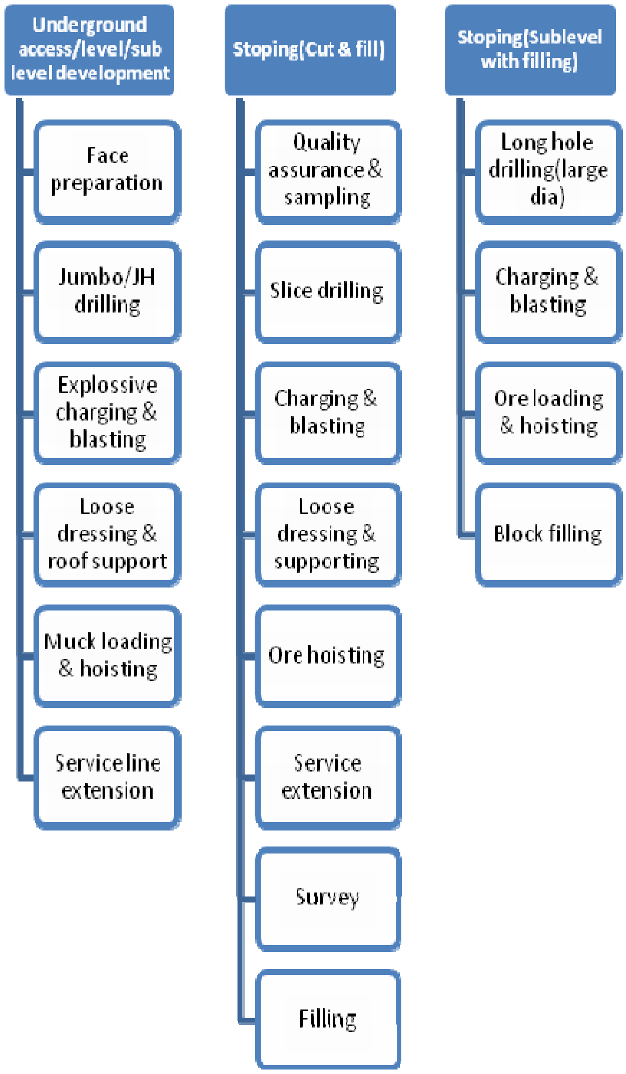


Figure-4.2: Process Flow Chart for Underground Stoping Operations

Sub level Open Stoping-

The ore body below the open pit will be accessed by Surface declines which will join to the underground ramps in each part of the deposit. The ramps and surface declines would be developed in the footwall side of the ore bands from which the drifts (level development in waste) and drives (level development in ore) will be developed. Sub level drifts will be developed at every 20 m interval between the main levels which shall be 110 m apart. Each part of the deposit will be divided to stope blocks of 110 m length along strike and 100 m depth. Each stope block will have 4 sub level drives in which small stope block of 10m x 10m x 20m will be sequentially extracted by cyclic mining operations as below.

1. Drilling of large dia. long holes in each stope block (10m x 10m x 20m) from top /bottom level drive in fan cut

2. Charging and blasting which will start from the slot side
3. Removal of the blasted ore from the draw points and stope with remote controlled LHDs
4. Barricading of the stope block after the clearance for filling.
5. Filling which will complete the block, and then move to the next block as in the diagram given in Plate-3.

Cut & fill stoping-

It is an entry method of mining in which the equipment and persons have to enter inside the stopes to carry out the cyclic mining operations. Cut and fill method has been planned to be executed in the comparatively softer formations than the lumpy chrome ore which is the mixed type material.

The minimum stope development required in this method is the bottom and top level drive and the ventilation connections. The sequence of work will be as below for which cyclic mining operations is to be carried out.

The bottom level ore drive is to be stripped to reach a dimension of 5m x 5m along the strike for the whole stope block length. If the ore body width is wider than 5 m then one or more parallel ore drives in similar manner shall be developed. Every stope drive after stripping (5m x 5m size) is to be filled up to roof to ensure stability before development of the next stope drive.

b) YEAR-WISE DEVELOPMENT & PRODUCTION PLAN:-

I. OPENCAST MINES:

The yearly proposed development and production plans for the next 5 years for OB X and OB II quarries is shown in drawing no.'s SCM/MP/06/12 and drawing no. SCM/MP/07/12.

The year wise, pit wise, quantities of overburden / associated mineral, salable ore, mineral rejects and beneficiable grade ore from opencast mines to be handled is given below in Table 4.2.1& 4.2.2.

Table- 4.2.1
OB-X Quarry (Opencast Mining)

Year	OB (CuM)	ROM Ore (Tonnes)	Saleable Ore (Tonnes) ($Cr_2O_3 \geq 42$ %)	Sub-Grade/ Beneficiable Grade Ore (Tonnes) ($Cr_2O_3 \geq 10$ % < 42 %)	Mineral Fines / Rejects (Tonnes)	Stripping Ratio (OB in CuM / ROM in Tonnes)	Total Excavation (CuM)
2012-13*	3900000	1054083	394434	638889	20760	3.70	4223166
2013-14	1300000	351361	131478	212963	6920	3.70	1407722
2014-15	5400000	1636618	575200	1031145	30274	3.30	5903773
2015-16	5900000	1743152	616432	1094276	32444	3.38	6436360
2016-17	6500000	1740492	590667	1118738	31088	3.73	7036883
2017-18	6500000	1818882	700057	1081980	36845	3.57	7056592
Total	29500000	8344588	3008268	5177991	158331	3.63	32064496

*figures for 2012-13 are from 12.01.2013 till 31.03.2013.

Table – 4.2.2
OB-II Quarry (Opencast Mining)

Year	OB (CuM)	ROM Ore (Tonnes)	Saleable Ore (Tonnes) ($Cr_2O_3 \geq 33\%$)	ROM to Low Grade Dump (Tonnes) ($Cr_2O_3 \leq 30\%$)	Mineral Fines / Rejects (Tonnes)	Stripping Ratio (OB in CuM / ROM in Tonnes)	Total Excavation (CuM)
2012-13*	187500	27188	26622	1750	2544	6.90	195202
2013-14	850000	120059	142800	5000	11506	7.08	884011
2014-15	850000	129788	159047	5000	12479	6.55	886767
2015-16	0	0	0	0	0		0
2016-17	0	0	0	0	0		0
2017-18	0	0	0	0	0		0
Total	1887500	277035	328469	11750	26529	6.81	1965980

*figures for 2012-13 are from 12.01.2013 till 31.03.2013.

The year wise, pit wise and bench level wise, quantities of overburden / associated mineral, salable ore and beneficiable grade ore from opencast mines to be handled to be handled is given below Table 4.2.3.

Table: 4.2.3

2012-13 (11.01.2013 onwards): OB X QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard		Beneficiable Grade		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 42$ %		$Cr_2O_3 \geq 10 < 42$ %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	***2***	***3***	***4***	***5***	***6***	***7***	***8***	***9=5+7***	***10=6+8***	***11=4+9***
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	160	170	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	150	160	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	140	150	0.00	0.00	0.00	0.02	0.05	0.02	0.05	0.02
	130	140	0.00	0.00	0.00	0.01	0.04	0.01	0.04	0.01
	120	130	0.00	0.00	0.00	0.01	0.02	0.01	0.02	0.01
	110	120	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	100	110	0.00	0.00	0.00	0.00	0.01	0.00	0.01	0.00
	90	100	0.00	0.00	0.00	0.01	0.04	0.01	0.04	0.01
	80	90	2.25	0.00	0.00	0.02	0.06	0.02	0.06	2.27
	70	80	2.31	0.01	0.03	0.07	0.22	0.08	0.26	2.39
	60	70	2.53	0.03	0.12	0.18	0.56	0.22	0.68	2.74
	50	60	1.69	0.08	0.28	0.20	0.60	0.27	0.88	1.96
	40	50	2.17	0.08	0.29	0.16	0.50	0.24	0.79	2.41
	30	40	1.12	0.06	0.22	0.01	0.02	0.07	0.24	1.18
	20	30	0.85	0.12	0.44	0.00	0.00	0.12	0.44	0.97
	10	20	0.09	0.00	0.00	0.00	0.00	0.00	0.00	0.09
	Total		13.00	0.38	1.38	0.70	2.13	1.08	3.51	14.08

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.70

2012-13 (12.01.2013 onwards):OB II QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM Processing $Cr_2O_3 \geq 30$ %		ROM to Low gr.dump Cr_2O_3 < 30 %		Total ROM (LCuM)		Total Excavation (in LCuM)
	Bottom RL	Top RL		CuM	LMT	CuM	LMT	CuM	LMT	
1	***2***	***3***	***4***	***5***	***6***	***7***	***8***	***9=5+7***	***10=6+8***	***11=4+9***
1100N to 1400N & 3400E to 4000E	165	175	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	155	165	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	145	155	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	135	145	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	125	135	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	115	125	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	105	115	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	95	105	0.000	0.000	0.001	0.000	0.000	0.000	0.001	0.00
	85	95	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	75	85	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	65	75	0.278	0.011	0.040	0.002	0.006	0.011	0.040	0.29
	55	65	0.314	0.009	0.032	0.001	0.005	0.009	0.032	0.32
	45	55	0.474	0.006	0.021	0.001	0.003	0.006	0.021	0.48
	35	45	0.664	0.047	0.167	0.007	0.025	0.047	0.167	0.71
	25	35	0.065	0.000	0.000	0.000	0.000	0.000	0.000	0.06
	Total		1.79	0.074	0.261	0.011	0.039	0.074	0.261	1.87

Stripping Ratio = Overburden (CuM) / ROM (MT) =6.888

2013-14: OB X QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10 < 42$ %		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	***2***	***3***	***4***	***5***	***6***	***7***	***8***	***9=5+7***	***10=6+8***	***11=4+9***
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	0.53	0.00	0.00	0.00	0.00	0.00	0.00	0.53
	160	170	1.10	0.00	0.00	0.00	0.00	0.00	0.00	1.10
	150	160	1.89	0.00	0.00	0.00	0.00	0.00	0.00	1.89
	140	150	2.29	0.00	0.00	0.08	0.23	0.08	0.23	2.37
	130	140	2.44	0.00	0.00	0.21	0.65	0.21	0.65	2.65
	120	130	3.12	0.00	0.00	0.19	0.58	0.19	0.58	3.31
	110	120	4.64	0.00	0.00	0.11	0.33	0.11	0.33	4.75
	100	110	5.74	0.00	0.00	0.11	0.33	0.11	0.33	5.85
	90	100	3.48	0.00	0.00	0.19	0.59	0.19	0.59	3.67
	80	90	4.73	0.00	0.00	0.29	0.89	0.29	0.90	5.03
	70	80	5.96	0.02	0.07	0.35	1.06	0.37	1.13	6.33
	60	70	4.80	0.07	0.25	0.28	0.84	0.35	1.10	5.15
	50	60	5.21	0.32	1.14	0.60	1.83	0.91	2.97	6.13
	40	50	3.36	0.40	1.45	0.91	2.79	1.31	4.24	4.67
	30	40	2.58	0.35	1.26	0.06	0.19	0.41	1.46	2.99
	20	30	1.13	0.51	1.87	0.00	0.00	0.51	1.87	1.64
	10	20	0.00	0.00	0.01	0.00	0.00	0.00	0.01	0.00
	Total		53.00	1.67	6.05	3.37	10.31	5.04	16.37	58.04

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.24

Prepared By: G K Guin
REGN. NO. RQP/ BBS / 044 / 2003 / A

2013-14: OB II QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM Processing		ROM to Low gr.dump		Total ROM (LCuM)		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 30$ %		Cr_2O_3 < 30 %				
				CuM	LMT	CuM	LMT	CuM	LMT	
1	***2***	***3***	***4***	***5***	***6***	***7***	***8***	***9=5+7***	***10=6+8***	***11=4+9***
1100N to 1400N & 3400E to 4000E	165	175	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	155	165	0.006	0.000	0.000	0.000	0.000	0.000	0.000	0.01
	145	155	0.240	0.000	0.000	0.000	0.000	0.000	0.000	0.24
	135	145	0.554	0.000	0.000	0.000	0.000	0.000	0.000	0.55
	125	135	0.456	0.000	0.000	0.000	0.000	0.000	0.000	0.46
	115	125	0.336	0.008	0.029	0.000	0.002	0.009	0.030	0.34
	105	115	0.354	0.028	0.098	0.002	0.008	0.030	0.106	0.38
	95	105	0.670	0.042	0.147	0.002	0.007	0.044	0.154	0.71
	85	95	1.188	0.048	0.170	0.001	0.002	0.049	0.172	1.24
	75	85	1.327	0.030	0.107	0.000	0.000	0.030	0.107	1.36
	65	75	1.491	0.040	0.142	0.000	0.000	0.040	0.142	1.53
	55	65	0.844	0.063	0.221	0.000	0.001	0.063	0.222	0.91
	45	55	0.758	0.043	0.153	0.001	0.003	0.044	0.156	0.80
	35	45	0.275	0.045	0.157	0.008	0.027	0.052	0.185	0.33
	25	35	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	Total		8.500	0.347	1.225	0.014	0.050	0.361	1.275	8.861

Stripping Ratio = Overburden (CuM) / ROM (MT) = 6.67

OB X QUARRY - 2014-15

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard		Beneficiable Grade		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 42$ %		$Cr_2O_3 \geq 10 < 42$ %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	***2***	***3***	***4***	***5***	***6***	***7***	***8***	***9=5+7***	***10=6+8***	***11=4+9***
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	1.73	0.00	0.00	0.00	0.00	0.00	0.00	1.73
	160	170	1.08	0.00	0.00	0.00	0.00	0.00	0.00	1.08
	150	160	2.18	0.00	0.00	0.00	0.00	0.00	0.00	2.18
	140	150	0.35	0.00	0.00	0.05	0.15	0.05	0.15	0.40
	130	140	-0.83	0.00	0.00	0.28	0.87	0.28	0.87	-0.54
	120	130	1.97	0.00	0.00	0.27	0.82	0.27	0.82	2.23
	110	120	4.39	0.00	0.00	0.20	0.61	0.20	0.61	4.59
	100	110	4.85	0.00	0.00	0.18	0.54	0.18	0.54	5.03
	90	100	7.98	0.00	0.00	0.08	0.26	0.08	0.26	8.06
	80	90	1.66	0.00	0.00	0.04	0.14	0.04	0.14	1.70
	70	80	1.74	0.01	0.04	0.06	0.19	0.07	0.22	1.81
	60	70	5.33	0.12	0.44	0.23	0.71	0.35	1.14	5.68
	50	60	7.72	0.24	0.86	0.09	0.26	0.32	1.12	8.04
	40	50	8.79	0.27	0.99	0.46	1.41	0.73	2.40	9.52
	30	40	4.98	0.30	1.08	1.58	4.82	1.87	5.90	6.86
	20	30	4.21	0.50	1.80	0.06	0.18	0.55	1.98	4.77
	10	20	0.86	0.36	1.29	0.00	0.00	0.36	1.29	1.22
	Total		59.00	1.79	6.49	3.58	10.94	5.36	17.43	64.36

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.38

2014-15: OB II QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM Processing		ROM to Low gr.dump		Total ROM (LCuM)		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 30$ %		$Cr_2O_3 < 30$ %				
				CuM	LMT	CuM	LMT	CuM	LMT	
1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9=5+7**	**10=6+8**	**11=4+9**
1100N to 1400N & 3400E to 4000E	165	175	0.158	0.000	0.000	0.000	0.000	0.000	0.000	0.16
	155	165	0.487	0.000	0.000	0.000	0.000	0.000	0.000	0.49
	145	155	0.563	0.000	0.000	0.000	0.000	0.000	0.000	0.56
	135	145	0.782	0.000	0.000	0.000	0.000	0.000	0.000	0.78
	125	135	0.726	0.000	0.000	0.000	0.000	0.000	0.000	0.73
	115	125	0.593	0.000	0.001	0.000	0.000	0.000	0.001	0.59
	105	115	0.512	0.013	0.047	0.000	0.000	0.013	0.047	0.53
	95	105	0.466	0.037	0.129	0.002	0.008	0.039	0.137	0.50
	85	95	0.508	0.039	0.137	0.002	0.008	0.041	0.145	0.55
	75	85	0.441	0.061	0.216	0.000	0.000	0.061	0.216	0.50
	65	75	0.459	0.101	0.355	0.000	0.000	0.101	0.355	0.56
	55	65	0.513	0.024	0.086	0.000	0.000	0.024	0.086	0.54
	45	55	0.482	0.001	0.003	0.000	0.000	0.001	0.003	0.48
	35	45	0.310	0.016	0.058	0.010	0.034	0.026	0.092	0.34
	25	35	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	Total		7.000	0.292	1.032	0.014	0.050	0.307	1.082	7.307

Stripping Ratio = Overburden (CuM) / ROM (MT) = 6.47

2015-16: OB X QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard		Beneficiable Grade		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		$Cr_2O_3 \geq 42$ %		$Cr_2O_3 \geq 10 < 42$ %				
				LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9=5+7**	**10=6+8**	**11=4+9**
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	1.08	0.00	0.00	0.00	0.00	0.00	0.00	1.08
	160	170	1.42	0.00	0.00	0.00	0.00	0.00	0.00	1.42
	150	160	2.34	0.00	0.00	0.00	0.00	0.00	0.00	2.34
	140	150	3.72	0.00	0.00	0.01	0.04	0.01	0.04	3.74
	130	140	3.78	0.00	0.00	0.20	0.61	0.20	0.61	3.98
	120	130	3.83	0.00	0.00	0.44	1.36	0.44	1.36	4.27
	110	120	5.12	0.00	0.00	0.39	1.20	0.39	1.20	5.52
	100	110	7.01	0.00	0.00	0.29	0.87	0.29	0.87	7.30
	90	100	5.22	0.00	0.00	0.23	0.70	0.23	0.70	5.45
	80	90	7.55	0.00	0.00	0.10	0.29	0.10	0.29	7.64
	70	80	4.35	0.00	0.01	0.03	0.08	0.03	0.09	4.38
	60	70	0.40	0.05	0.17	0.05	0.15	0.09	0.32	0.49
	50	60	4.01	0.04	0.14	0.11	0.33	0.15	0.47	4.15
	40	50	6.19	0.24	0.86	0.16	0.49	0.40	1.35	6.58
	30	40	5.99	0.36	1.30	0.52	1.58	0.87	2.88	6.86
	20	30	1.81	0.28	1.02	1.09	3.34	1.37	4.36	3.19
	10	20	1.18	0.61	2.21	0.00	0.00	0.61	2.21	1.79
	Total		65.00	1.58	5.72	3.61	11.04	5.18	16.76	70.18

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.88

2016-17: OB X QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10 < 42$ %		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9=5+7*	*10=6+8*	*11=4+9*
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	1.51	0.00	0.00	0.00	0.00	0.00	0.00	1.51
	160	170	1.66	0.00	0.00	0.00	0.00	0.00	0.00	1.66
	150	160	1.83	0.00	0.00	0.00	0.00	0.00	0.00	1.83
	140	150	2.13	0.00	0.00	0.00	0.00	0.00	0.00	2.13
	130	140	2.16	0.00	0.00	0.04	0.13	0.04	0.13	2.20
	120	130	2.94	0.00	0.00	0.24	0.73	0.24	0.73	3.18
	110	120	3.62	0.00	0.00	0.56	1.70	0.56	1.70	4.18
	100	110	5.83	0.00	0.00	0.51	1.55	0.51	1.55	6.34
	90	100	7.03	0.00	0.00	0.16	0.49	0.16	0.49	7.19
	80	90	6.69	0.00	0.00	0.17	0.52	0.17	0.52	6.86
	70	80	6.49	0.03	0.10	0.09	0.28	0.12	0.39	6.61
	60	70	7.44	0.19	0.68	0.04	0.11	0.22	0.79	7.66
	50	60	5.41	0.30	1.10	0.23	0.71	0.54	1.81	5.95
	40	50	4.53	0.27	0.99	0.20	0.60	0.47	1.59	5.00
	30	40	3.17	0.42	1.53	0.18	0.55	0.60	2.08	3.77
	20	30	2.54	0.31	1.14	1.11	3.41	1.43	4.55	3.97
	10	20	0.00	0.32	1.17	0.00	0.00	0.32	1.17	0.32
	Total		65.00	1.85	6.72	3.53	10.79	5.38	17.51	70.38

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.71

2017-18: OB X QUARRY

Section	Bench Level		Overburden Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10 < 42$ %		Total ROM		Total Excavation (in LCuM)
	Bottom RL	Top RL		LCuM	LMT	LCuM	LMT	LCuM	LMT	
1	*2*	*3*	*4*	*5*	*6*	*7*	*8*	*9=5+7*	*10=6+8*	*11=4+9*
2000N to 2700N & 1700E to 4200E	230	240	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	220	230	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	210	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	200	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	190	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	180	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	170	180	3.54	0.00	0.00	0.00	0.00	0.00	0.00	3.54
	160	170	3.67	0.00	0.00	0.00	0.00	0.00	0.00	3.67
	150	160	4.46	0.00	0.00	0.00	0.00	0.00	0.00	4.46
	140	150	4.96	0.00	0.00	0.00	0.00	0.00	0.00	4.96
	130	140	4.92	0.00	0.00	0.00	0.00	0.00	0.00	4.92
	120	130	4.97	0.00	0.00	0.00	0.01	0.00	0.01	4.97
	110	120	5.28	0.00	0.00	0.03	0.09	0.03	0.10	5.31
	100	110	5.57	0.04	0.13	0.19	0.59	0.23	0.72	5.79

	90	100	5.21	0.06	0.22	0.37	1.13	0.43	1.35	5.64
	80	90	5.17	0.07	0.27	0.41	1.27	0.49	1.54	5.66
	70	80	4.53	0.06	0.23	0.44	1.35	0.51	1.58	5.03
	60	70	4.10	0.13	0.48	0.44	1.33	0.57	1.81	4.67
	50	60	3.71	0.34	1.22	0.41	1.27	0.75	2.49	4.46
	40	50	2.49	0.42	1.52	0.36	1.10	0.78	2.61	3.26
	30	40	1.23	0.33	1.19	0.01	0.04	0.34	1.22	1.57
	20	30	1.09	0.41	1.50	0.00	0.00	0.41	1.50	1.50
	10	20	0.11	0.41	1.49	0.00	0.00	0.41	1.49	0.52
	Total		65.00	2.27	8.24	2.67	8.18	4.94	16.42	69.94

Stripping Ratio = Overburden (CuM) / ROM (MT) = 3.96

II. UNDERGROUND MINES:

The proposed yearly plans for the next 5 years for the OB X and OB II ore bands/ zones through underground means is shown in drawing no. SCM/MP/17/12 and drawing no. SCM/MP/18/12.

During the initial years, the waste generation shall be restricted to the development of declines and shafts only. Incidental production from the OB-X mine would start from 2014-15, and in OB-II mine it would be during 2013-14. OB-II mine is planned to reach close to its rated capacity by year 2015-16. The year wise, pit wise, quantities of waste/ associated mineral, salable ore, mineral rejects and beneficiable grade ore from underground mines to be handled is given below in Table 4.2.3& 4.2.4.

Table:4.2.3
OB-X mine (Underground Mining)

Year	Waste (CuM)	ROM Ore (Tonnes)	Saleable Ore (Tonnes) ($Cr_2O_3 \geq 42$ %)	Sub-Grade/ Beneficiable Grade Ore (Tonnes) ($Cr_2O_3 \geq 10$ % < 42 %)	Mineral Fines / Rejects (Tonnes)	Total Excavation (CuM)
2012-13*	0	0	0	0	0	0
2013-14	12165	0	0	0	0	12165
2014-15	48617	0	0	0	0	48617
2015-16	136140	35531	14565	20199	767	146731
2016-17	193637	166488	104055	56957	5477	241834
2017-18	267433	535200	224527	298856	11817	429080
Total	657992	737219	343147	376012	18061	878427

*figures for 2012-13 are from 12.01.2013 till 31.03.2013.

Table: 4.2.4
OB-II mine (Underground Mining)

Year	Waste (CuM)	ROM Ore (Tonnes)	Saleable Ore (Tonnes) ($Cr_2O_3 \geq 33\%$)	Sub-Grade to Low Grade Dump (Tonnes) ($Cr_2O_3 \leq 30\%$)	Mineral Fines / Rejects (Tonnes)	Total Excavation (CuM)
2012-13*	2500	0	0	0	0	2500
2013-14	12984	0	0	0	0	12984
2014-15	44653	18704	22921	0	1870	49952
2015-16	91494	70056	109348	12000	5806	111340
2016-17	95546	272106	281090	12000	26011	172630
2017-18	103929	309176	330400	4000	30518	191514
Total	351106	670042	743759	28000	64205	540920

*figures for 2012-13 are from 12.01.2013 till 31.03.2013.

c) YEARWISE DEVELOPMENT:

Both OB X & OB II areas shall be operated by opencast and underground means for few years till the opencast mineable reserve are exhausted and the underground rated production capacity is achieved. While the opencast mining would be going on, the underground development job shall be pursued simultaneously. During the initial years of underground development the declines, shafts shall be driven to reach the ore body. The year-wise generation of overburden/ waste and ROM at both OB-X and OB-II quarry by both opencast and underground means is given in Table 4.3.3 & 4.3.4. The year wise and bench-level wise ROM ore to be mined from the various quarries is given below in Table 4.3.1 and 4.3.2.

Table: 4.3.1
Year wise, Level wise ROM Production
OB-X Quarry

Bench Level		**Year wise ROM Production (Lakh MT)**					
Bottom RL	**Top RL**	**2012-13**	**2013-14**	**2014-15**	**2015-16**	**2016-17**	**2017-18**
1	***2***	***4***	***5***	***6***	***7***	***8***	***9***
140	150	0.05	0.23	0.15	0.04	0.00	0.00
130	140	0.04	0.65	0.87	0.61	0.13	0.00
120	130	0.02	0.58	0.82	1.36	0.73	0.01
110	120	0.00	0.33	0.61	1.20	1.70	0.10
100	110	0.01	0.33	0.54	0.87	1.55	0.72
90	100	0.04	0.59	0.26	0.70	0.49	1.35
80	90	0.06	0.90	0.14	0.29	0.52	1.54
70	80	0.26	1.13	0.22	0.09	0.39	1.58
60	70	0.68	1.10	1.14	0.32	0.79	1.81
50	60	0.88	2.97	1.12	0.47	1.81	2.49
40	50	0.79	4.24	2.40	1.35	1.59	2.61
30	40	0.24	1.46	5.90	2.88	2.08	1.22
20	30	0.44	1.87	1.98	4.36	4.55	1.50
10	20	0.00	0.01	1.29	2.21	1.17	1.49
Total		**3.51**	**16.37**	**17.43**	**16.76**	**17.51**	**16.42**

Table: 4.3.2
Year wise, Level wise ROM Production
OB-II Quarry

Bench Level		ROM Quantity (Lakh MT)		
Bottom RL	**Top RL**	**2012-13***	**2013-14**	**2014-15**
1	***2***	***4***	***5***	***6***
165	175	0.000	0.000	0.000
155	165	0.000	0.000	0.000
145	155	0.000	0.000	0.000
135	145	0.000	0.000	0.000
125	135	0.000	0.000	0.000
115	125	0.000	0.030	0.001
105	115	0.000	0.106	0.047
95	105	0.001	0.154	0.137
85	95	0.000	0.172	0.145
75	85	0.000	0.107	0.216
65	75	0.040	0.142	0.355
55	65	0.032	0.222	0.086
45	55	0.021	0.156	0.003
35	45	0.167	0.185	0.092
25	35	0.000	0.000	0.000
Total		0.261	1.275	1.082

* ROM Production figures of OB-X & OB-X quarry for 2012-13 is from 11.01.2013 till 31.03.2013

Table 4.3.3

Yearly Development Plan: OB-X Quarry

YEAR	Opencast/ Underground	Overburden Volume (L CuM)	ROM to Stackyard $Cr_2O_3 \geq 42$ %		Beneficiable Grade $Cr_2O_3 \geq 10$ % < 42 %		Total ROM (LCuM)		Total Excavation (in LCuM)
			L CuM	LMT	L CuM	LMT	L CuM	LMT	
2012-13*	Opencast	13.00	0.38	1.38	0.68	2.13	1.06	3.51	14.08
	Underground	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Total	13.00	0.38	1.38	0.68	2.13	1.06	3.51	14.08
2013-14	Opencast	54.00	1.67	6.05	3.33	10.31	5.00	16.37	59.04
	Underground	0.12	0.00	0.00	0.00	0.00	0.00	0.00	0.12
	Total	54.12	1.67	6.05	3.33	10.31	5.00	16.37	59.16
2014-15	Opencast	59.00	1.79	6.49	3.53	10.94	5.32	17.43	64.36
	Underground	0.49	0.00	0.00	0.00	0.00	0.00	0.00	0.49
	Total	59.49	1.79	6.49	3.53	10.94	5.32	17.43	64.85
2015-16	Opencast	65.00	1.71	6.22	3.56	11.19	5.27	17.40	70.37
	Underground	1.36	0.04	0.15	0.06	0.20	0.11	0.36	1.47
	Total	66.36	1.76	6.37	3.63	11.39	5.38	17.76	71.84
2016-17	Opencast	65.00	2.03	7.37	3.50	10.82	5.53	18.19	70.57
	Underground	1.94	0.30	1.10	0.18	0.57	0.48	1.66	2.42
	Total	66.94	2.33	8.46	3.68	11.39	6.01	19.85	72.98
2017-18	Opencast	65.00	1.98	7.20	2.71	8.40	4.70	15.61	69.73
	Underground	2.67	0.65	2.36	0.97	2.99	1.62	5.35	4.29
	Total	67.67	2.64	9.57	3.68	11.39	6.31	20.96	74.02
Total	Opencast	321.00	9.56	34.72	17.30	53.79	26.87	88.51	348.14
	Underground	6.58	1.00	3.61	1.21	3.76	2.20	7.37	8.78
	Total	327.58	10.56	38.33	18.51	57.55	29.07	95.88	356.93

*figure for 2012-13 is from 12.01.2013 till 31.03.2013

Table 4.3.4

Yearly Development Plan: OB-II Quarry

YEAR	Opencast/ Underground	Overburden Volume	ROM Processing		ROM to Low Grade dump		Total ROM (LCuM)		Total Excavation (in LCuM)
			$Cr_2O_3 \geq 30$ %		$Cr_2O_3 < 30$ %				
		(L CuM)	L CuM	LMT	L CuM	LMT	L CuM	LMT	
2012-13*	Opencast	1.88	0.072	0.254	0.005	0.018	0.077	0.272	1.95
	Underground	0.03	0.000	0.000	0.000	0.000	0.000	0.000	0.03
	Total	1.90	0.072	0.254	0.005	0.018	0.077	0.272	1.98
2013-14	Opencast	8.50	0.326	1.151	0.014	0.050	0.340	1.201	8.84
	Underground	0.13	0.000	0.000	0.000	0.000	0.000	0.000	0.13
	Total	8.63	0.326	1.151	0.014	0.050	0.340	1.201	8.97
2014-15	Opencast	8.50	0.354	1.248	0.014	0.050	0.368	1.298	8.87
	Underground	0.45	0.053	0.187	0.000	0.000	0.053	0.187	0.50
	Total	8.95	0.406	1.435	0.014	0.050	0.421	1.485	9.37
2015-16	Opencast	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	Underground	0.91	0.164	0.581	0.034	0.120	0.164	0.581	1.11
	Total	0.91	0.164	0.581	0.034	0.120	0.164	0.581	1.11
2016-17	Opencast	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	Underground	0.96	0.737	2.601	0.034	0.120	0.737	2.601	1.73
	Total	0.96	0.737	2.601	0.034	0.120	0.737	2.601	1.73
2017-18	Opencast	0.00	0.000	0.000	0.000	0.000	0.000	0.000	0.00
	Underground	1.04	0.865	3.052	0.011	0.040	0.865	3.052	1.92
	Total	1.04	0.865	3.052	0.011	0.040	0.865	3.052	1.92
Total	Opencast	18.88	0.752	2.653	0.033	0.118	0.785	2.770	19.66
	Underground	3.51	1.819	6.420	0.079	0.280	1.819	6.420	5.41
	Total	22.39	2.570	9.073	0.113	0.398	2.604	9.191	25.07

*figure for 2012-13 is from 12.01.2013 till 31.03.2013

d) COMPOSITE YEAR WISE PLAN & SECTION :

i). EXCAVATION

Year wise plan and sections of both OB-X & OB-II quarry by opencast and underground means have been prepared and coloured distinctly in drawing no. SCM/MP/06/12 & SCM/MP/07/12 & to represent the year wise excavation limits.

ii). DUMPING

Waste generated during the planned period of five (5) years will be dumped in the non-mineralized area in additional area of 100ha and shall be backfilled in OBII quarry subsequently after exhaustion of space in additional area. The manner in which the year wise dumping shall be carried out is given in drawing no. SCM/MP/08/12.

e) PROPOSED RATE OF PRODUCTION AND LIFE OF THE MINE :

The year-wise production proposal for chromite ore and pyroxenite ore ROM & Finished ore at Sukinda Chromite Mine from both opencast and underground mining during the planned period is given in the Table 4.5.1 & 4.5.2 below.

Table 4.5.1
Production of ROM in MT

YEAR	OPENCAST			UNDERGROUND		TOTAL		
	OB X	OB II	PYROX	OB X	OB II	OB X	OB II	PYROX
2012-13*	351361	27188	0	0	0	351361	27188	0
2013-14	1636618	120059	0	0	0	1636618	120059	0
2014-15	1743152	129788	0	0	18704	1743152	148492	0
2015-16	1740492	0	0	35531	70056	1776023	70056	0
2016-17	1818882	0	250000	166488	272106	1985370	272106	250000
2017-18	1560537	0	500000	535200	309176	2095737	309176	500000
TOTAL	8851042	277034	750000	737219	670042	9588261	947076	750000

*figures for 2012-13 are from 12.01.2013 till 31.03.2013.

Table 4.5.2
Production of Marketable/ Finished Ore in MT

YEAR	OPENCAST			UNDERGROUND			TOTAL		
	OB X	OB II	PYROX	OB X	OB II	PYROX	OB X	OB II	PYROX
2012-13*	131478	26622	0	0	0	0	131478	26622	0
2013-14	575200	142800	0	0	0	0	575200	142800	0
2014-15	616432	159047	0	0	22921	0	616432	181968	0
2015-16	590667	0	0	14565	109348	0	605232	109348	0
2016-17	700057	0	250000	104055	281090	0	804112	281090	250000
2017-18	684433	0	500000	224527	330400	0	908960	330400	500000
TOTAL	3298267	328469	750000	343147	743758	0	3641414	1072227	750000

*figures for 2012-13 are from 11.01.2013 till 31.03.2013.

Yearly maximum rate of production of Chrome Ore ROM will be around 2.4 Million Tonnes (from Table no. 4.5.1) as per the production Chromite ore during planned period. By considering the existing geological resource, the life of the mine will be 20 years (i.e. till 2033)

f) CONCEPTUAL MINING :

Present Method of Mining and allied activities:

Mining:

Presently conventional opencast method of mining with shovel-dumper combination is being practiced. As there is no further space for disposal of the overburden/ associated minerals (consisting of nickelliferous limonite, serpentinite and quartzite) within the mining lease of 406 ha, the same is being stacked within the additional area of 100 ha that has been allotted for the purpose (Ref. para 2.4) since early 2006. As per conditions stipulated in the approval for its forest clearance, this additional area was released in phases of 25 ha each and so far 75 ha has already been released and almost fully saturated. We have applied for the balance 25 ha to the concerned authority and the same is expected to be obtained shortly.

The rate of production from the respective pits is decided on the basis of the available grades of ore (physical & chemical characteristics) and its end use requirements and plant capacities.

Deep hole drilling and controlled blasting techniques like presplit blasting, use of Nonel, SME (Site Mixed Emulsion) is being practiced to minimize vibration, back breaks and fly rock generation.

Based on geotechnical studies on slope stability conducted by CMRI, Dhanbad and SRK, South Africa approval was obtained from DGMS vide letter no. BCU/CH-7/P-106/2005, dated 03.07.2006 (enclosed as Annexure-18) for a permitted maximum overall pit slope angle of 35 degree in OB-X quarry and 45 degree in OB-II quarry. While the recommendations of CMRI, Dhanbad was same as the above permitted overall slope angles, the recommendations of SRK after a more detailed slope stability analysis was 5 degrees higher than the above permitted overall slope angles. Copy of the relevant portions of the slope stability analysis report of SRK, South Africa is enclosed as Annexures-19. The relevant portions of the CMRI report of March 2000 & SRK report of November 2004 for recommended pit slope dimensions is enclosed as Annexure-20 & 19 respectively. Accordingly, the ultimate pit depth had been conceptualized as (–) 25 mRL in OB-X quarry and 25 mRL in OB-II quarry as per approved Scheme of Mining.

Regular slope monitoring through precise level surveys is being carried out both departmentally as well as with the assistance of CIMFR, Dhanbad to ensure the safety of the opencast workings. Horizontal holes (length- 50 m) are being drilled at intervals along the entire strike length of the lower benches to dewater and depressurize the slopes and improving its stability condition. The depth of the water table is also being monitored with the aid of piezometers that have been installed for the purpose. High capacity electric pumps that have been mounted on pontoons have been provided to effectively dewater the pits. The present extents of the quarries are given in Table No. 4.6.1 below.

The stripping ratios as per the approved Scheme of Mining for the period 2008-13 was planned to be:

Friable Ore band: 5.12
Lumpy Ore band: 7.53

Table No. 4.1.1
Extent of Present Workings

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL				No. of benches								Overall slope angle, degrees							
			North side		South side		Northern Band		Middle Band		Northern Band				Middle Band				Northern Band				Middle Band			
											North side		South side		North side		South side		North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-X	2503	572	124.3	139	125	204	20	52	52	85	13	15	6	9	5	7	5	11	23	33	18	28	15	21	16	24

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL		No. of benches				Overall slope angle, degrees			
			North side		South side		Min.	Max.	North side		South side		North side		South side	
			Min.	Max.	Min.	Max.			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-II	1151	449	118	132	152	230	33	45	12	15	16	21	22	33	24	32
Pyroxenite	329	207	136	146	123	132	102	107	5	7	5	7	17	22	15	21

However as mentioned in the Review chapter, the above planned stripping ratios could not be partially achieved. This was mainly due to higher operational losses. The availability of the dumpers was lower than the plan and was adversely impacted due to the increasing depth, gradients and lead distance of the mine. But more importantly, the operational efficiency and cycle time was affected as a result of cueing of the dumpers due to the restricted bench widths, space available in the dumping grounds and slower speeds of the dumpers due to traffic congestion. Such constraints in the space for overburden dumping arose as the area was released in a phased manner of 25 ha only as per the condition no. 11 stipulated in the approval letter for forest clearance (Ref. to Annexure-21). This also led to difficulties in stacking of the different types of associated minerals separately. Although the company had time and again requested the MoEF siting the above constraints for the modification of such condition, no clear order was passed. However, despite the same the overall slope angles of the pits as evident from the supporting transverse sections of the pits enclosed and enumerated above under Table No. 4.6.1, are still well within the maximum permitted slope angles approved by the DGMS. The safety of the mine and operations have therefore not been jeopardized.

Mineral Beneficiation:

Company has installed a chrome ore beneficiation plant since 1991 which is based on gravity separation principle. The various processes in beneficiation process involve feed preparation by crushing in jaw crusher and cone crusher, grinding in ball mill, removal of gangue using scrubber. In the beneficiation circuit hydrocyclones, screw classifiers, floatex density separators to recover the finer concentrate and to segregate the product based on size. The process water is recycled back into the process after treatment in thickener. The underflow of thickener is discharged into the tailing pond; clean water from the same is recycled back and reused in the COB plant.

Similarly ROM from OB-II quarry is processed at LOPP and N Kumar Plant and +10mm marketable grade material is shifted to stackyard for onward dispatch to various customers through road. The -10mm fines material is stacked separately for future use.

Environment Management:

Presently ambient air, noise and water quality is being monitored twice a week at six locations within the core zone and ambient air quality monitoring is done once in a quarter in 10 villages in buffer zone. Water quality within the core zone is monitored at ETP, STP, WTP, tube wells and oil & grease separation pit.

Plantation is being taken up in dumps progressively after it reaches its ultimate capacity as per the method stated in the approved scheme of mining. Adequate measures are being taken towards post plantation care, watering, watch and ward. Many a places the dump slopes are being coir matted using biodegradable geotextiles to prevent erosion and formation of rain cuts and gullies.

Conceptual Exploration:

As may be observed from the updated surface geological plan, all the ore bands have been explored by drilling mostly inclined boreholes to cover the total strike length from extremities in the west to the east of the lease hold. As is evident from the updated longitudinal geological section, the depth of proving for the opencast mining upto the ultimate pits limits has been more than adequate with a closely spaced grid interval of 50- 100m. Most of the mineable reserves for opencast workings are in the proved category in all the three ore bodies. In view of the dwindling resources of chrome ore for opencast mining, the company has since long taken up further core drilling program to assess the resources at greater depths. Feasibility studies for underground mining have been completed based on the resources so established.

So far till 01.04.2012 the deepest intersections in the northern, middle and southern ore bands have been (-) 375, (-) 235 & (-) 175 mRL respectively. The average depth up to which the resources have already been explored and brought to the indicated (G2) category has been (-) 150, (-) 190 & (-) 125 mRL respectively.

The future exploration has been conceptualized to further prove the depth persistency of all the three ore bodies and progressively bring the inferred or (G3) resources to the (G2) category by way of drilling more number of boreholes from the surface. The remaining gaps in exploration of the ore bodies have been identified. Accordingly the future proposals of drilling deeper boreholes from the surface to fill these gaps and also further explore the depth persistency during the next 5 years have been proposed, to be in consonance with MOM directives for conditions under 27(3) of MCR, 1960. Considering the near vertical nature of the ore bodies, drilling to still greater depths from the surface would require the drilling to be done at fairly steep angles due to lack of space laterally. Moreover, there is also a likelihood of missing the ore body while attempting to drill very long holes at steep angles. Accordingly, the maximum depths of such surface boreholes have been conceptualized. The deepest intersections proposed for the northern, middle and southern ore bands are (-) 450, (-) 450 & (-) 400 mRL respectively. Further, in order to bring the indicated (G2) category of resources at the deeper horizons to the measured (G1) category of resources, the exploratory drilling at lateral intervals of 50-100m and 30-50m vertical depth has be proposed to be undertaken from the proposed underground haulage drives/ drifts. The targeted intersection points for the proposed boreholes have been indicated in the longitudinal geological section for all three orebodies.

Besides the above, boreholes for infrastructure development of the major underground infrastructures like declines and vertical shafts have also been planned to be done at the earliest.

Conceptual Method of Mining and allied activities:

Opencast Mining:

While the general method of opencast mining shall remain the same, a change in the design parameters of the benches, ultimate pit bottom limits and conceptual plan as per this modification to the Scheme of Mining & subsequent Mining Plan is enumerated below.

Conceptual mining plans have been prepared to cover the life of the mine as per the present scenario and have been shown in Drawing no.’s SCM/MP/05/12. The conceptual mining plan is developed considering the long-term future requirements of ore for the end users and the method of Mining to be followed. Considering the reserves, it is envisaged that this requirement shall be fully met from the existing opencast operations. The open pits are designed from the bottom upwards to arrive at the pit limits and accordingly year-wise excavation plans are developed.

It may be observed that the design of pit slopes for OB-II quarry as recommended by CMRI, Dhanbad and SRK, South Africa was meant to have a lower bench width than the bench height in order to achieve the recommended ultimate pit slope of 45^0 (enclosed as Annexures 19 & 20). However, the DGMS permission letter for relaxation under Reg. 106 2(b) under MMR, 1961 (enclosed as Annexure-18) stipulates that the overall pit slope shall not exceed 45^0 and the bench width shall not be less than the bench height. In accordance with the said order of the Director of Mines Safety, the ultimate pit for OB-II quarry has been redesigned. The bench widths for the future mining to be undertaken have been now considered to be same as the bench height. The conceptual ultimate pit design has been shown in the Conceptual Plan and Conceptual Transverse Section (Drawing No.'s SCM/MP/05/12 & SCM/MP/05A /12). With this the overall pit slope angle of the ultimate pit shall be less than 45^0. With this modification, the ultimate pit bottom depth of OB-II quarry (after providing for a sump for dewatering purposes) is 30 mRL and no longer 25 mRL as proposed earlier in the approved Scheme of Mining.

Standalone, the opencast mining limits is typically constrained by the following factors:

a) The available reserves and resources,
b) The lease boundary limits for the lateral extension of the pit top,
c) The ultimate safe & permitted maximum overall pit slope angles for the depth extension of the pits
d) The stripping ratios and the economic viability with respect to the lead distances and depth extension of the pits
e) The expected rate of generation of overburden and the holding capacity of the area available for its disposal.
f) The balance areas that shall be available within the lease after the future extension of the pits for providing for the other allied activities viz. mineral and mineral rejects storage, future infrastructures etc.

The major part of the overburden to be generated in the course of future mining of the chrome ore by opencast method is going to be from the OB-X quarry. Further, considering the dwindling reserves of the low/ beneficiable grade ores which is mostly available from the middle friable ore band, it was felt necessary to rehandle a part of the overburden dump (Dump No. 2A) lying in the south eastern part of the OB-X quarry in order to increase the ultimate pit depths of the mid band area.

The balance holding capacity within the 100 ha allotted for disposal of the overburden as on 01.04.2011 is only 177 lakh cum. The estimates for further overburden generation from the opencast workings are given in the Table no. 4.6.2 below.

Table No. 4.6.2
Estimates for further overburden generation upto ultimate pit bottom limits
(as on 01.04.2011)

Quarry/ Section	Ultimate Pit bottom Depth, mRL	Volume of OB, lakh cum
OB-II	30	24
OB-X	(-)25	337
OB-X (addl. from deepened mid band section)	15	197
Rehandling of OB from the SE old OB dump		26
Total		584

It is evident from the above that the constraints in the area for overburden disposal and the area required for other allied activities and future infrastructure development emerges as the predominant factor for deciding the ultimate pit bottom limits.

Thus, considering the dwindling resources of OB-II quarry it has been first conceptualized to exhaust the mineable reserves and then resort to backfilling of the OB-II pit with the overburden to be generated from the OB-X quarry. Thereafter it is proposed to exhaust the middle ore band section of the OB-X quarry to backfill the same with the remaining overburden to be generated from the northern band of the OB-X quarry.

The holding capacity of the OB-II quarry area and mid band section of the OB-X quarry by way of backfilling has been estimated as given in Table 4.6.3 below.

Table 4.6.3
Estimates for holding capacity for overburden by backfilling of the pits

Quarry	Backfill holding capacity, lakh cum
OB-II	272
OB-X (Mid band section upto 70 mRL)	60
Total	332

The net holding capacity for the overburden including the remaining area within the allotted 100 ha for overburden dumping and the backfilling opportunities within the exhausted pits is ~ 509 lakh cum.

It is evident from the above, that the volume of the overburden to be generated (584 lakh cum) far exceeds the holding capacities of the remaining area of 100 ha and the pits/ sections proposed to be backfilled.

In view of the above constraints, it is now conceptualized as per this modification to the Scheme of Mining and Mining Plan to optimize the future overburden generation from the north band section of the OB-X quarry in order to accommodate the same by way of backfilling within the OB-II and still later in the mid band sections of the OB-X quarry.

Accordingly, the ultimate pit for OB-X quarry has been re-designed and the proposed ultimate pit bottom of the north band section of the OB-X quarry is now 10 mRL (after providing for the sump for dewatering purposes). This would also allow for some space that would be required in future for accommodating the waste from the future underground development work. With this change the revised estimates for OB generation shall be

Quarry/ Section	Ultimate Pit bottom Depth, mRL	Volume of OB, lakh cum
OB-II	30	24
OB-X	10	209
OB-X (addl. from deepened mid band section)	15	197
Rehandling of OB from the SE old OB dump		26
Total		456

The manner in which the overburden dumping shall be done during the balance Scheme of Mining period is shown in Drawing No.'s SCM/MP/08/12.

The manner in which the backfilling of the pits shall be done up to the end of the mining plan period of 5 years after this scheme period and thereafter at intervals of every 5 years till the end of life of the open pit operations have been indicated in the drawings for Conceptual Plan & Conceptual Section (Drawing no.s SCM/MP/05/12 & SCM/MP/05A /12)

No backfilling is however being contemplated in the northern band section of the OB-X quarry at this stage. This is with the view of allowing for further studies to be undertaken for establishment of any novel methods of mining the remaining resources of the higher grades of soft friable chrome blocked between the presently defined ultimate pit limits and the crown pillar proposed to be left in the hard & competent ore for future underground mining as shown in the longitudinal geological section.

As there is no change in the prevailing hydro geological condition and monsoon pattern no change is being proposed in the dewatering mechanism of the quarry.

The chrome ore deposit is dipping almost vertically. The grade of chrome ore changes both in the strike and dip direction as well as with depth. In order to cater to the stringent chemical specifications of the user industries, it is necessary to mine all along the entire strike direction to meet the grade specifications for its end use. The specifications of the user industries are thereby met by suitable blending.

Stripping Ratio: Consequent to the changes in the ultimate pit designs as well as updating of the geological sections as on 01.04.2012 as stated above, a re-assessment of the reserves & resources for opencast mining as per the guidelines of the UNFC was done. Similarly, based on the fresh estimates of the overburden to be generated for future opencast mining upto the proposed ultimate pit limits and rehandling of the old dump no. 2A, the revised stripping ratio for the quarries as on 01.04.2012 are given below:

OB-X quarry: 3.78

OB-II quarry: 6.88

From the above it is evident that the overall stripping ratio of 4.44 at OB-X quarry achieved during the period 2008-11 (as mentioned in the Review chapter), was thus more than adequate for its development. The year-wise mine development plan from 2011-12 onwards has accordingly been conceptualized and modified with the above stripping ratio of 3.78 for OB-X quarry.

The estimates for the stripping ratio of OB-II quarry has come down from 7.53 to a level of 6.88 as per this modification. The overall stripping ratio achieved during the period 2008-11 was 5.85 which were marginally lower than the planned overall stripping ratio of 6.32 during the above period. The year-wise mine development plans from 2011-12 onwards has accordingly been conceptualized and modified with the above stripping ratio of 6.88 for OB-II quarry.

The backfilling of the OB-II quarry has been conceptualized to begin from 2014-15 after the external dumping area of 100 ha is saturated/ full. This backfilling during 2014-15 of the Mining Plan period shall be done between the sections 3100E to 3600E as shown in the Drawing No. SCM/MP/07/12. The OB-II quarry at its finishing stages of mining has also been proposed to be carried out during this year. The mining has been conceptualized and accordingly planned to be restricted between the sections 3650E to 4000E and thus synchronized with the backfilling to be undertaken simultaneously during this year. A safety margin of 50m has been left between the proposed backfill and mining areas. Further care shall be taken to do the backfilling from the bottom upwards in a retreating manner.

Though the mining operation shall not attain its ultimate pit depth during the plan period till 2013, the ultimate pit dimension as envisaged at the end of opencast mining is given below in Table 4.6.4.1 & 4.6.4.2

Table 4.6.4.1

Ultimate Pit Dimension of OB-X Quarry

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL			
			North side		South side		Northern Band		Middle Band	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-X	2503	615	114	137	120	225	10	30	15	40
			No. of benches							
			Northern Band				Middle Band			
			North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
			14	16	6	9	5	7	5	21
			Overall slope angle, degrees							
			Northern Band				Middle Band			
			North side		South side		North side		South side	
			Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
			30	35	34	35	27	35	35	35

Table 4.6.4.2

Ultimate Pit Dimension of OB-II Quarry & Pyroxenite Section

Quarry	Avg. Length (m)	Avg. Breadth (m)	Pit top, mRL				Pit bottom, mRL	
			North side		South side		Min.	Max.
			Min.	Max.	Min.	Max.		
OB-II	1151	449	118	132	152	230	30	35
Pyroxenite	467	224	136	145	123	129	73	73
Quarry	No. of benches		Overall slope angle, degrees					
	North side		South side		North side		South side	
	Min.	Max.	Min.	Max.	Min.	Max.	Min.	Max.
OB-II	12	15	16	21	30	33	35	40
Pyroxenite	10	11	8	9	35	36	35	36

However, in order to meet the chrome ore requirement for end use in a sustained manner keeping in view the proposed expansion, necessary infrastructure development for underground mining is required to be started at the earliest.

However, in order to meet the chrome ore requirement for end use in a sustained manner keeping in view the proposed expansion, necessary infrastructure development for underground mining is required to be started at the earliest.

Life of the Deposit:

While open cast mining can continue for another 10 years at the present level of production, certain pits like the OB-II lumpy ore quarry and the mid band section of OB-X quarry which is producing beneficiable grade chrome ore shall be depleted much earlier. This is therefore going to be a brown field project wherein the mine is proposed to be operated both by opencast & underground means in the near future and then convert to a totally underground operation in the long term.

Extending the life of mine & Underground Mining:

From inception of mining operation at Sukinda Chromite Mine, both the OB-X and OB-II quarry were planned to operate at overall pit slope of 25^0. Slope stability study was undertaken through SRK, South Africa and CMRI, Dhanbad. Based on their recommendations, various measures were implemented in mines and permission for higher slope angle of 30^0 in OB-X and 40^0 in OB-II was obtained from DGMS. After successful implementation of the recommendation, the stability of pit slope was again reassessed by CMRI, Dhanbad and subsequently permission for higher pit slope angle of 35^0 for OB-X and 45^0 for OB-II was obtained from DGMS. Due to the above increase in overall pit slope angle, mining can now be carried out at greater depth with similar lease boundary restrictions as earlier which in turn has increased the life of mine.

Feasibility study for underground mining was conducted by SRK, South Africa. This study includes structural mapping and modeling, geotechnical assessment, hydrogeological study, insitu stress measurement, numerical modeling for the design of mine entries and its layout and design, stoping method with support system, hoisting/ haulage, ventilation assessment, drainage etc. and techno-economic assessment. The relevant portions of the above reports are enclosed as Annexure-22.

An independent assessment was also conducted by CIMFR, Nagpur for the geotechnical design of underground stoping methods as well as for subsidence prediction. Numerical modeling has been done for all such assessments. The relevant portions of the above reports are enclosed as Annexure-36.

Both these studies advocate the feasibility of underground mining in all three ore bodies. The recommendations of both these studies are more or less on the same lines and were found to complement each other. Accordingly, the design parameters for the underground mine, crown & sill pillars, stope design have been conceptualized considering both these independent studies.

The long-term ROM production requirement from the mine has been conceptualized to be around 2.4 MTPA of chrome ore. While this ROM production level is proposed to be achieved by the end of the Mining Plan period (i.e. 2017-18) and met from both opencast and underground workings put together, the underground mine is proposed to have an ultimate rated capacity of ~ 2.4 MTPA. In view of the same, the underground mine is proposed to be a fully mechanized mine with trackless method of mining. The underground loading and hauling equipment shall be diesel operated.

The long-term production requirement and conceptualized production build up from the opencast and underground workings to meet the same has been discussed in Table 4.6.6 and 4.6.7 respectively for both OBX and OB-II quarry.

The location of the mine entries and other underground infrastructures of the underground mine has been shown in the conceptual underground plan & sections in drawing no.'s SCM/MP/15/12 & SCM/MP/22/12.

The access to the underground mine has been proposed to be made by 2 declines and a common vertical shaft to service the 3 ore bodies. The location of the shaft has been decided on the basis of the centroid of the resource for all 3 ore bands as well as the available vacant space to accommodate the other ancillary infrastructure. Similarly, the location of the north decline has been conceptualized based on the available vacant space near the shaft so as to also facilitate the sharing of the common surface infrastructures to be built up around the shaft. In the absence of any suitable vacant space for locating the southern decline it is proposed to re-handle the low grade dump situated at the north western corner of the OB-II quarry to accommodate the same.

The incidental ore production during the development period shall mostly be through the declines. After commissioning of the vertical shaft it shall serve for both production and man and material winding. The cross cuts connecting the shaft to all three ore bodies are proposed to be developed at 100 m intervals at (–) 95 and (–) 195 mRL. The shaft is proposed to be sunk to a depth of 365m (i.e upto – 240 mRL) to cater to the two main levels as stated above. The shaft may be further deepened at a later point of time after further proving of the ore bodies at deeper levels. The surface declines are proposed to be driven upto a depth of (–) 35 mRL. Thereafter, access to lower levels and sub-levels shall be through a network of ramps as shown in the conceptual plan and sections.

The development schedule of the underground mine has been conceptualized taking into account the life of the opencast workings in all 3 ore bodies & its ultimate pit limits. The time required for developing the underground mine for production build up from the stopes have been synchronized to have a smooth transition from opencast to underground mining.

The tentative road map for the programme being initiated for shifting over to underground mining is given below as Table.4.6.5.

Table: 4.6.5
Tentative Road map for Underground Mining

Activity	**H1'11**	**H2'11**	**Jan' 12**	**Feb**	**Mar**	**Apr**	**May**	**Jun**	**July**	**Aug**	**Sept**	**Oct**	**Nov**	**Dec**
Feasibility Report														
Approval of Mining Plan														
Prior Environmental Clearance& CTO														
DGMS Approval														
Capital Scheme Approval														
Detail Design														
Pretender														
Tender														
Pre project Activity														
Start Decline(S)														

*Statutory jobs include approval of Modified Scheme of Mining, Environment Clearance, and Consent to Operate from State Pollution Control Board etc.

Considering the extents of the orebodies along the strike direction and the use of trackless diesel equipment for its development, adequate ventilation is perceived to be an important criterion in the development stages of the underground mine. In order to address this issue quickly during the development stage of the mine, two drifts in the waste rocks are proposed to be made at (–) 35 mRL to serve the northern and southern sections of the underground mine respectively. These ventilation drifts shall be connected to the surface at intervals along the strike direction by ~ 3.5m dia. ventilation shafts and shall thus serve as the main ventilation outlet for the mine. As these drifts are meant to serve throughout the life of the underground mine, it is proposed to be adequately supported to ensure its long-term stability.

The development within the orebody is proposed to commence below the crown pillars which has been shown in the longitudinal geological section for all 3 orebodies (Drawing no.'s SCM/MP/21/12).

A geotechnical study and subsequent numerical modeling work has been carried out to determine the safe crown pillar dimensions for all three ore bands. Insitu stress measurement and physical and engineering properties of the ore has been studied. As per geotechnical investigation report (enclosed as Annexure-22, Ref. pp-53) it is stated that a crown pillar will be unstable in the soft friable chrome ore material and pillars can therefore not be developed in this material.

Accordingly, the crown pillar has been designed to be below the weathered horizon of the northern and middle ore bands and within the hard and competent ore. As per the numerical modeling report (enclosed as Annexure-23, Ref.: pp-59) the ratio of the required crown pillar size to orebody width was determined to be 1:1 based on the insitu stress measurement done by MeSy. The same ratio was determined to be 1:2.5 based on the stress regime postulated by SRK's desktop study. However, presently the thickness of the crown pillar for all three orebodies has been proposed to be 60m in line with the requirements of the statute. The possibility of reduction of the crown pillar thickness shall be taken up later after further insitu stress measurements and detailed investigations from the underground workings and seeking permission for the same from the appropriate authority.

All the three orebodies are steeply dipping (80^0 to 90^0). The ore bodies are sufficiently far apart with the distance between the northern and middle ore bands being 250-270m. Essentially therefore there is no footwall and hanging wall side for these near vertical ore bands. With a view to optimize the development requirement, the ramps and the haulage drives/ drifts have been proposed to be located between the northern and middle orebodies. With this the number of intersections through the orebody shall be reduced to a great extent and shall thereby reduce the blockage of ore reserves in pillars and shall also avoid intersections through weak rock conditions. Since the distance between the northern and middle orebodies is far apart, the influence of mining (considering back filling of the stopes) on such infrastructure is therefore not likely. The advantages of the proposal of placing these infrastructures between the two ore bands far outweigh the disadvantages for the given geo-mining conditions.

The southern side of the southern body is having inferior strata conditions due to the presence of the Mahagiri fault between the quartzites and ultramafics together with shears and dykes along the strike direction of the orebody. In view of the presence of this zone of faulting and shears on the southern flank along the orebody, the haulage drives and ramp have been proposed to be located in the relatively more competent strata towards the north of the near vertical southern ore body. Moreover, the northern side is also closest to the vertical shaft position and avoids the cross cuts from the shaft to intersect the orebody. The above proposal of locating these infrastructures has been ratified from the geo-technical aspect by CIMFR, Nagpur in their report.

Methods of Underground Mining:

The selection of the mining method was based on extensive geotechnical studies and numerical modeling done by SRK, South Africa and CIMFR, Nagpur. The recommendations of both the studies were:

- The friable and soft chrome ore with weak soil like hanging and footwalls in the Northern & Middle friable ore bands up to the weathering horizons should not be mined from underground as it will be extremely unsafe;
- Below the weathering zone in the Northern & Middle friable ore bands both Cut and Fill method and Sub-level Open Stoping with post filling is possible depending upon the hardness of the orebody, and; similarly
- Both Sub-Level Open Stoping with post filling and Cut and Fill methods shall be applicable in the Southern lumpy ore band depending upon the extent of disturbance/ fracturing and soundness of the orebody and its wall rocks.

Sub-level Open Stoping:

The schematic diagram showing the method of Sub-level Open Stoping is given as Fig-4.1. The stope block is mined out from the "bottom-up", as a panel between sub-levels is mined and filled, the fill is then used as a platform for the above panel is be mined. Before the panel is mined the in-stope development, on the top and bottom, to the limits of the block must be completed. A raise or slot is developed up from the bottom ore drive to the top ore drive. This slot provides a free-breaking face for the ring blasting and ventilation.

The sub-level open stoping and its many derivatives, is relatively flexible in that it can accommodate varying widths down to a minimum of 2 m. The maximum mineable width is dependent on the rock mass characteristics of the ore body and the surrounding host rock.

Cut & Fill Stoping:

The cut and fill mining method is based on longitudinal drifts mined on strike. The drifts shall be 5 m wide x 5 m high and the length dependent of distance apart of the access crosscuts. Fig 4.2 shows a typical drift and fill mining with long anchor support for the hanging wall.

Access to the ore body is from ramps and the ore is mined in horizontal cuts and carried out in a "bottom-up" sequence. The drift is filled and mining always takes place on top of the fill material. Drift and fill mining is generally used in rock conditions that do not allow the opening up of the stope to the full width of the ore body.

The filling of the stopes shall be done with the rejects generated from the underground development drives as well as with the indigenously available overburden and mill tailings as a paste fill. A separate plant shall be set up in the surface to pump the fill material for the stowing of the voids in the underground stopes.

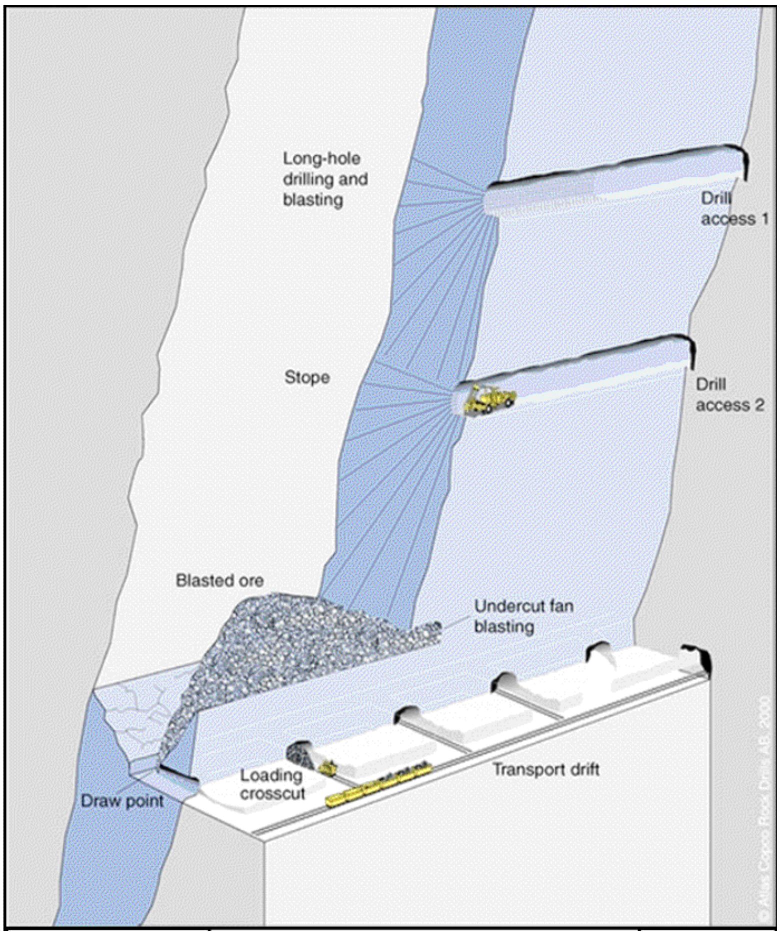


Fig. 4.1: Typical layout for Sub-level Open Stoping Method

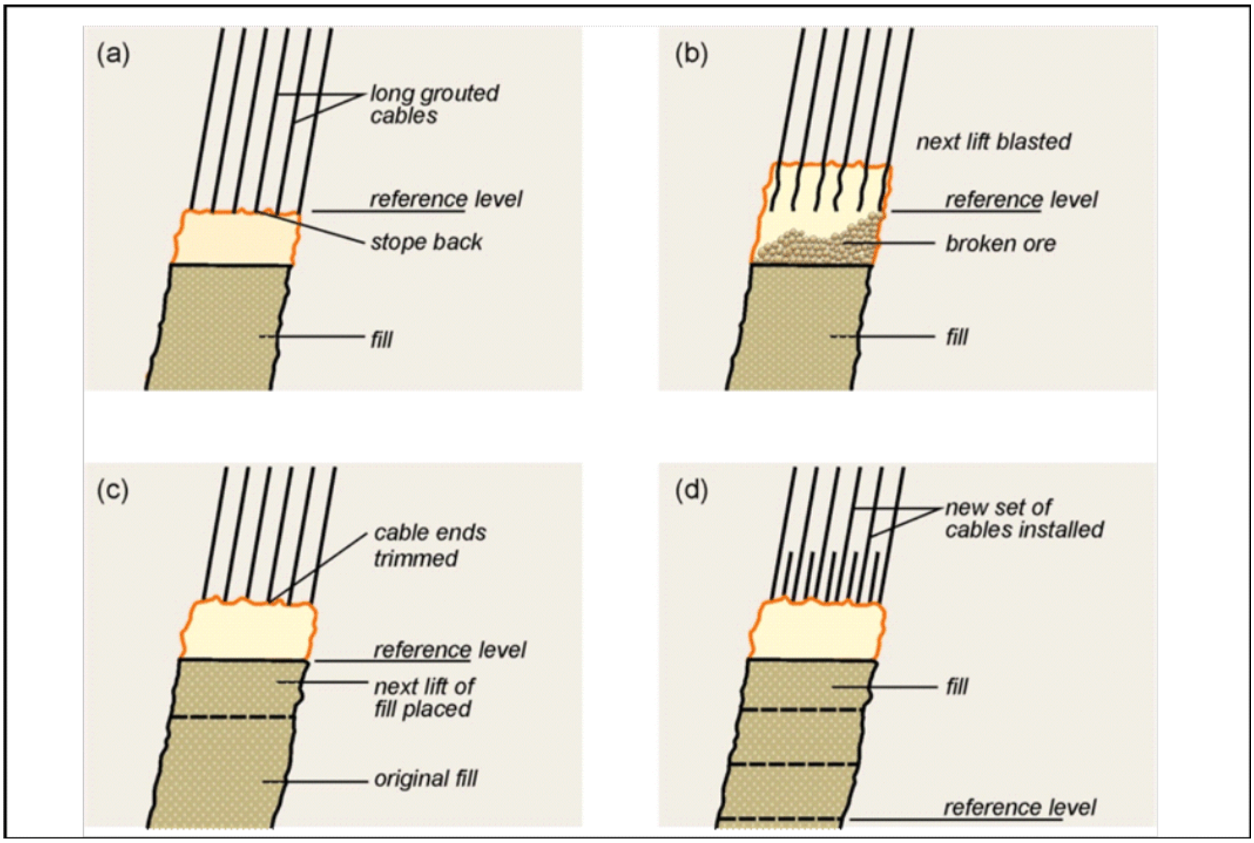


Fig. 4.2: Mining sequence in typical Cut & Fill Method in poor ground

The cross cuts from the haulage drives/ drifts are proposed to be driven to the orebodies during the Mining Plan period at intervals of 110m to sub-divide the ore body between the levels into panels. The dimension of the ore drives and sub-level intervals within the panels to be subsequently stoped out are proposed to be the same for both the above stoping methods, as further detailed in the Mining chapter. Presently, all stoping is being proposed within the ore horizons which are in the probable reserve category. Based on the feasibility study, the proportions of cut and fill mining in all three ore bodies have been estimated to be as follows:

- Northern Band: 50%
- Middle Band: 22%
- Southern Band: 40%

In order to save on time and cost of drilling the > 300-500m long boreholes at close interval from the surface, further exploratory drilling have been proposed to be undertaken from the proposed underground haulage drives/ drifts at a strike interval of 50-100m and ~ 30-50 m in the dip direction to bring the ore reserves to the proved category. The company proposes to engage its own team of geologists and geo-technical engineers to make a further detailed study of the geo-mining conditions of each of these panels after such levels are developed, drilling and geo-technical logging work is completed. Accordingly, the panels within the two levels shall be divided into different geo-technical domains for selecting the appropriate support system and any one of the above two applicable stoping methods. The cut and filling stoping method shall be applied mostly in the comparatively weaker zones. The company as per Rule 26 of the MCDR, 1988 and Rule 107 of MMR, 1961 shall accordingly apply for stoping permission well in advance (not less than 60 days). Stoping shall commence only after receiving such permission. The areas in which stoping shall be undertaken during the plan period to meet the year wise production requirements have been indicated in different colours in the longitudinal section for each of the three ore bodies.

CONEPTUAL PRODUCTION:

Chromite ore mining by opencast means shall continue till 2020-21 from the OB-X quarry. In order to obtain rated capacity of ROM production @ 2.4 million tone/ annum underground incidental production from OB-II quarry shall start early during 2012-13 where as incidental chrome ore production from OB-X quarry shall commence from 2014-15. Subsequently while the underground production rate would increase to attain the rated capacity and the ROM production arte from the opencast mining shall reduce for healthy changeover. While the mine shall achieve rated ROM production capacity of 2.4 Million tonnes/ annum of chrome ore ROM from 2017-18 onwards from both opencast and underground mining, the underground mine shall achieve the rated ROM production capacity of 2.4 Million tones/ annum after 2022-23 and shall continue with the same thereafter. The year wise ROM production rate as envisaged during the life of the mine by both opencast and underground mining is given in Table 4.6.6.

Similarly, the year wise finished/ marketable grade ore production rate for next 20 years is envisaged as given below in Table 4.6.7. While the mine shall achieve rated production capacity of 10.80 Lakh tones / annum of finished/ marketable grade chrome ore and 5 lakh tones/ annum of pyroxenite ore production from 2017-18 onwards from both opencast and underground mining, the underground mine shall achieve the rated finished/ marketable grade ore production capacity of 10.80 lakh tonnes / annum of chrome ore and 5 lakh tones/ annum of pyroxenite ore production after 2022-23 and shall continue with the same thereafter.

Table: 4.6.6
Year wise Chrome Ore ROM Production
(All Figures in Million tonne/ annum)

Year	Opencast Mining			Underground Mining			Total (Opencast & Underground)		
	OB-X	OB-II	Total	OB-X	OB-II	Total	OB-X	OB-II	Total
2011-12	1.28	0.11	1.39	0.00	0.00	0.00	1.28	0.11	1.39
2012-13	1.41	0.11	1.51	0.00	0.00	0.00	1.41	0.11	1.51
2013-14	1.64	0.12	1.76	0.00	0.00	0.00	1.64	0.12	1.76
2014-15	1.74	0.13	1.87	0.00	0.02	0.02	1.74	0.15	1.89
2015-16	1.74	0.00	1.74	0.04	0.07	0.11	1.78	0.07	1.85
2016-17	1.82	0.00	1.82	0.17	0.27	0.44	1.99	0.27	2.26
2017-18	1.56	0.00	1.56	0.54	0.31	0.84	2.10	0.31	2.40
2018-19	0.25	0.00	0.25	1.85	0.30	2.15	2.10	0.30	2.40
2019-20	0.20	0.00	0.20	1.90	0.30	2.20	2.10	0.30	2.40
2020-21	0.14	0.00	0.14	1.96	0.30	2.26	2.10	0.30	2.40
2021-22	0.00	0.00	0.00	2.10	0.30	2.40	2.10	0.30	2.40
2022-33	0.00	0.00	0.00	2.10	0.30	2.40	2.10	0.30	2.40

Table: 4.6.7
Year wise Finished/ Marketable Grade Ore Production
(All Figures in Lakh tonne/ annum)

Year	Open Cast			Underground			Total		
	OB-X	OB-II	Pyroxenite	OB-X	OB-II	Pyroxenite	OB-X	OB-II	Pyroxenite
2011-12	3.58	0.99	0.25	0.00	0.00	0.00	3.58	0.99	0.25
2012-13	5.26	1.06	0.00	0.00	0.00	0.00	5.26	1.06	0.00
2013-14	5.75	1.43	0.00	0.00	0.00	0.00	5.75	1.43	0.00
2014-15	6.16	1.59	0.00	0.00	0.23	0.00	6.16	1.82	0.00
2015-16	5.91	0.00	0.00	0.15	1.09	0.00	6.05	1.09	0.00
2016-17	7.00	0.00	2.50	1.04	2.81	0.00	8.04	2.81	2.50
2017-18	6.84	0.00	5.00	2.25	3.30	0.00	9.09	3.30	5.00
2018-19	2.09	0.00	5.00	7.06	3.24	0.00	9.15	3.24	5.00
2019-20	1.68	0.00	5.00	7.47	3.24	0.00	9.15	3.24	5.00
2020-21	1.18	0.00	5.00	7.97	3.24	0.00	9.15	3.24	5.00
2021-22	0.00	0.00	5.00	9.15	3.24	0.00	9.15	3.24	5.00
2022-23	0.00	0.00	5.00	9.15	3.24	0.00	9.15	3.24	5.00

Post Mining Land Use:

The Conceptual Mining Plan has been furnished as shown in Drawing No. SCM/MP/05/12. As mentioned in approved Scheme of Mining, back filling of quarries and its subsequent reclamation and rehabilitation shall be taken up after the economic pit bottom depths are reached along the entire strike of the deposit. Overburden/ associated mineral generated in 2011-12 to 2013-14 shall be dumped in the additional area of 100ha. Similarly the overburden/ associated mineral generated during 2014-15 to 2017-18 shall be backfilled in OB II quarry in a manner as shown in the drawing no SCM/MP/08/12. The land use pattern at the begening of the scheme period and at end of planned period (2017-18) is given below in Table 4.6.8; as per Drawing No. SCM/MP/13B/12 & SCM/MP/13A/12. The manner in which the area under different category is changing has been shown in the drawing no. SCM/MP/13D/12.

Table 4.6.8
Land use Pattern

Activity	As at the beginning of plan period (2012-13) in ha.	As at the end of Scheme period in ha.
Area to be excavated	201.50	175.49**
Storage of Top Soil	0.00	0.00
Over Burden Dump	37.47	76.11#
Mineral Storage	30.70	32.86
Infrastructure (Workshop, magazine etc)	7.92	7.92
Roads (Present L= 7812m, W= 25m)	19.05	19.05
Railways	0.00	0.00
Greenbelt	38.53	29.69
Tailing Pond	15.95***	10.00****
Effluent Treatment Plant	0.32	0.32
Mineral Separation Plant	18.38	18.38
Township Area*	36.18	36.18
Others (to be specified)	0.00	0.00
Area which will remain untouched	0.00	0.00
Total	406.00	406.00

Remarks:

* - includes roads within the colony area
** - includes pyroxenite section & excludes OB-II area as backfilled
*** - includes area of pyroxenite section for temporary storage of tailings
**** - excludes area of pyroxenite section
# - includes OB-II backfilled area

Mineral Beneficiation:

No such change is being proposed in the mineral beneficiation process during the plan period till 11.01.2013 apart from what has been proposed in the tailing management process is as enumerated below.

The water from the tailings is further proposed to be filtered by the introduction of parallel plate press filters as shown in Fig 8.2 of chapter-8. 3 such units having a capacity of 27 t/ hr each is proposed to be installed. The tails after dewatering shall then be in cake form and shall then be transported in dumpers to the proposed disposal site. The schematic layout of the manner in which these parallel plate press filters shall be installed within the Chrome Ore Beneficiation Plant as described in chapter-8.

The tailing dam constructed within the lease to store the tailings in slurry form within the pond so created is almost full. The stability study conducted by IIT, Kharagpur suggests that further heightening of the dam would be unsafe. In the absence of any further vacant space within the leasehold to store the tailings from the COB Plant, the company has identified and applied for an additional area of 73.685 ha having 65.315 ha of non-forest Government land for future solid disposal of the tailing after filtration of the water within the plant, which is under process. Till such time pyroxenite section of OB-II quarry had been temporarily discontinued to store tailings from the COB Plant. The Modified Scheme of Mining in this respect was approved vide letter no. 314(3)/2011-MCCM(CZ)/MS-08, dated: 22/07/11 (Annexure-7). While the installation of the filter presses is going as per schedule, it may take a further period of at least two years to acquire the above mentioned applied land after seeking forest clearance for the proposed road leading to the same. In order to take care of such eventuality, it is therefore necessary to make alternate arrangements to store the tailings for another two years beyond the end of this scheme period so that the production from the mine is not hindered. Accordingly, it is further proposed to store the solid tailings (after filtration in the filter presses proposed to be installed) in the present pyroxenite section area till 2014-15. It is expected that it shall take a year to evacuate the tailings so stored and rehandle the same to the applied area. Thus, the reopening of the pyroxenite section for the production of pyroxenite is proposed to be deferred to 2016-17.

Conceptual Environment Management:

The various environment parameters shall continue to be monitored as per conditions laid in the EC, Consent to Operate, and as per notifications issued from time to time. The following environmental protective measures that are being taken and further proposed to be maintained and continued in future are:

(i) Water sprinkling and wet drilling for dust suppression,
(ii) Wet beneficiation process and water jet system at feed hopper of processing plants to allay the dust,
(iii) Garland drains and toe wall at the foot of the overburden dumps to prevent soil erosion,
(iv) Treatment of mine discharge water and surface run-offs in the effluent treatment plants to ensure the discharge to be hexavalent chrome free,
(v) Recycling and reuse of the water from the tailings pond,
(vi) Oil and grease trap at the workshop areas to prevent soil contamination,
(vii) Treatment of domestic effluents in the Sewage treatment plant.
(viii) Acoustic enclosures for DG sets, rubber lining in the transfer points & discharge chutes to reduce noise levels
(ix) Development of the green barriers between the mining and colony area as well as along the lease boundary
(x) Safe storage and disposal of hazardous wastes such as used/ waste oil, old batteries, waste containing oil and ETP sludge to authorized recyclers and facility operators.

Conceptual reclamation

The Steel Company proposes to reclaim the open pits by back filling after the same reaches its ultimate pit bottom depths and the mineable reserves by opencast methods are exhausted. Wherever possible such backfilling shall be done from the bottom upwards in a retreating fashion. Thereafter, plantation is proposed to be taken up over such completely backfilled areas shall be taken up to rehabilitate the area. The mined out areas and benches that have not been proposed to be backfilled shall be reclaimed by plantation. Similarly, the overburden dumps which are proposed to be made in a retreating manner shall be reclaimed and rehabilitated by afforestation in a phased manner after it attains the designed extent. The areas proposed to be so reclaimed and rehabilitated have been shown in the conceptual plan.

The afforestation of the dumps and mining benches will be done by the following methods.

a) Method of pitting and planting.

(i) The worked out benches of the quarry and dumps will be reclaimed by making pits 0.5 m x 0.5 m x 0.5 m size 2m apart. The pits will then be filled with sweet earth, sand and cow-dung.

(ii) Neem cake powder is proposed to be applied in the pit to protect the plants from white ants.

(iii) Such ground preparation is proposed to be done before the monsoons after which appropriate varieties of saplings will be planted during the monsoon.

b) Method of planting by contour trenching

This method is proposed for slopes wherein contour trenches are dug at 3m interval along the contour. The excavated earth is stacked on the edge of the trench on the lower slope side to arrest the water flow that comes due to rains and accumulates on the trenches and gradually seeps through the strata enabling the planted saplings to get water and nutrients regularly for healthy growth.

Time bound action plan shall be drawn to reclaim the dead dumps, vacant area within the colony and to develop greenbelt. Year wise reclamation progarmme within the mining lease, additional area of 100ha and applied area of 73.685 ha is as given below in Table 4.6.9

Table 4.6.9

Year wise Reclamation Plan

Year	Within the Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total Plantation	
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)
2011-13	6.4	15875	12.1	30125	0.0	0.0	18.4	46000
2013-18	13.5	33750	39.9	99750	8.5	21125	61.9	154625
2018-23	42.0	105000	13.0	32500			55.00	137500
2023-28	68.4	170875					68.35	170875
2028-33	85.2	213025					85.21	213025
Total	215.4	538525	65.0	162375	8.5	21125	288.8	722025

The phase wise reclamation programme as per the approved FDP to cover the entire additional allotted area of 100 ha. is as follows:

Phase	Area (ha)	No. of saplings to be planted
Phase – 1	25	62500
Phase – 2	25	62500
Phase – 3	25	62500
Phase – 4	25	62500

The phase wise stabilization and rehabilitation programme for 25 ha. each shall be strictly adhered to, the annual programme may undergo slight changes depending upon the field conditions and market situations encountered. However company shall try to plant more trees to match the existing greenery of Mahagiri hills.

g) OPENCAST MINES :-

There are two separate quarries, which have been developed within the leasehold to produce lumpy chromite, friable chromite and pyroxenite ore. These quarries are:

- OB II Quarry
- OB X Quarry

Out of these, OB II is located in the foot hill of Mahagiri range which has been developed for mining the lumpy ore and OB X for mining friable ore band, which occurs between small hillocks of Mahagiri range. The pyroxenite section falls as a part of the OB II quarry.

Opencast mining is being carried out in all the quarries employing the shovel-dumper combination. While the high & medium grade ore is processed & marketed directly, the low & below low grade ores / beneficiable grade ores are stacked separately for further beneficiation through the Chrome Ore Beneficiation Plant.

The height of the benches varies between 6 m and 8m and the width varies between 12 m and 16 m. Tata Hitachi hydraulic Excavators (3.5 Cum. bucket capacity) and 35 T BEML Haulpak dumpers are used for removing overburden in the mine. The ore benches are being worked in slices by 0.9-2 cum capacity shovels and transported to Stack yard for dressing, sorting and stacking.

The requirement of drilling and blasting at Sukinda Chromite Mine is mostly confined to the OB II quarry where the overburden consists of hard quartzite and hard ultrabasic rock (serpentinites) and also for ore raising in both the quarries.

During the course of mining, a substantial quantity of low grade ore is generated. In order to conserve the scarce mineral, the company has set up the largest Chrome Ore Beneficiation Plant within the lease in the year 1990. A totally wet gravity separation process is used for the beneficiation of the ore. The details of the processing plants and chrome ore beneficiation plant have been mentioned in the approved Mining Plan

The detailed pit dimesion of the existing quarries has been discussed in Table 4.6.4.1 & 4.6.4.2. The pits viz. OB-II & OB-X quarries for the lumpy and friable ores respectively shall now be designed as per the revised permission received with respect to the overall pit slopes.

The stripping ratios for the various quarries on the basis of which developmental planning is done are given in Table 4.7.2 below:

Table 4.7.2

Quarry	OB X	OB II
Stripping Ratio, (Cum/t)	3.78	6.88

Mechanized opencast mining using shovel-dumper combination is being adopted for mining. Before mining the ore, the exposed ore body is demarcated into various marketable grade-zones by drawing channel samples across the strike direction at regular intervals. On this basis, the crude ore is raised or removed in slices of 3 to 4m each. The process of mining involves drilling, blasting, shoveling and trucking and the same is described below.

i) Drilling

The drilling in Sukinda Chromite Mine is mostly confined to grey lumpy zone where the overburden consists of hard quartzite and ultrabasics and also for ore raising in both hard lumpy and friable bands. In friable zone the blasting is required in 15% of the total overburden, which is generally soft and amenable to direct excavation by machines. Generally 150 mm dia. holes are drilled in the overburden benches and 100 mm dia. holes are drilled in the ore body. The drilling patterns followed at the mine are given below:

	Burden (m)	Spacing (m)	Depth (m)
OB II overburden	3.5 - 4.5	4.0 - 5.0	9.0 -11.0
OB X overburden	4.0 - 5.0	4.5 - 5.0	8.0 -9.0
Ore body	3.0 - 3.5	4.0 -4.5	5.0 -6.0

A sub grade drilling of 10% is usually done in OB II quarry in order to avoid toe formation. As a measure of dust suppression, the drills are provided with the wet drilling facility.

ii) Blasting:

On an average 25 -30 holes are blasted in a round using conventional explosive. The base charge is mainly of Indo Prime, OCG, PG1, PG2 with Primex and the column charge consists of Indogel, GN1, and ANFO etc. Recently the use of SME (Site Mixed Emulsion) has started for better safety, lesser ground vibration and better fragmentation.

Storage of Explosives-

One new magazine have been constructed for storage of explosives having E/ HQ/OR/ 22/ 178 (E23229). For taking shelter during blasting, total 6 nos. of portable type blaster’s shelters are provided. For transporting explosives from magazine to site 2nos of explosive vans of approved type have been provided (OR 04 - 5840 & OAU - 4078) having license no. E/EC/OR/22/57(E19284) and E/EC/OR/25/50/ (E19210) respectively.

The entire blasting operation is placed under the charge of two blasting foremen who report to the Under Manager, the overall in-charge of mining operation. 4 blasters assist the blasting foremen. In general one round blasting per day is done which occasionally goes upto two rounds. The issue and return of explosives is being carried on as per the regulation. The magazine in-charge keeps all the record of the same.

Pre-split Blasting:

The main purpose of controlled blasting technique by pre-splitting is to reduce back break and minimized damage to the final excavation line of the pit walls from the production blast, thereby, improved high wall stability leading to better safety. Ultimate pit slope can be steeper than the previous with improved face stability also resulting reduction in over burden excavation. So pre-split blasting is one of the controlled blasting techniques for slope stability, which is successfully practiced in Sukinda Chromite Mine.

It is a technique where a row of parallel holes closely spaced, along the excavation perimeter or final excavation, are fired simultaneously prior to the primary blast using decoupled charges or low strength and energy explosives. Final Excavation line of the bench is decided. Production drill hole of single row/multiple row of dimension (burden 3 m, spacing 3.5 m) is marked in square pattern.

Depth of the drill holes as per bench height +10% sub grade drilling and dia of drill holes is 150mm. Pre-split drill hole position are marked along the final excavation line. The pre-split line is of burden 3 m from the last row of production line and the spacing between the holes is 1.9 m. Priming cartridge of dia 150 mm and 6.25 kg is tied with detonating fuse to avoid toe formation. Column charge of cartridges 50 mm dia and 1 kg each is tied in 1.5 m intervals with detonating fuse.

All the activities above are conducted on the surface. The whole cartridges with detonating fuse are then inserted into the hole with a rope tied with it. Care is taken so that all the cartridges are in the straight line with detonating fuse by tightening the rope on surface.

The stemming length is 2 m from collar of the hole. Gasbag of 150 dia is used for air decking. It also provides very clean shear line. Production holes are charged and all the holes are stemmed properly. All the pre-split holes and production holes are connected in rows. Pre-split drill hole trunk line is connected with front row production blast trunk line by 500 ms delay detonator in one side and the production rows by 25 ms delay detonator. Site Mixed Emulsion Explosive is now being used.

iii) Shoveling

Both front-end and backhoe shovels having bucket capacity in the range of 0.9 cm to 3.6 cm have been deployed in the mine. In general the shovels having bucket capacity of 2.6 cm or more are being used with Haulpak / Hindustan dumpers for overburden removal and ore raising as well.

iv) Hauling/ transporting

For transporting the ore from the quarry to the stackyard or to the low-grade dump as well as for transporting the chrome concentrate and products of lumpy ore plant 25T Tetra/ Volvo dumpers are being used.
Extent of Mechanization:-

Considering the production and quantum of overburden to be excavated in the next 5 years as mentioned under section 4, the required fleet size has been calculated for Shovel, Dumper given below in the Table no. 4.7.3 to 4.7.4.

Table No. 4.7.3

SHOVEL										
QUARRY	**OB-X**							**OB II**	**OB II**	**Pyroxenite**
	Northern Benches				**Southern Benches**					
Model and Type	CK-300	EX-300LC	CK-300(BH)	PC 600	EX-600V	CK-170	CK-300(BH)	CAT 365	CK-300(FE)	CK-300(FE)
Bucket cap., Cum	2.7	1.6	2.7	3.7	3.0	2	2.7	3.7	3.7	3.7
Formations	Beneficiable Grade Ore	Ore to Stackyard / FOPP	OB	OB	OB	Ore	OB	Ore	OB	Pyroxenite
Fill factor	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.8	0.7	0.8
Swell factor	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.54	0.5	0.54
Cycle time, sec.	31.3	30	33	33	32	31.3	33	35	35	35
Dumper type	Volvo	Volvo	Volvo	Haulpak	Haulpak	TDV	TDV	HM	HM	HM
Dumper cap.(insitu), cum.	9	9	9	11	11	3	3	7	7	7
Spotting time, sec.	60	60	60	60	60	60	60	60	60	60
Loading time, min.	3.92	5.73	4.08	3.75	4.29	2.31	2.03	3.55	4.15	3.55
Capacity, cum/hr.	137.7	94.3	132.3	176.1	154.0	77.8	88.8	118.2	101.1	118.2
No. of day/annum	276	276	276	276	276	276	276	300	300	300
No of shifts/day	2	2	3	3	3	2	3	1	3	1
hrs/shift	6	6	6	6	6	6	6	6	6	6
Capacity/shovel/annum, cum	456068	312382	657498	875021	765102	257556	441223	212680	546091	212680
Volume, Lakh Cum	4	3.5	12	12	15	8	5	1	10	2.5
Working fleet required	0.88	1.12	1.83	1.37	1.96	3.11	1.14	0.47	1.83	1.18
Recommended working fleet	1	1	2	1	2	3	1	1	2	1
Availability	80	70	85	70	70	80	80	60	85	85
Fleet required	1.25	1.43	2.35	1.43	2.86	3.75	1.25	1.67	2.35	1.18
Recommended fleet	1	2	2	1	3	4	1	2	2	1
Remarks: One extra shovel recommended for Drain Cutting, Sump Cleaning and Dressing of the Benches.										

Table No. 4.7.4

DUMPER										
QUARRY	**OB-X**							**OB-II**	**OB-II**	**Pyroxenite**
	Northern Benches				**Southern Benches**					
Type	Volvo	Volvo	Volvo	HM 1035	Haulpak	Volvo	Volvo	HM 1025	HM 1025	HM 1025
Formation	Beneficiable Grade Ore	Ore to Stackyard / LOPP	OB	OB	OB	Ore	OB	Ore	OB	Pyroxenite
Dumper cap.(insitu), cum.	9	9	9	12	11	9	9	7	7	7
Loading time, min.	3.92	5.73	4.08	3.75	4.29	2.31	2.03	3.55	4.15	3.55
Lead distance, km.	2.8	2.3	5	5	5	1.3	4	1.5	3.5	0.5
Speed, km/hr.	25	25	25	20	20	25	25	20	20	20
Travel time, min.	13.44	11.04	24.00	30.00	30.00	6.24	19.20	9.00	21.00	3.00
Unloading time, min.	1	1	1.5	1.5	1.5	1	1	1.5	1.5	1.5
Total cycle time, min.	18.36	17.77	29.58	35.25	35.79	9.55	22.23	14.05	26.65	8.05
Dumper cap.(insitu), cum/hr.	29.41	30.40	18.26	20.43	18.44	56.52	24.30	29.88	15.76	52.14
No. of day/annum	276	276	276	276	276	276	276	300	300	300
No of shifts/day	2	2	3	3	3	2	3	1	3	1
hrs/shift	6	6	6	6	6	6	6	6	6	6
Capacity/dumper/annum, cum	97404	100673	90693	101482	91626	187184	120698	53790	85093	93859
Volume, Lakh Cum	4	3.5	12	12	15	8	5	1	10	2.5
Working fleet required	4.1	3.5	13.2	11.8	16.4	4.3	4.1	1.9	11.8	2.7
Recommended working fleet	4	3	13	12	16	4	4	2	12	3
Availability	85	85	85	90	85	90	90	80	80	76
Fleet required	4.71	3.53	15.29	13.33	18.82	4.44	4.44	2.50	15.00	3.95
Recommended fleet	5	4	15	13	19	4	4	3	15	4

Pumping Capacity:

Pumping of ground and surface run-off water from each quarry describing quantities and no. of pumps deployed for the purpose is given in Table: 4.7.5

Table: 4.7.5

Particulars	OB X Quarry		OB II Quarry		Total	
	Maximum	Minimum	Maximum	Minimum	Maximum	Minimum
Total (KL/ Day)	11865.6	5136	7910	3424	19776	8560
Pumping Hrs	24	24	24	24	24	24
Effective Pumping Capacity (KL /Hr.)	494	214	330	143	824	357
Efficiency	70%	70%	70%	70%	70%	70%
Rated Pumping Capacity (KL /Hr.)	706	306	471	204	1177	510
Rated Pump Capacity (Big)	288	288	288	288	288	288
Rated Pump Capacity (Small)	144	144	144	144	144	144
Effective Pump Capacity (Big)	202	202	202	202	202	202
Effective Pump Capacity (Big)	101	101	101	101	101	101
Required No. of Working Pumps (Big)	2.0	1.0	1.0	0.0	3.0	1.0
Required No. of Working Pumps (Small)	1	0	1	2	2	2
Availability	54%	54%	54%	54%	54%	54%
Total No. of Pumps (Big)	4	2	2	0	6	2
Total No. of Pumps (Small)	2	0	2	3	4	3
Number of pumps for emergency (small)	1		1		2	
Total number of pumps deployed	7		5		12	

For allied and miscellaneous operation, the following equipment are to be deployed as mentioned in Table 4.7.6;

Table 4.7.6

List of Service Equipment

Equipment	Capacity	Numbers	Equipment	Capacity	Numbers
Explosive van	110 HP	3	Diesel Tanker	12 KL	2
Flat body truck	110 HP	4		9 KL	2
Water sprinkler	28 KL	2	Cranes	5 Tonne	1
	25 KL	4		10 Tonne	1
	10 & 12 KL	4	Placer Dumper	110 HP	2
Track Dozer	430 HP	9	Box Tipper		2
	320 HP	3	Welding Van		1
	212 HP	2	Generator	100 KVA	1
	165 HP	1		320 KVA	1
Motor Grader	180 HP	3		275 KVA	1
Pay Loader	1.6 Cum	10		1000 KVA	5
	3.6 Cum	1	Service Van	110 HP	4

Apart from above ambulances, pit vehicles for supervision shall be used in operation. Old equipments shall be replaced with new modern efficient equipments based on hours of running in phased manner over the life of the mine.

h) UNDERGROUND MINES :

i. Mode of entry:

The schematic layout of the current opencast mining operation in respect to the three ore bodies is shown below in Fig.4.5. Being a brown field project, the entries to the proposed underground mine have been located considering the available space, ultimate open pit limits, and existing infrastructures. The number, type and size of the entries to the underground mine have been determined in order to service the ultimate underground production capacity of 2.4 Mtpa and also cater to the needs of ventilation and transport of men and material to the underground mine. Trackless mining method, as further elaborated below has been considered necessary to achieve the required/ designed levels of production from the underground mine.

Based on the above considerations, the access to the underground operations shall be serviced by the following infrastructure:

Surface Declines: Two declines shall provide the initial access to develop the underground infrastructure. While one decline shall serve the Northern and Middle Bands, another decline shall serve as an access to the Southern Band. The declines shall be of 5 m x 5 m X-section at an angle of 7° to 8°;

Vertical Shaft: A single vertical shaft of 6.3 m diameter shall be sunk to provide central rock hoisting facilities with access for men and material as required. The shaft shall service the underground operations via two main platform levels at Level -95 and Level -195 mRL's. With the increase in depth of the underground mine in the distant future, this shaft shall also be equipped and be capable of hoisting the entire production from the underground mine;

Ramps: The underground workings shall be serviced by four ramps. These ramps or underground declines shall be connected to the main surface declines at Level -35 mRL. 3 of such underground declines shall service the Northern and Middle Bands and 1 for the Southern Band.

The schematic diagram showing the location of the main entries to the underground mine is given in Fig.4.6. The drawing showing the location of these main entries is enclosed as Drawing No. SCM/MP/15/12.

The coordinates of the main entries to the underground mine is given in Table No. 4.8.1 below.

Table No.4.8.1
Coordinates of the Main Entries to the Underground Mine

Description	Northing/ Y-coordinate	Easting/ X-coordinate
Declines & Shaft		
North Decline	1900.0	3000.0
South Decline	1450.0	3100.0
Vertical Shaft	2007.4	2887.7

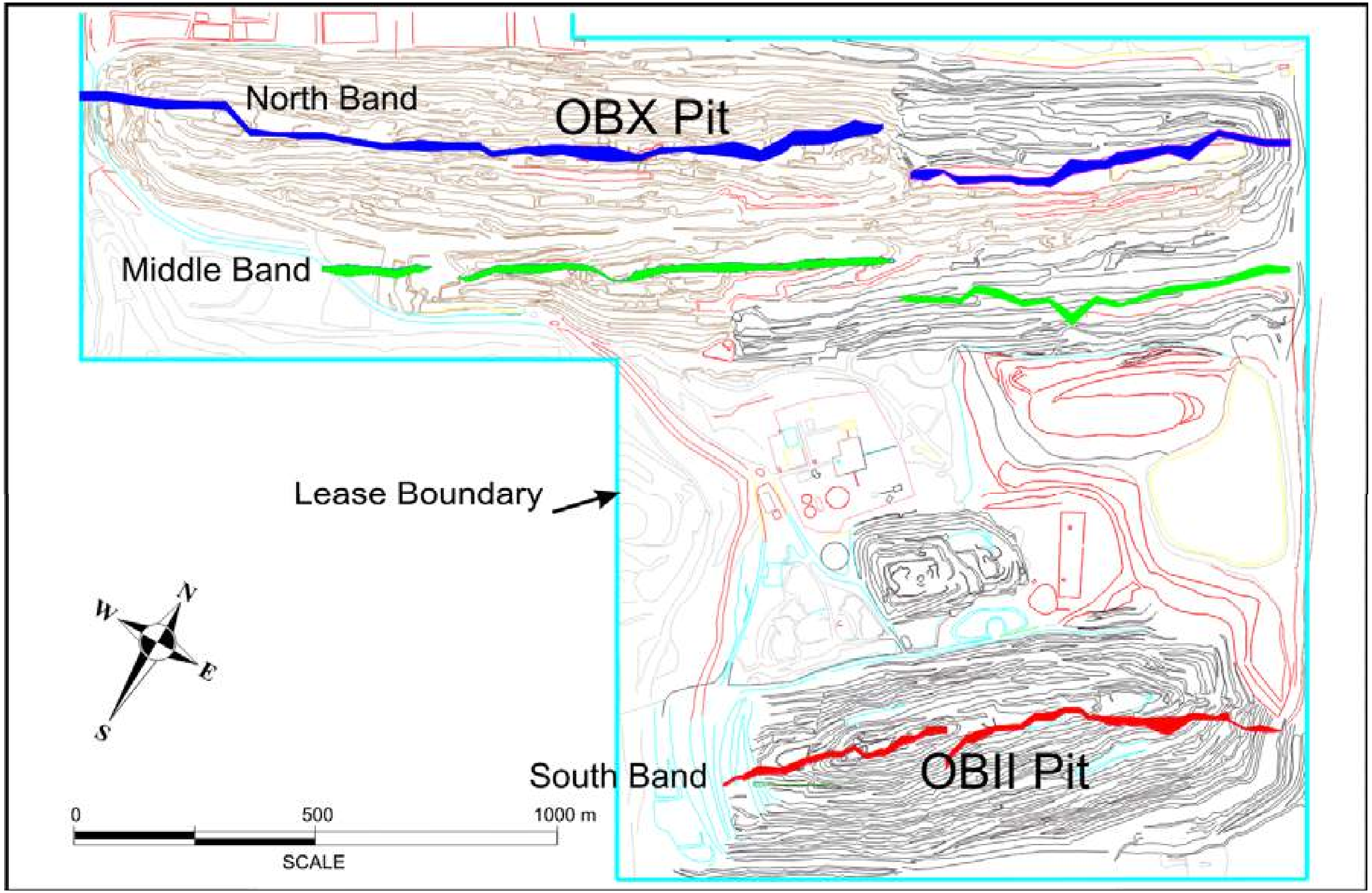


Fig.4.5: Schematic Plan View of the Mining Operations

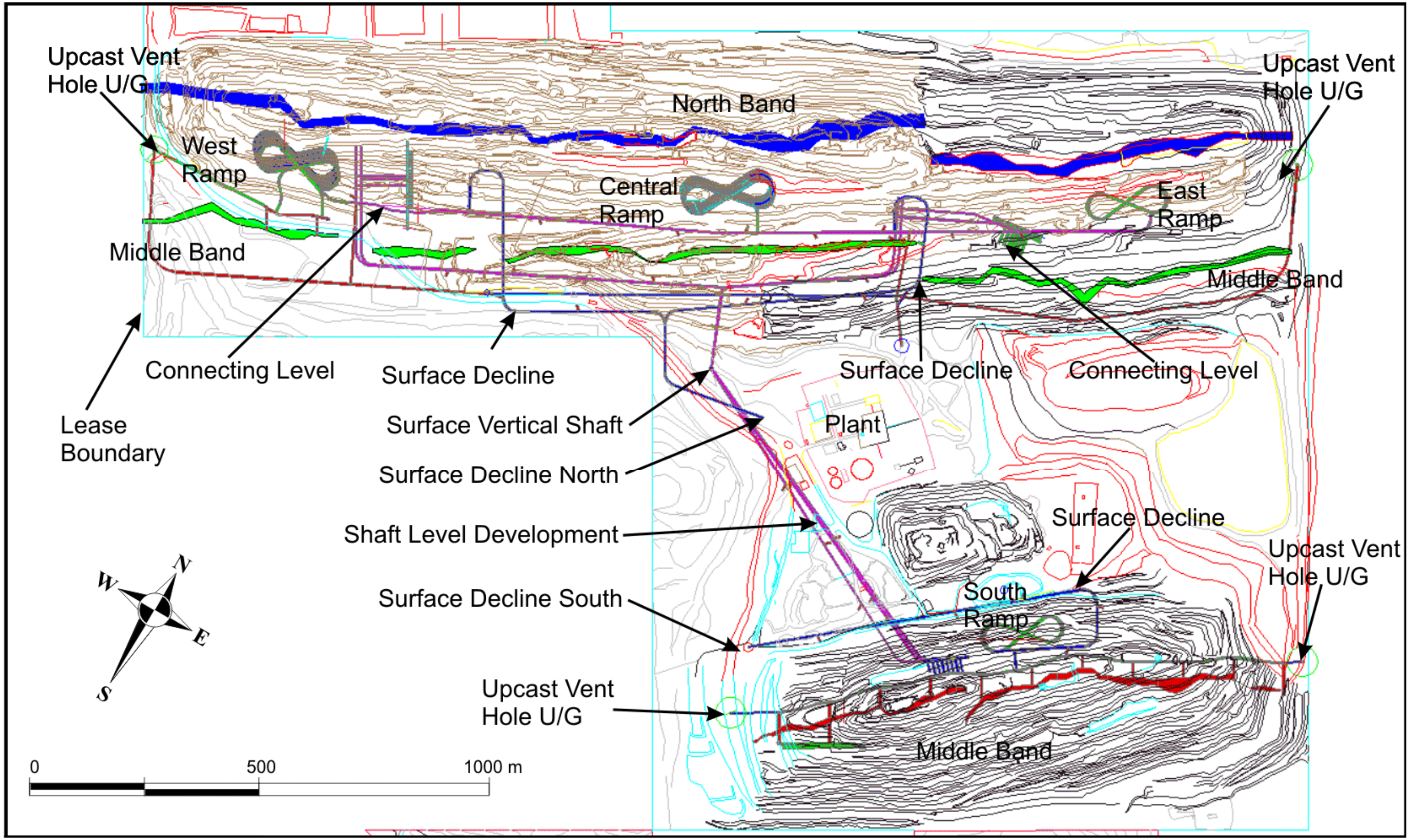


Fig. 4.6: Schematic Plan View of Mine Entries for Underground Mine

Surface Declines:

The schematic cross section of the declines is given in Fig. 4.7.

Trackless mining method has been considered for meeting the desired production levels. Surface declines has been considered as the most effective and quick means of entry to the underground mine In order provide easy access of the tyre mounted mining equipments which has been further elaborated below. The use of trackless equipments allows for greater flexibility of operations. The transport of waste and ore during development phase of the underground mine shall be through these declines. The declines shall be of 5 m x 5 m X-section and driven at an angle of 7^0 to 8^0. These surface declines shall also serve as intakes for the fresh air to the underground workings.

Vertical Shaft:

The schematic cross section of the vertical shaft is given in Fig. 4.8.

This common vertical shaft of 6.3 m diameter shall be sunk to provide access for men and material as well as central rock hoisting facilities. The shaft shall service the underground operations via two main platform levels at Level -95 and Level -195 mRL's for all three ore bands. All the main levels shall be connected by shaft cross cuts which shall serve as access to the shaft and ore transfer stations in each main level. This main vertical shaft opening shall be a downcast shaft and serve as one of the main intakes of fresh air to the underground workings.

The shaft shall be equipped to accommodate 2 cages of 10 t payload or 35 persons capacity each. It shall also be provided with two skips of 12 t capacity each.

Ramps:

The underground workings shall be serviced by four ramps, 3 of which shall service the Northern and Middle Bands and 1 for the Southern Band. The access surface declines shall connect the main ventilation drive at -35mRL which is around 160m below the general surface ground level. This is the starting level for the ramps for all the three bands. Three ramps shall be developed to access different level and sublevel horizons of the Northern & Middle ore bands. The size of the ramps shall be 5m x 5m both for the development as well as production stages of the mine. During the development stage two exhaust ducts and one forcing duct is to be established in the ramps to simultaneously allow the movement of the mine trucks within it. The waste rocks to be transported by mine trucks shall be through these ramps. Compressed air, drilling water and drinking water lines shall be laid along these ramps as permanent fixtures, to serve the service requirements for the underground faces and stopes.

The ramps shall be developed in a figure of eight pattern as shown in the schematic diagram in Fig.4.9.

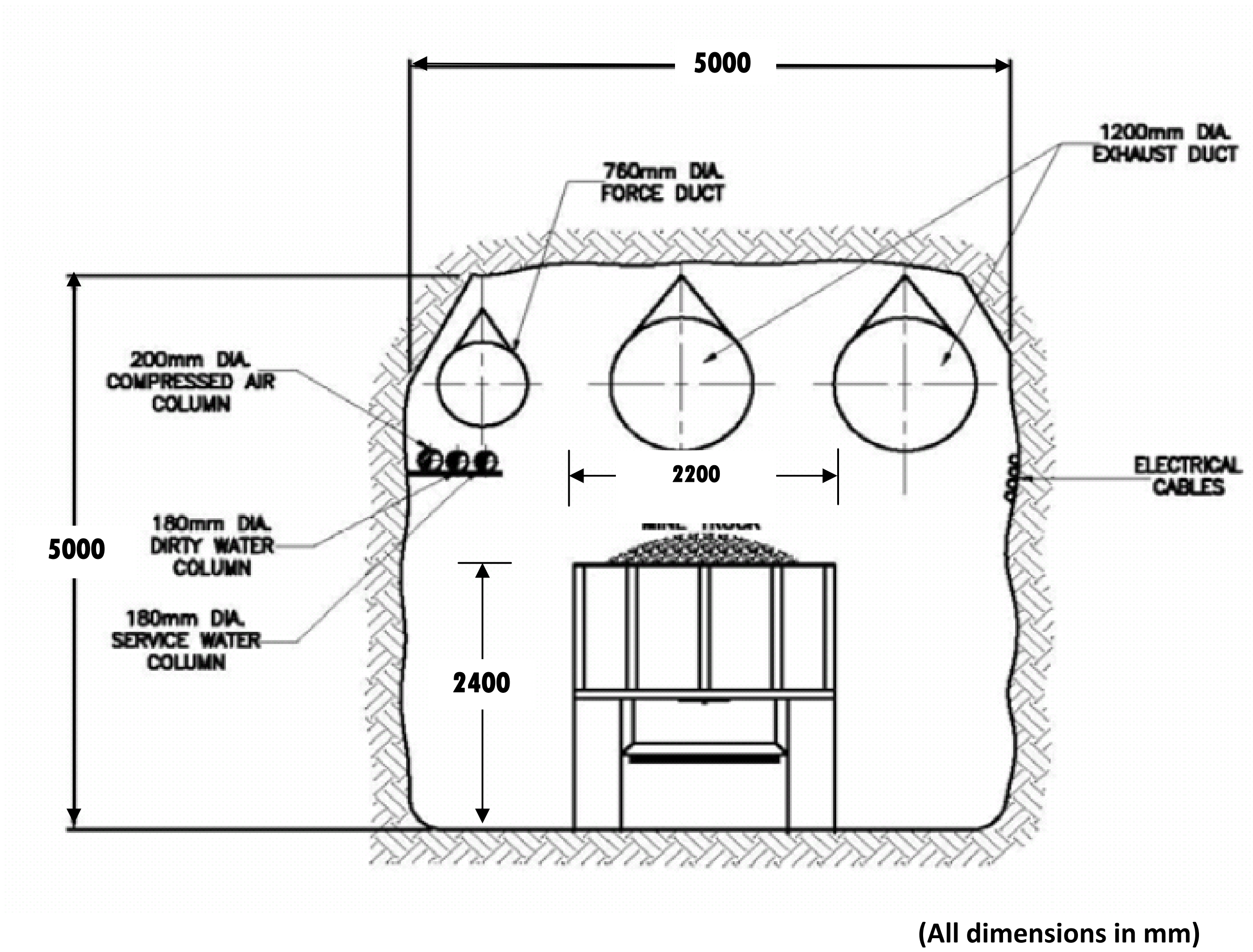
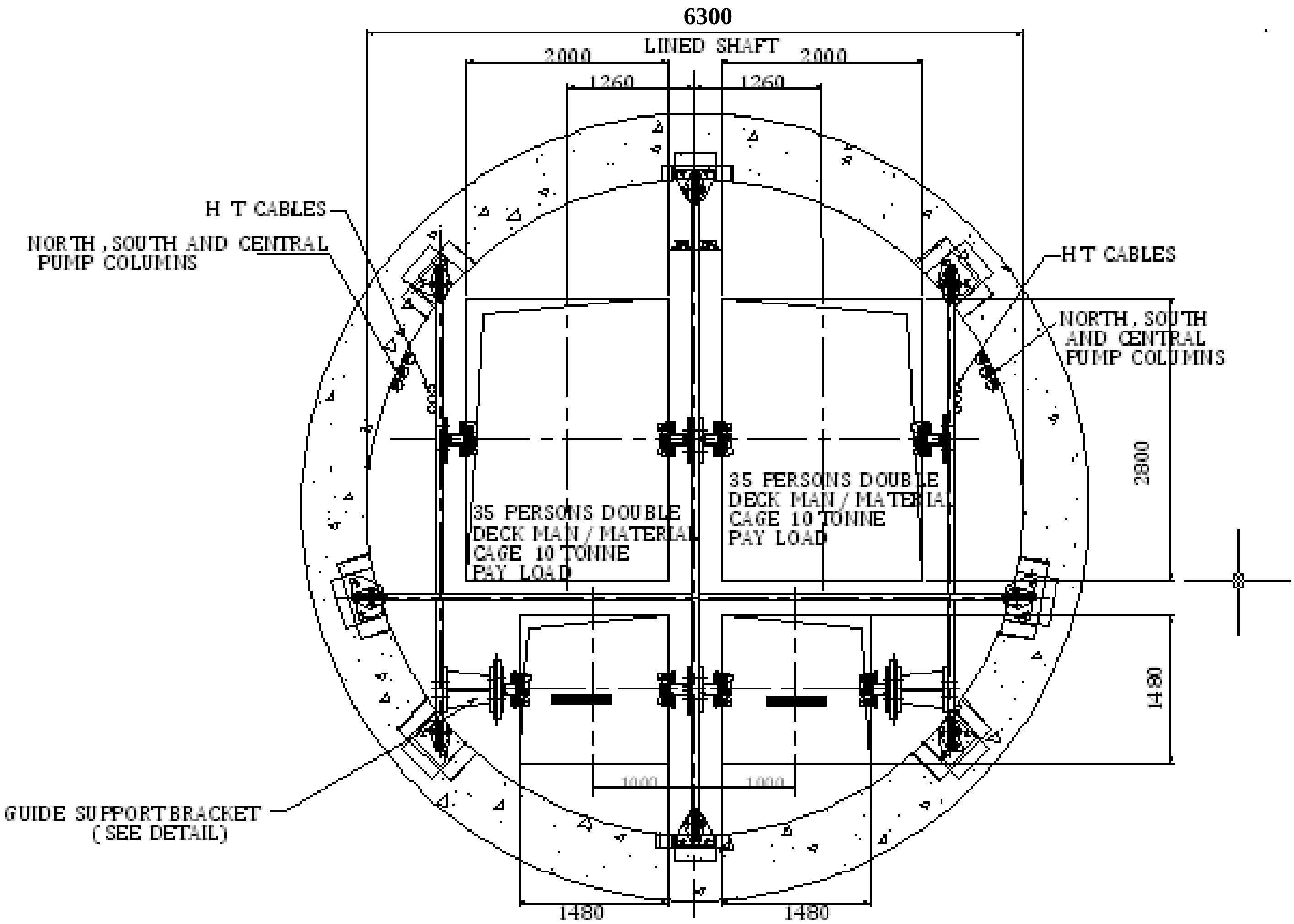


Fig. 4.7: Schematic X-section of the Declines



(All dimensions in mm)

Fig. 4.8: Schematic X-section of the Main Vertical Shaft

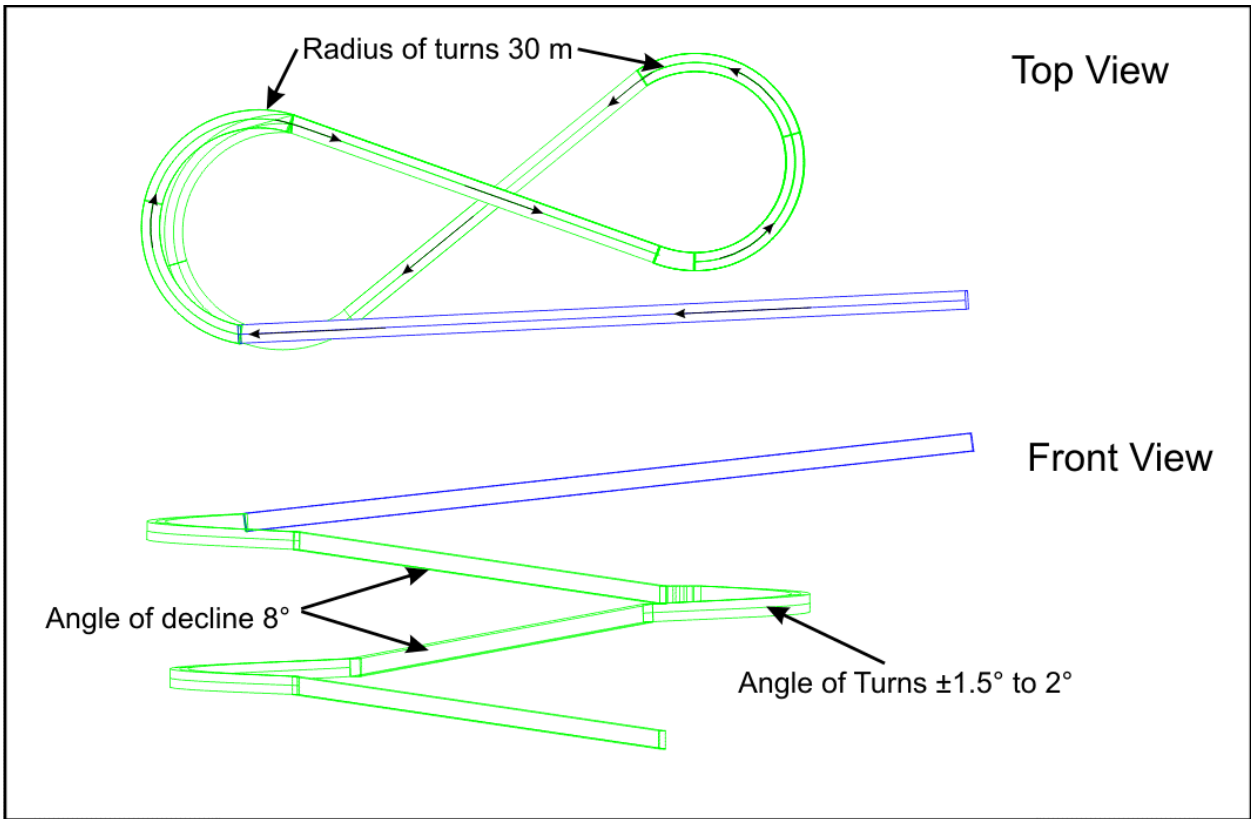


Fig. 4.9: Schematic Plan & Elevation View of the Ramps

ii. System of winding and hoisting:

The vertical shaft shall be initially sunk to a depth of 365 m and shall be fitted with two counterbalancing cages and skips. Considering the constraints of space for surface infrastructure and future increased depths of the underground workings, tower mounted Koepe winding system is proposed to be adopted. The winding and hoisting system of the mine shall follow the following sequence from development stage to the final production stage.

a. During the development phase of the mine the entire ore and waste shall be transported through the declines with trackless equipments like LHD (Load Haul Dump) of 4-7 m3 and LPDT (Low Profile Dump Truck) of 15-20 m3 capacity. Loaded LPDTs from each sublevel shall move to the nearest level drive to reach the decline.

b. After commissioning of the shaft all ore hoisting shall be done by the shaft skips of 12 t capacity. There shall be two such skips of counterbalancing type. The skips shall be loaded at the skip loading station located at -215 mRL which is connected by ore passes from the grizzly of each main production level at -95 and -195 mRL's. Two 10 tonne pay load capacity cages (35 person capacity) shall also facilitate the lowering and hoisting of man and material from/to the underground workings.

The schematic vertical section of the shaft and its main levels is given in Fig. 4.10.

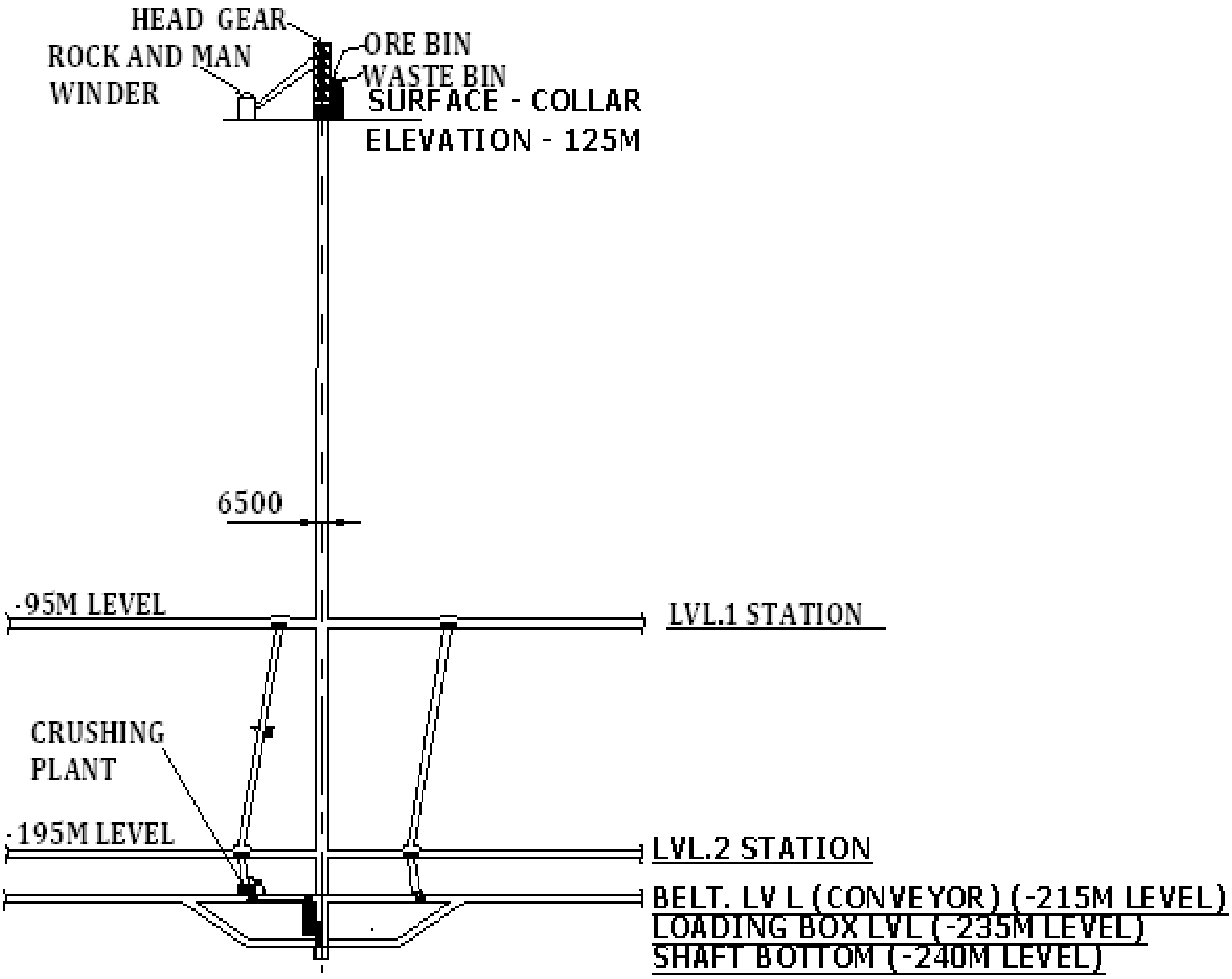


Fig. 4.10: Schematic view of the Vertical Section of the Shaft

Skip and cage hoisting capacity:

2 No.'s of Double drum tower mounted Koepe winders shall be installed having a rating of 2 MW each. Both the cage and skips shall be counterbalancing type, so no counterweights shall be there. The main shaft & winder specifications are as follows:

Required skip hoist capacity = 8000 tons per day of ore.

Assuming,

2 number of 12 t skips each at an operating speed = 15 m/ sec
2 number of cages (Type- double deck) with 35 men carrying capacity (10 t payload) each at an operating speed = 5.5 m/ sec

Skip capacity calculation:

- Loading time- 20 sec
- Acceleration time -10 sec
- Travel time- 20 sec
- Deceleration time -10 sec
- Dumping in surface ore bin - 10 sec
- Total time for one cycle - 70 sec, i.e. 1 min 10 sec
- Trips in an hour - 51 trips
- Running hour in a day - 2 shifts - 14 hr, (assuming, I hr maintenance in each shift)
- Total trips in a day- 51 x 14 = 714 trips
- Tonnage in a day- 714 x 12 t(Skip capacity) = 8568 tonnes
- Steady state production capacity = 8000 t /day ~ 2.4 Mtpa

Cage capacity calculation

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- Acceleration time- 20 sec
- Deceleration time- 20 sec
- Traveling time(to -195 m level)- 60 sec
- Total traveling time for one down trip- 100sec
- Trips in an hour(½ mode cage travel time, i.e. 30 min)- 18 trips
- Man shifting capacity- 35 x 18 = 630 persons/hr
- Proposed highest direct underground manpower in a shift ~ 600

Linkage to ventilation system:

The 3D schematic diagram given below in Fig. 4.11 shows the manner in which the main vertical shaft and north decline together with other ventilation shafts that are further proposed, shall serve as upcast or downcast airways for the ventilation system of the underground mine.

The two up cast airways to surface and the return airway on Level -35 are important for the opening up the mine but shall form part of the permanent ventilation system once the connected to the East and West up cast airways. There shall also be temporary return airways that shall follow the ramps so as to minimize the length of return air ducting that shall require to be installed in the ramp as they are developed. These temporary ramp airways shall be connected to the return airway on Level -35. The return airway on Level - 35 shall be 5 m x 5 m. There are two additional downcast airways that feed fresh air to Level -95.

The 3D schematic diagram given below in Fig. 4.12 shows the manner in which the main vertical shaft and south decline together with other ventilation shafts that are further proposed, shall serve as upcast or downcast airways for the ventilation system of the underground mine.

The coordinates at which the various ventilation shafts shall be located and its dimensions are given in Table 4.8.2 below.

Table 4.8.2:
Coordinates & Dimensions of Ventilation Shafts

Description	Northing/ Y-coordinates	Easting/ X-coordinates	Diameter, m
Ventilation Shafts			
Northwest - Upcast	2483.3	1674.9	4.1
Northeast - Upcast	2450.3	4162.9	4.1
North Decline West - Upcast	2172.0	2397.9	2.5
North Decline East - Upcast	2056.6	3301.4	2.5
South Decline - Upcast	1488.4	3704.9	1.5
South West - Upcast	1255.8	2928.5	3.5
South East - Upcast	1366.4	4173.4	3.5
North West - Downcast	2188.1	2461.6	3.5
North East - Downcast	2087.7	3230.8	3.5

The drawing showing the location of all mine openings (vertical shaft, declines and ventilation shafts) to the underground mine workings is enclosed as Drawing no. SCM/MP/15/12.

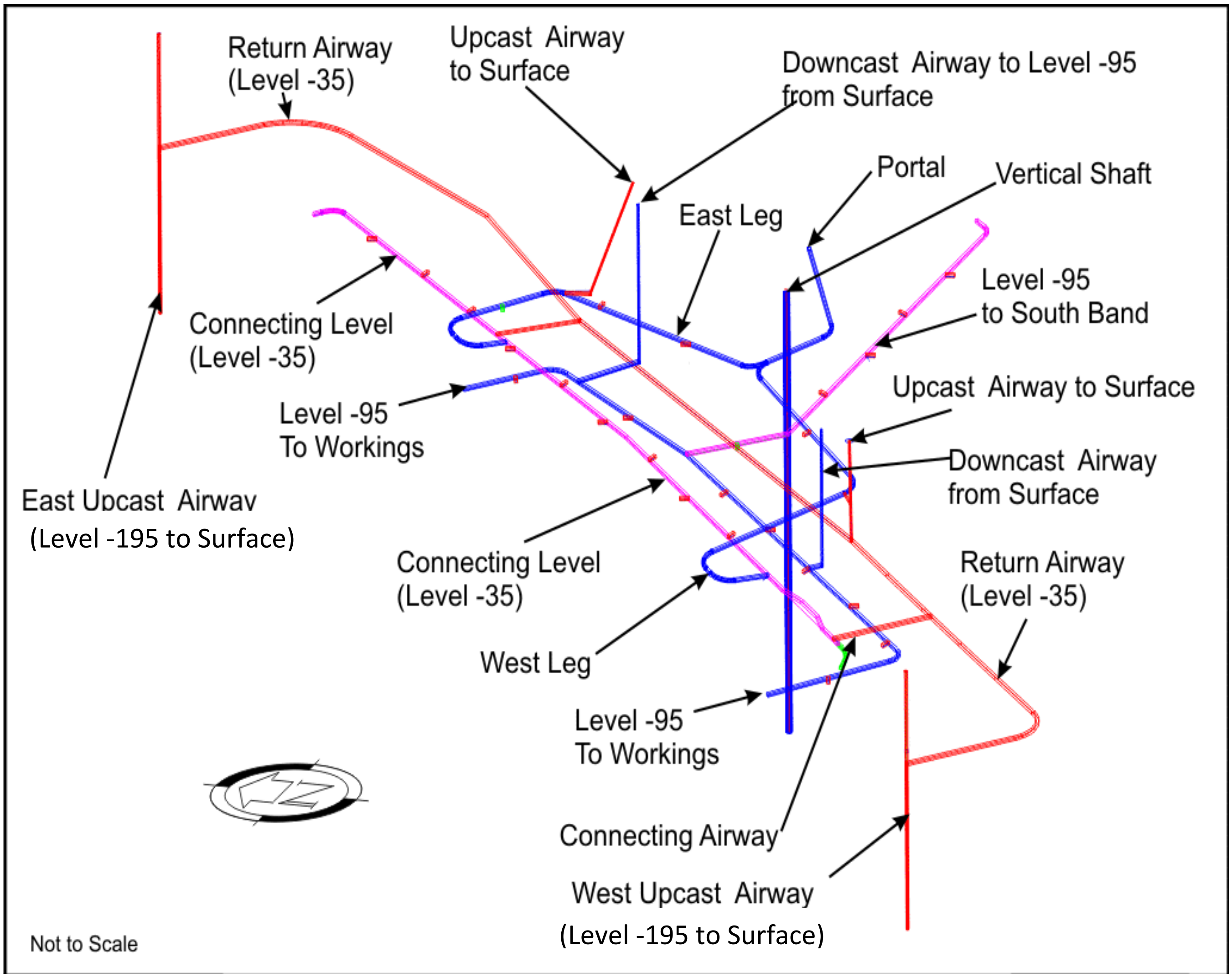

Fig. 4.11: 3D Schematic View of the Ventilation System for North & Middle Band sections

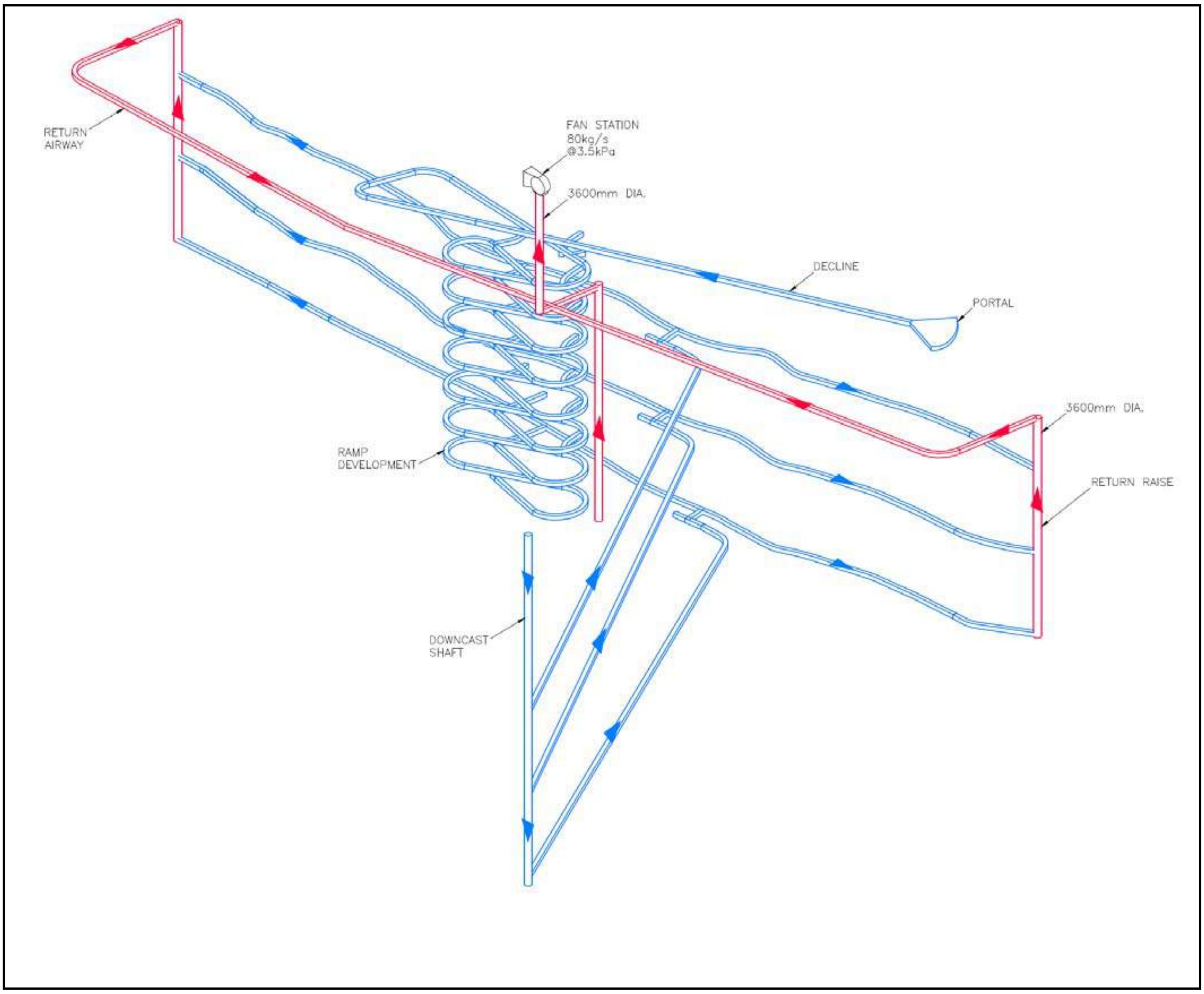

Fig. 4.12: 3D Schematic View of Ventilation System for South Band section

iii. Underground layout:

Vertical Shaft and South Band Infrastructure

The vertical shaft shall service the mining operations on two main levels, Level -95 and Level -195 mRL's. Fig.4.13 given below shows the infrastructure for the vertical shaft and the Southern Band. Access to the sub-levels shall be from the ramp. The sub-levels shall be 20 m apart.

North and Middle Band Infrastructure

The initial access to the North and Middle Bands shall be via a single decline from the surface that splits into two in order to reach the west and east sections of the two ore bodies. The decline goes down to Level -35 which is around 160 m below the general ground/ surface level. Fig.4.14 shows a plan view of the main infrastructure from the north surface decline. The level development to the North & Middle bands split to service the East and West sections of the mining operations.

A drive shall be developed on Level -35 to link up the declines and the three ramps. Fig.4.15 given below shows a schematic plan view of this drive at Level -35 mRL. The detailed drawing for Level -35 mRL is enclosed as Drawing No. SCM/MP/18/12 (sheet no. 1)

Similarly, Fig.4.16 below indicates the schematic plan view of Level -195 mRL. The detailed drawings for Level -95 and -195 mRL are enclosed as Drawing No.'s SCM/MP/18/12 (sheet no. 1) & SCM/MP/18/12 (sheet no. 2) respectively. The dimensions of the underground development drives & raises are given in Table No. 4.8.3 below.

Table 4.8.3
Summary of dimensions of underground infrastructure

Description	Width (m)	Height (m)
Declines	5	5
Haulage/ Waste Drives	5	5
Stope / Ore Drives & X-cuts	4	4
Raises (diameter)	1.8 - 4.1	

The above dimensions of the decline and waste drives have been considered to allow for the free movement of the underground dump trucks and also accommodate the service lines inside it (Refer Fig.4.7). The main decline and waste drives shall also serve as an air intake path to the levels and sub levels respectively. The sizes of the raises to serve as the ventilation return paths from the levels and sub levels will vary in size from 1.8m to 4.1m diameter depending upon the ventilation requirement of the area and stopes.

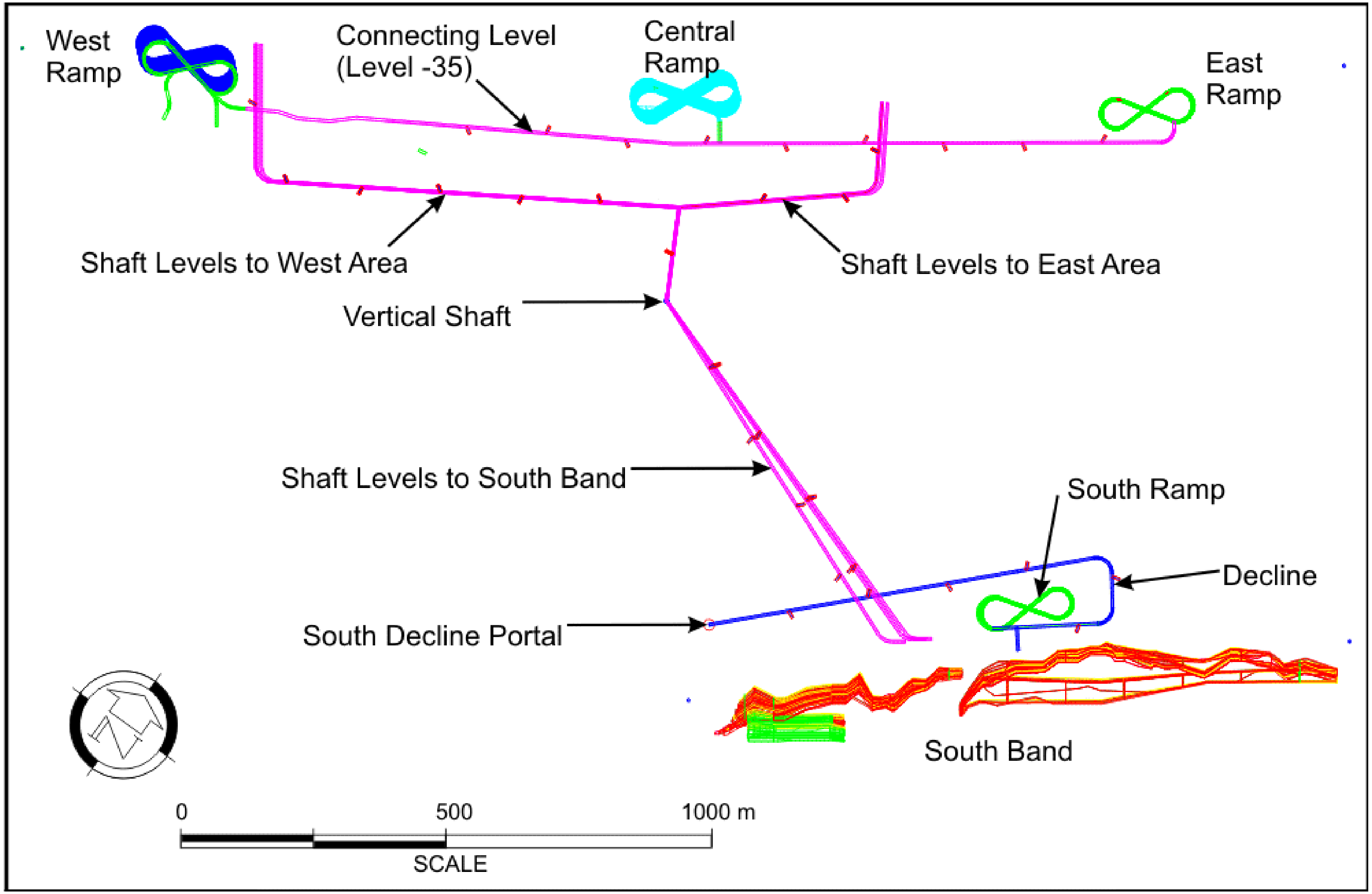

Fig.4.13: Mining Layout of the Southern Band infrastructure & Shaft connectivity

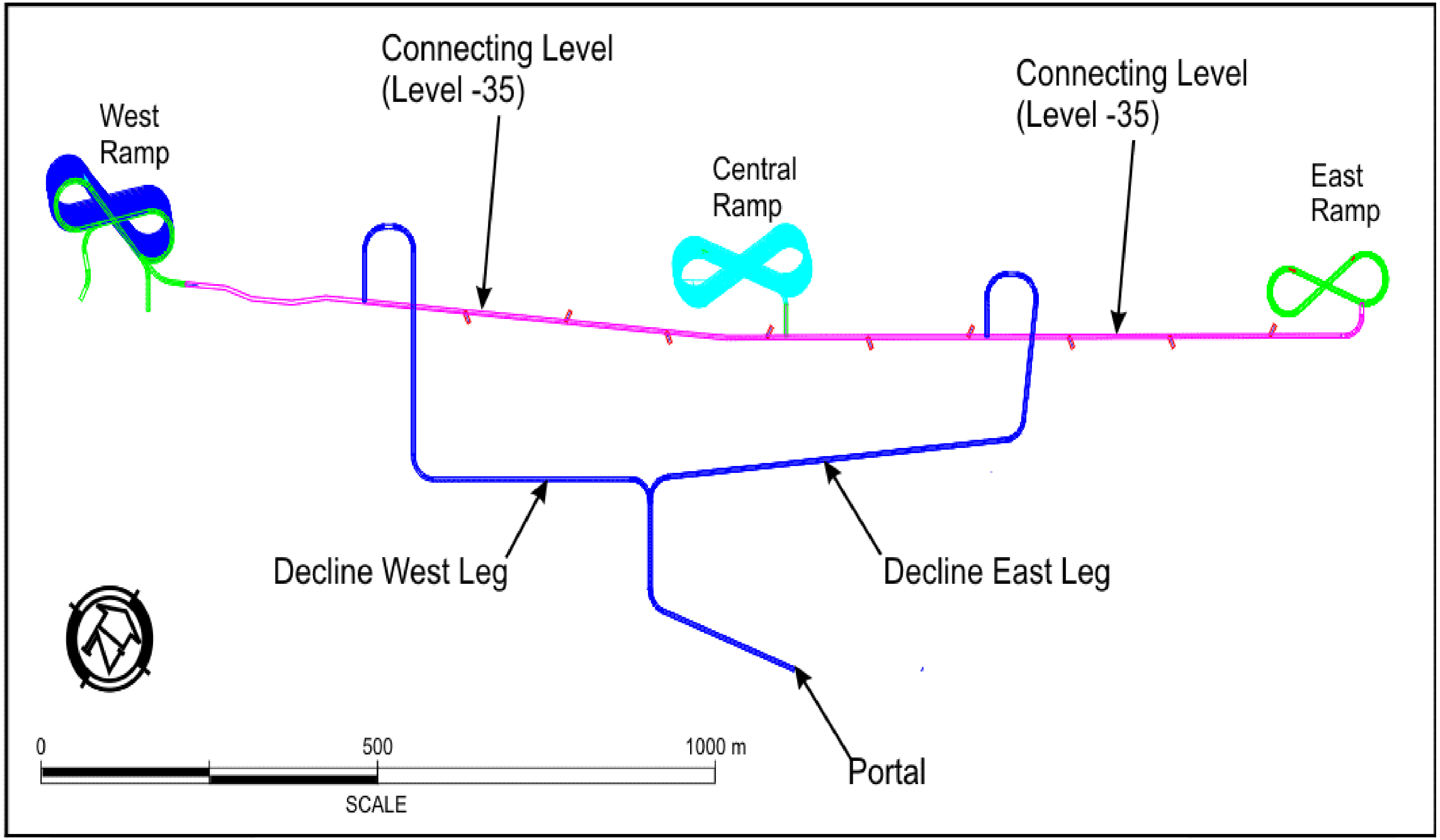

Fig. 4.14: Mining Layout of the Northern & Middle Band infrastructure

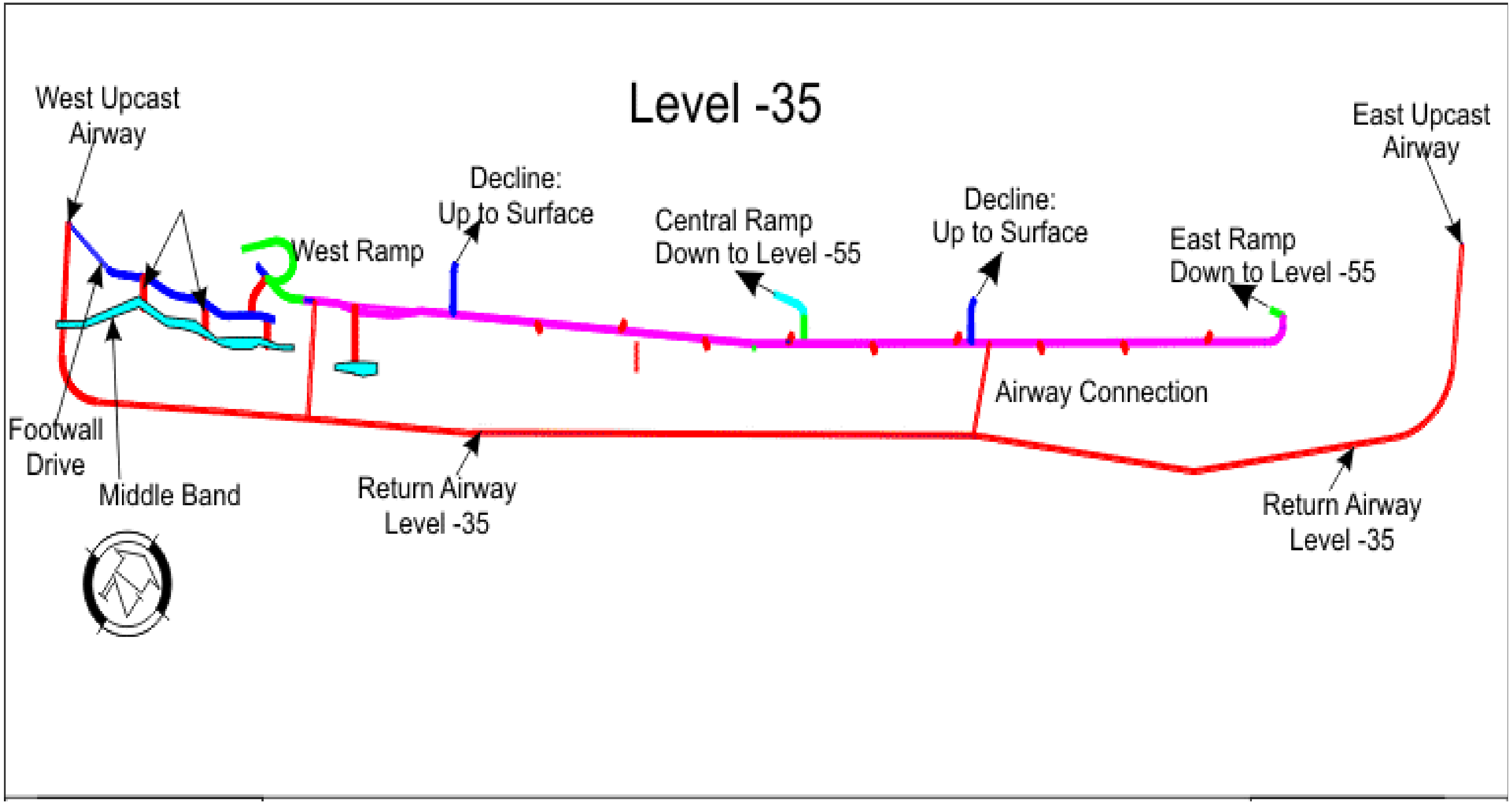

Fig. 4.15: Schematic Mining Layout of Drive at Level -35 mRL

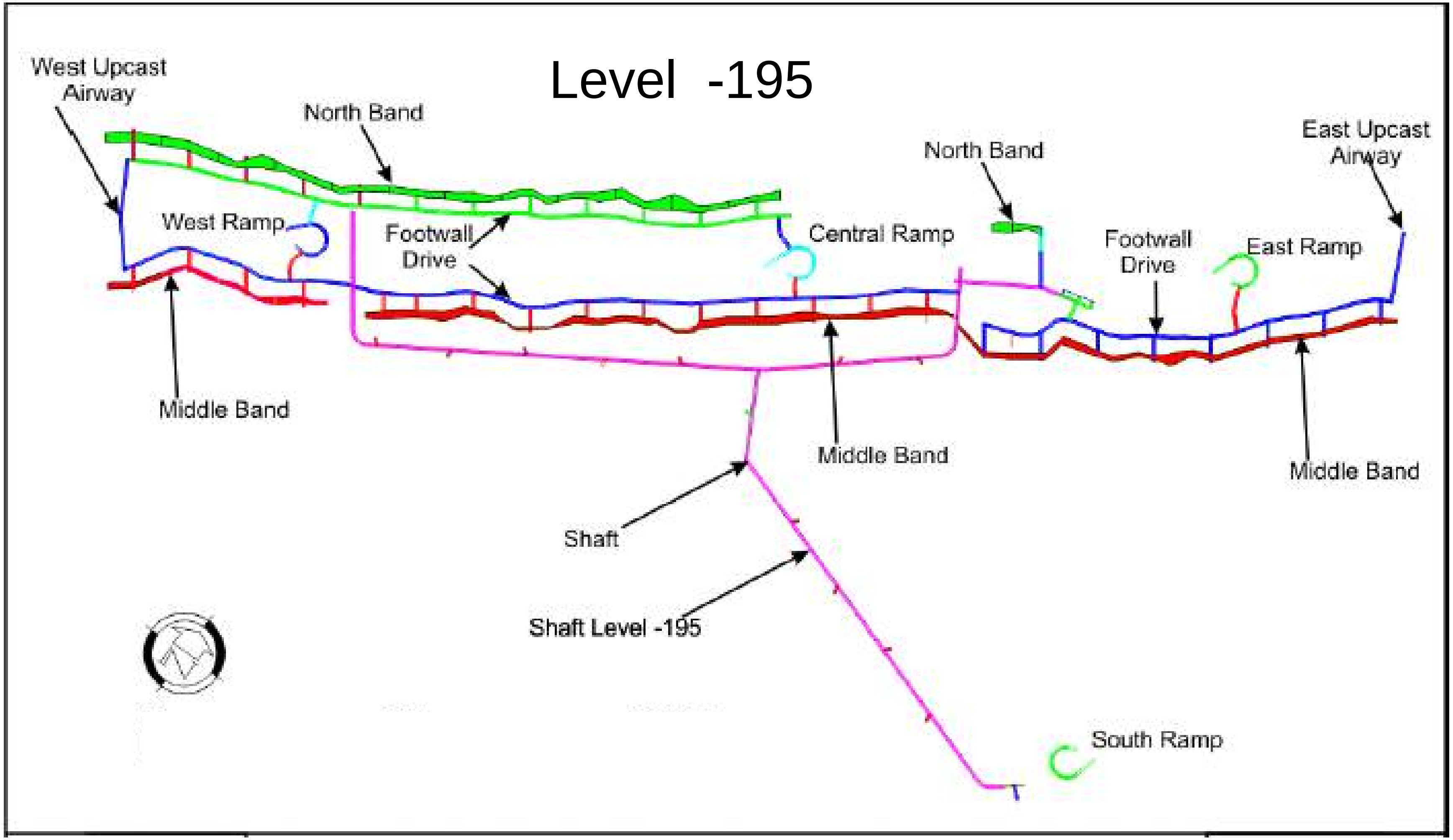

Fig. 4.16: Schematic Mining Layout of Drive at Level -195 mRL

Depth of Underground Workings:

As already stated earlier under the Geology chapter; deep seated weathering has been observed through exploratory drilling in the North & Middle Friable ore bands. The physical character of the weathered material in both these friable chromite bands within the top weathered zone is soft and crumbles to sand like powdery fine material. The adjacent strata consisting of nickelliferous limonites on both sides of these near vertical friable ore bodies are also soft and soil-like within and above the weathered horizon. Below the weathered zone the unweathered ore is hard and lumpy and there is a transition zone where the ore characteristic changes from soft-friable to hard-lumpy. The adjacent strata and wall rocks also changes below this weathered horizon to hard serpentinites.

The depth of the weathering varies from about 0 mRL in the west to below -200 mRL in the east for the North Band, while it is less than -200 mRL in the Middle Band. The schematic longitudinal sections for the North & Middle friable ore bands showing the depth of weathering are given in Fig.'s 3.6 & 3.7 above.

Detailed geo-technical studies by expert agencies have been undertaken. However, no underground mining method was found to be applicable and feasible for mining the soft friable ore below the ultimate open pit limits and within the weathered horizon.

In view of the above adverse physical nature of the North & Middle ore bodies and the adjacent strata conditions, the mining of the ore below the weathered zone, i.e. in the hard strata at greater depths, has been planned. Conventional and established underground mining methods as described later in this document has been found to be feasible for mining only the hard-lumpy ore that was found to be occurring at greater depths.

The schematic longitudinal sections of the mineable zones for the North and Middle ore bodies based on the geological exploratory data and proposed to be taken up for underground mining during the plan period are indicated in green and red colour in the Fig.'s 3.8 & 3.9 above.

Sub-Level and Ore Body Development

The ore bodies shall be extracted from sub-level drives developed off the ramps at a distance of approximately 30 m from the ore body, and crosscuts into the ore body are developed at a spacing of 110 m. The drives and crosscuts are sized at 5 m x 5 m. The sub-levels shall be spaced at 20 m. The sub-level drives shall be developed for the entire strike length of the available ore body.

The crosscuts shall be developed through the ore body to determine the width of the ore body. Reef drives shall be positioned, based on the width of the ore body and number of blocks across the ore body, and then developed in an east and west direction on strike to delineate the stope. The drives shall be stopped at least 5 m from the extent of the stope, so as to form a barrier of 10 m between the stope drives. The stope drives shall be developed at this stage, rather than later, to provide an early production of ore. The stope drives is proposed to be developed at 4 m x 4 m, to improve the stability of the excavation.

The Rock Passes

The broken ore and rock from the North and Middle Band shall be handled by three rock pass systems. The west ore pass system shall handle rock from the western and central areas and shall cover Level -35 to Level -195mRL. The eastern section of the North Band shall be served by the north east ore pass system that shall cover Level -175 to Level -195mRL. There is no mining in this eastern section of the North Band because the ore above Level -175 is in the weathered zone and is not mined. This ore pass shall also be able to handle the ore from the east-central area of the North Band and the Middle Band.

The eastern section of the Middle Band is served by the Middle East ore pass system from Level -35mRL to Level -195mRL. The South Band is served by the south ore pass system and shall handle rock from Level -35mRL to Level -95mRL. The rock pass systems that service the North and Middle Band consists of three passes, two for ore and one for waste, so as to keep the different ore and waste rock types separate. Fig. 4.17 shows the positions of the various rock pass systems.

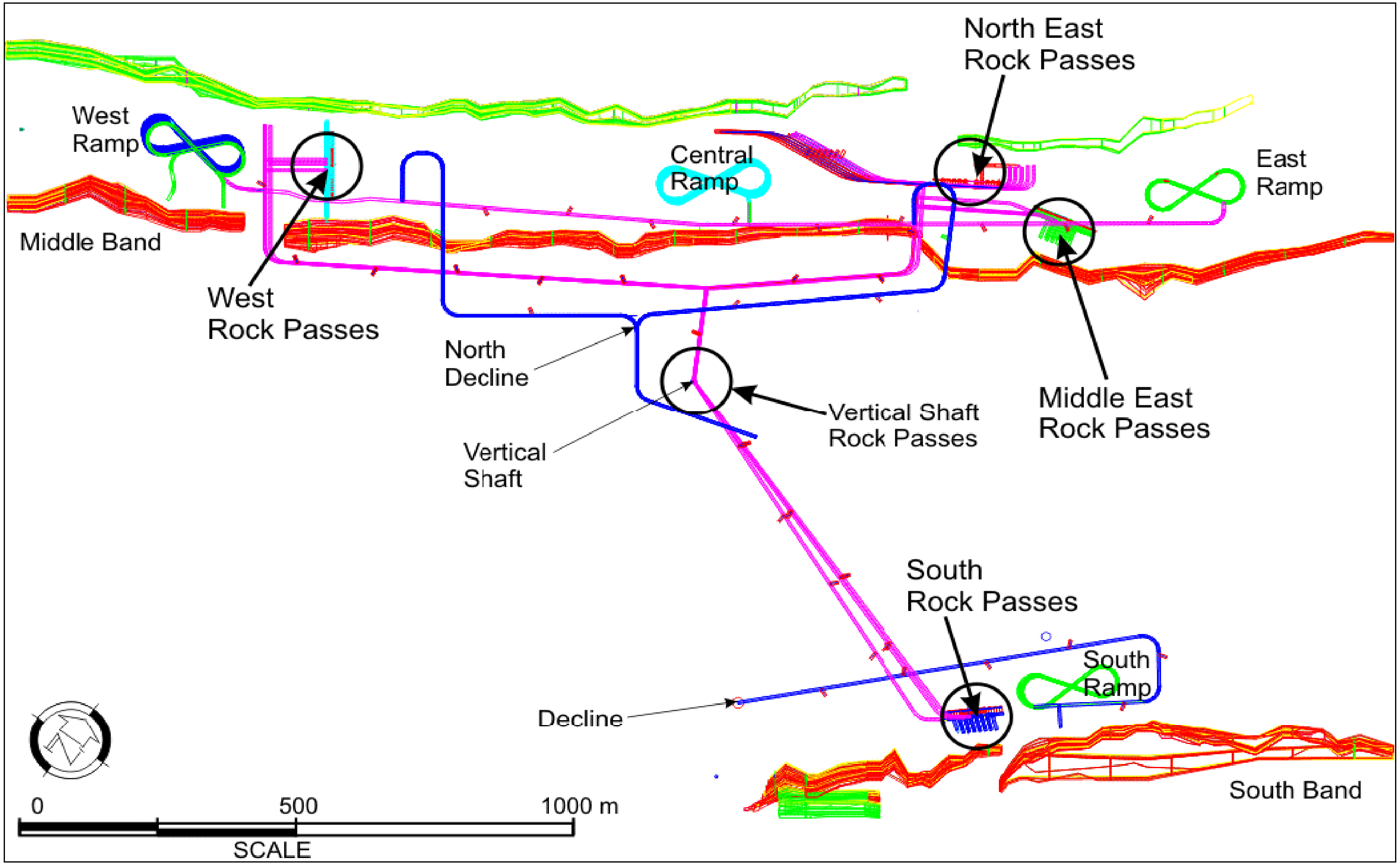

Fig. 4.17: Schematic Diagram showing the location of various Rock Passes

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REGN. NO. RQP/ BBS / 044 / 2003 / A

Proposed Extent of Development:

The extent of surface declines, shaft and connected level developments in waste rock required and proposed during the plan period is given in Table 4.8.4 below.

Year-wise Development:

The year-wise development proposed to be undertaken during the plan period is given below in Table 4.8.4 to 4.8.8.

The drawings showing the year-wise developments proposed to be undertaken during the plan period is enclosed as Drawing no.'s SCM/MP/17/12 and SCM/ MP/18/12

The drawing showing the vertical section of the developments to be undertaken during the plan period is enclosed as Drawing no. SCM/ MP/21/12.

Table 4.8.4
Level wise and Year wise Development of Main entries to the underground mine

Face/Heading	Start RL	Meterage up to 2017-18	2013-14*		2014-15		2015-16		2016-17		2017-18	
			Meterage	Level RL	Meterage	Level RL	Meterage	Level RL	Meterage	Level RL	Meterage	Level RL
Decline North	130	360	300	87	60	79						
Eastern limb	78	918			585	3	333	-35				
Western limb	78	831			410	19	421	-35				
Decline South	77	805	500	6	305	-35						
Eastern ramp	-35	667							217	-55	450	-96
Western ramp	-35	1583					740	-102	218	-122	625	-179
Central ramp	-35	1780					700	-99	218	-118	862	-195
South ramp	-35	1780			280	-60	780	-131	720	-195		
Vertical Shaft	133	365	100	33	265	-232						

* Starting from 12.01.2013

Table No. 4.8.5
North & Middle Band level wise and yearwise waste developments

Oreband	Level	Face/Heading	Meterage up to 2017-18	Period	2013-14		2014-15		2015-16		2016-17		2017-18	
				Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
NB/MB	Common	Decline	2109	52725	300	7500	1080	27000	729	18225				
		Eastern ramp	667	16675							217	5425	450	11250
		Western ramp	1583	39575					740	18500	218	5450	625	15625
		Central ramp	1780	44500					700	17500	218	5450	862	21550
		West ore pass	130	1225									130	1225
		Return airway for west ramp	80	251							40	126	40	126
		Return airway for central ramp	60	188							20	63	40	126
		Return airway for east ramp	60	188							20	63	40	126
		Northwest – Upcast(VS-1)	305	5292					180	3121	125	2168		
		Northeast – Upcast(VS-2)	350	6073					181	3139	169	2931		
		North Decline West – Upcast(VS-3)	145	1088	50	377	95	717						
		North Decline East – Upcast(VS-4)	182	1365	50	377	132	996						
		North East – Downcast(VS-5)	241	3181										
		North West– Downcast(VS-6)	205	2706										
		Temp Decline return airway(VS-7)	50	173	50	173								
		Vertical shaft	365	13651	100	3737	265	9904						
		Shaft connection drift(-95ML)	1980	49500			400	10000	400	10000	600	15000	580	14500
		Shaft connection drift(-195ML)	1980	49500					170	4250	1100	27500	710	17750
		Ventilation drive at -35ml	2469	61725					250	6250	1400	35000	819	20475
		Ramp connection drive	2205	55125					700	17500	920	23000	585	14625
		Exploration pockets	200	3200					40	640	80	1280	80	1280
		Sub total			**550**	**12165**	**1972**	**48617**	**4090**	**99125**	**5127**	**123454**	**4961**	**118656**

Table 4.8.5 (contd.)
North & Middle Band level wise and yearwise waste developments

Oreband	Level	Face/Heading	Meterage up to 2017-18	Period	2013-14		2014-15		2015-16		2016-17		2017-18	
				Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
MB	-55ML	Decline access	65	1625					65	1625				
		Footwall drift	624	15600					624	15600				
		Ore drive	291	4656					291	4656				
		Cross cuts(Waste)	125	2000					125	2000				
		Cross cuts(Ore)	10	160					10	160				
		Ore pass flat development	170	4250							168	4200		
		Level wise Waste(Meter)(Volume)			0	0	0	0	814	19225	168	4200		
		Levelwise ore(Meter)(Volume)			0	0	0	0	301	4816	0	0		
	-75ML	Decline access	57	1425							57	1425		
		Footwall drift	697	17425							697	17425		
		Ore drive	425	6800							425	6800		
		Cross cuts(Waste)	114	1824							114	1824		
		Cross cuts(Ore)	15	240							15	240		
		Ore pass flat development	210	5250										
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	868	20674		
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	440	7040		
	-95ML	Decline access	50	1250							50	1250		
		Footwall drift	936	23400							550	13750	386	9650
		Ore drive	609	9744							410	6560	199	3184
		Cross cuts(Waste)	260	4160							186	2976	74	1184
		Cross cuts(Ore)	30	480							24	384	6	96
		Ore pass flat development	210	5250									210	5250
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	786	17976	670	16084
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	434	6944	205	3280

Table 4.8.5 (contd.)
North & Middle Band level wise and yearwise waste developments

Oreband	Level	Face/Heading	Meterage up to 2017-18	Period	2013-14		2014-15		2015-16		2016-17		2017-18	
				Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
MB	-115ML	Decline access	205	5125									205	5125
		Footwall drift	1800	45000									1800	45000
		Ore drive	885	14160									885	14160
		Cross cuts(Waste)	310	4960									310	4960
		Cross cuts(Ore)	50	800									50	800
		Ore pass flat development	0	0										
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	0	0	2315	55085
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	0	0	935	14960
	-135ML	Decline access	120	3000									120	3000
		Footwall drift	800	20000									800	20000
		Ore drive	730	11680									730	4380
		Cross cuts(Waste)	670	10720									320	5120
		Cross cuts(Ore)	36	576									36	576
		Ore pass flat development	0	0										
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	0	0	1240	28120
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	0	0	766	4956

Table 4.8.5 (contd.)

North & Middle Band level wise and yearwise waste developments

Oreband	Level	Face/Heading	Meterage up to 2017-18	Period	2013-14		2014-15		2015-16		2016-17		2017-18	
				Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
NB	-75ML	Decline access	48	1200					48	1200				
		Footwall drift	590	14750					590	14750				
		Ore drive	324	5184					324	5184				
		Cross cuts(Waste)	115	1840					115	1840				
		Cross cuts(Ore)	25	400					25	400				
		Ore pass flat development	0	0					0	0				
		Levelwise Waste(Meter)(Volume)			0	0	0	0	753	17790				
		Levelwise ore(Meter)(Volume)			0	0	0	0	349	5584				
	-95ML	Decline access	80	2000							80	2000		
		Footwall drift	845	21125							845	21125		
		Ore drive	565	9040							565	9040		
		Cross cuts(Waste)	263	4208							263	4208		
		Cross cuts(Ore)	60	960							60	960		
		Ore pass flat development	0	0							0	0		
		Levelwise Waste(Meter)(Volume)			0	0	0	0	0	0	1188	27333		
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	625	10000		

Table 4.8.5 (contd.)
North Band level wise and yearwise ore and waste developments

Oreband	Level	Face/Heading	Meterage up to 2017-18	Period	2013-14		2014-15		2015-16		2016-17		2017-18	
				Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
NB	-115ML	Decline access	50	1250									50	1250
		Footwall drift	680	17000									680	17000
		Ore drive	580	9280									580	9280
		Cross cuts(Waste)	208	3328									208	3328
		Cross cuts(Ore)	20	320									20	320
		Ore pass flat development	0	0									0	0
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	0	0	938	21578
		Levelwise ore(Meter)(Volume)			0	0	0	0	0	0	0	0	600	9600
	-135ML	Decline access	50	1250									50	1250
		Footwall drift	900	22500									900	22500
		Ore drive	740	11840									740	11840
		Cross cuts(Waste)	260	4160									260	4160
		Cross cuts(Ore)	70	1120									70	1120
		Ore pass flat development	0	0									0	0
		Level wise Waste(Meter)(Volume)			0	0	0	0	0	0	0	0	1210	27910
		Level wise ore(Meter)(Volume)			0	0	0	0	0	0	0	0	810	12960
MB	**Total**	**Total Yearwise Waste(Meter)(Volume)**			**0**	**0**	**0**	**0**	**814**	**19225**	**1822**	**42850**	**4225**	**99289**
		Total Yearwise Ore in (Meter)(Tonne)			**0**	**0**	**0**	**0**	**301**	**15652**	**874**	**45448**	**1906**	**75387**
NB	**Total**	**Total Yearwise Waste(Meter)(Volume)**			**0**	**0**	**0**	**0**	**753**	**17790**	**1188**	**27333**	**2148**	**49488**
		Total Yearwise Ore in (Meter)(Tonne)			**0**	**0**	**0**	**0**	**349**	**19879**	**625**	**35600**	**1410**	**80313**
MB + NB	**Grand Total**	**Total Yearwise Waste(Meter)(Volume)**			**550**	**12165**	**1972**	**48617**	**5657**	**136140**	**8137**	**193637**	**11334**	**267433**
		Total Yearwise Ore in (Meter)(Tonne)			**0**	**0**	**0**	**0**	**650**	**35531**	**1499**	**81048**	**3316**	**155701**

Table 4.8.6

South Band level wise & yearwise ore and waste developments

Level	Face/heading	Meterage upto 2017-18	Period	2012-13		2013-14		2014-15		2015-16		2016-17		2017-18	
			Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
	Decline	1021	25525	300	7500	500	12500	221	5525						
	Shaft connection -95ml	820	20500											820	20500
	Ramp	1780	44500					280	7000	780	19500	720	18000		
	Ore pass	184	1733							80	754	104	980		
Common	Return airway for South Ramp	155	487							40	126	115	361		
	East Return airway, South band(VS-8)	390	5148							106	1399	200	2640	84	1108.8
	West return airway, South band(VS-9)	320	4224							100	1320	195	2574	25	330
	Decline upcast shaft, South band(VS-10)	140	484			140	484.4								
	Ventilation drift at -35ml	1285	32125					720	18000	565	14125				
	Sub total	**6095**	**134726**	**300**	**7500**	**640**	**12984**	**1221**	**30525**	**1671**	**37223**	**1334**	**24555**	**929**	**21939**
-35ML	Decline access	52	1300					52	1300						
	Footwall drift	1268	31700					380	9500	888	22200				
	Ore drive	1087	17392					314	5024	773	12368				
	Cross cuts(Waste)	398	6368					208	3328	190	3040				
	Cross cuts(Ore)	58	928					20	320	38	608				
	Ore pass flat development	148	3700											148	3700
	Level wise Waste(Meter)(Volume)	**1866**	**43068**	**0**	**0**	**0**	**0**	**640**	**14128**	**1078**	**25240**	**0**	**0**	**148**	**3700**
	Level wise ore(Meter)(Volume)	**546**	**10068**	**0**	**0**	**0**	**0**	**208**	**3328**	**811**	**3040**	**0**	**0**	**148**	**3700**

Table 4.8.6 (contd.)
South Band level wise & yearwise ore and waste developments

Level	Face/heading	Meterage upto 2017-18	Period	2012-13		2013-14		2014-15		2015-16		2016-17		2017-18	
			Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
-55ML	Decline access	45	1125							45	1125				
	Footwall drift	1288	32200							910	22750	378	9450		
	Ore drive	1302	20832							405	6480	897	14352		
	Cross cuts(Waste)	453	7248							241	3856	212	3392		
	Cross cut (Ore)	59	944							35	560	24	384		
	Ore pass flat development	148	3700											148	3700
	Level wise Waste(Meter)(Volume)	**1934**	**44273**	**0**	**0**	**0**	**0**	**0**	**0**	**1196**	**27731**	**590**	**12842**	**148**	**3700**
	Levelwise ore(Meter)(Volume)	**1361**	**21776**	**0**	**0**	**0**	**0**	**0**	**0**	**440**	**7040**	**921**	**14736**	**0**	**0**
-75ML	Decline access	52	1300							52	1300				
	Footwall drift	1285	32125									1285	32125		
	Ore drive	1317	21072									1317	21072		
	Cross cuts(Waste)	425	6800									425	6800		
	Cross cuts (Ore)	58	928									58	928		
	Ore pass flat development	148	5476											148	3700
	Level wise Waste(Meter)(Volume)	**1910**	**45701**	**0**	**0**	**0**	**0**	**0**	**0**	**52**	**1300**	**1710**	**38925**	**148**	**3700**
	Level wise ore(Meter)(Volume)	**1375**	**22000**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1375**	**22000**	**0**	**0**

Table 4.8.6 (contd.)
South Band level wise & yearwise ore and waste developments

Level	Face/heading	Meterage upto 2017-18	Period	2012-13		2013-14		2014-15		2015-16		2016-17		2017-18	
			Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume	Meterage	Volume
-95ML	Decline access	45	1125									45	1125		
	Footwall drift	1280	32000									571	14275	709	17725
	Ore drive	1312	20992									54	864	1258	20128
	Cross cuts(Waste)	530	8480									239	3824	291	4656
	Cross cut(Ore)	58	928									10	160	48	768
	Ore pass flat development	0	0												
	Level wise Waste(Meter)(Volume)	**1855**	**41605**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**855**	**19224**	**1000**	**22381**
	Levelwise ore(Meter)(Volume)	**1370**	**21920**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**64**	**1024**	**1306**	**20896**
-115ml	Decline access	52	1300											52	1300
	Footwall drift	1279	31975											1279	31975
	Ore drive	930	14880											930	14880
	Cross cuts(Waste)	487	7792											487	7792
	Cross cut(Ore)	35	560											35	560
	Ore pass flat development	0	0												
	Level wise Waste(Meter)(Volume)	**1818**	**41067**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1818**	**41067**
	Levelwise ore(Meter)(Volume)	**965**	**15440**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**965**	**15440**
-135ML	Decline access	48	1200											48	1200
	Footwall drift	210	5250											210	5250
	Ore drive	0	0												
	Cross cuts(Waste)	62	992											62	992
	Cross cut(Ore)	0	0												
	Ore pass flat development	0	0												
	Level wise Waste	**320**	**7442**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**320**	**7442**
	Level wise ore	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
	Total Yearwise Waste (Meter)(Volume)			**300**	**7500**	**640**	**12984**	**1861**	**44653**	**3997**	**91494**	**4489**	**95546**	**4511**	**103929**
	Total Year wise Ore in (Meter)(Tonne)			**0**	**0**	**0**	**0**	**334**	**18704**	**1251**	**70056**	**2360**	**132160**	**2271**	**127176**

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Table 4.8.7
Yearwise and level wise extent of development Declines and Ramps

Decline & Ramps	Start RL	Meterage up to 2017-18	2013-14		2014-15		2015-16		2016-17		2017-18	
			Meterage	Level RL	Meterage	Level RL	Meterage	Level RL	Meterage	Level RL	Meterage	Level RL
Decline North	130	2109	300	87	60	79						
Eastern limb	78	918			585	3	333	-35				
Western limb	78	831			410	19	421	-35				
Decline South	77	805	500	6	305	-35						
Eastern ramp	-35	667							217	-55	450	-96
Western ramp	-35	1583					740	-102	218	-122	625	-179
Central ramp	-35	1780					700	-99	218	-118	862	-195
South ramp	-35	1780			280	-60	780	-131	720	-195		

Table 4.8.8
Summary of Year wise Developments

Ore band	**Headings**	**Development in meters**						
		2012-13	**2013-14**	**2014-15**	**2015-16**	**2016-17**	**2017-18**	
North & Middle band	Decline & ramps		300	1080	2169	653	1937	
	Ventilation shafts & raises		150	227	361	374	120	
	Common waste developments			400	1560	4100	2822	
	Level waste developments				1567	3010	6373	
	Level ore developments				650	1499	3316	
Southern band	Decline & ramps	300	500	501	780	720	820	
	Ventilation shafts & raises		140		246	510	109	
	Common waste developments			720	645	104		
	Level waste developments			640	2326	3155	3552	
	Level ore developments			334	1251	2360	2271	
Main shaft			100	265				
Total		300	1190	4167	11555	16485	21320	55017

Support System:

The requirement for ground supports in the proposed mine excavations was assessed using the Q Tunneling Index (Barton), UNWEDGE analyses and MRMR (Laubscher) assessment.

Six classes of ground support are proposed to cover the range of strata conditions that shall be encountered in the course of development of the mine and stopes. Each of these 6 classes represents a group of material and rock types that shall behave in a similar manner once exposed to the mining environment. They are in effect geotechnical design domains and the different mine sections shall be so classified during the course of development of the mine. Table 4.8.16 given below describes these 6 classes of support and its relationship with Q.

Table 4.8.16
Geotechnical and Support Classes

Geotechnical Description	Support Class	Q, Barton Index
Soil – like Material	1	<0.1
Very Poor Quality Deformable Material	2	0.1-0.4
Very Poor Quality Rock	3	0.4-1.0
Poor Quality Rock	4	1.0-4.0
Fair Quality Rock	5	4.0-10.0
Good to Very Good Quality Rock	6	>10.0

The ground support system for development is to be categorized as follows:

- Permanent mine openings: access decline, haulage drives, waste cross cuts to stopes;
- Temporary mine openings: all development within the ore body.

Further details of the systematic support design proposed for the Permanent & Temporary Mine Openings is given in Table 4.8.17 & 4.8.18 respectively.

Table 4.8.17
Permanent Development Support Design

Support Class	Q Values Range	Support Elements	Dimension			
			4 to 5m Tunnel		6 to 8m intersection	
			Walls	Roof	Walls	Roof
1	<0.1	FRS	100mm	150mm	100mm	150mm
		1.8m BS	1.2m		1.2m	
		2.4m BS		1.0m		1.0m
		Steel Mesh	Yes	Yes	Yes	Yes
		4m CBS	1.5m		1.5m	
		6m CBS		1.5m		1.5m
2	0.1-0.4	FRS	60mm	80mm	80mm	100mm
		1.8m BS	1.5m		1.5m	
		2.4m BS		1.3m		1.3m
		Steel Mesh				
		4m CBS	1.5m		1.5m	
		6m CBS		1.5m		1.5m
3	0.4-1.0	FRS	30mm	60mm	50mm	80mm
		1.8m BS	1.8m		1.8m	
		2.4m BS		1.5m		1.5m
		Steel Mesh				
		4m CBS				
		6m CBS				
4	1.0-4.0	FRS	nil	30mm	nil	50mm
		1.8m BS	2.0m	1.8m	2.0m	1.8m
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				
5	4.0-10.0	FRS	nil	nil	nil	30mm
		1.8m BS	nil	2.1m	nil	2.1m
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				
6	>10.0	FRS	nil	nil	nil	nil
		1.8m BS	nil	2.1m	nil	2.1m
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				

Where: FRS = Fibre Reinforced Shotcrete; BS = Bolt Spacing; CBS = Cable Bolt Spacing;

Table 4.8.18

Temporary Development Support Design

Support Class	Q Values Range	Support Elements	Dimension			
			4 to 5m Tunnel		6 to 8m intersection	
			Walls	Roof	Walls	Roof
1	<0.1	FRS	80mm	100mm	80mm	100mm
		1.8m BS	1.2m		1.2m	
		2.4m BS		1.0m		1.0m
		Steel Mesh	Yes	Yes	Yes	Yes
		4m CBS				
		6m CBS				
2	0.1-0.4	FRS	40mm	60mm	40mm	60mm
		1.8m BS	1.5m		1.5m	
		2.4m BS		1.3m		1.3m
		Steel Mesh				
		4m CBS				
		6m CBS				
3	0.4-1.0	FRS	Nil	Nil	Nil	30mm
		1.8m BS	Nil	1.8m	Nil	1.8m
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				
4	1.0-4.0	FRS	Nil	Nil	Nil	Nil
		1.8m BS	Nil	2.1m	Nil	2.1m
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				
5	4.0-10.0	FRS	Nil	Nil	Nil	Nil
		1.8m BS	Nil	Nil	Nil	Nil
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				
6	>10.0	FRS	Nil	Nil	Nil	Nil
		1.8m BS	Nil	Nil	Nil	Nil
		2.4m BS				
		Steel Mesh				
		4m CBS				
		6m CBS				

Where: FRS = Fibre Reinforced Shotcrete; BS = Bolt Spacing; CBS = Cable Bolt Spacing;

IV) Method and Sequence of Stoping:

The method of stoping to be adopted for extraction of the ore in general is governed by the type of ore body (width, depth, dip, strike length, direction etc.), other characteristics (strength, joint sets pattern, joint infill etc.) of both the ore and adjoining strata as well as the occurrence of other major disturbances (faults, shear zones, dykes etc.). Detailed geotechnical studies have been undertaken by the Company with the assistance of expert agencies & geotechnical engineers over an extended period of time to establish the applicable stoping method.

While the geology of the orebody with respect to its dimensions, mode of occurrence, dip and strike directions and resources has been covered under the chapter on Geology, a brief description of the various types of chrome ore and the wall rocks/ strata conditions to be encountered at greater depths in the mine is given below.

Rock Types:

The various rock types that were identified by the geology group and logged as per the rock type codes are as follows:

- HUB: Hard Ultrabasics, category does not recognize metamorphic influences, unweathered grey rock
- SUB: Soft Ultrabasics, weathered HUB, rock
- LIM: Limonite, Completely weathered SUB, soil-like
- CHT: Chert, locally overprinted on limonite to varying degrees, soil/ soft rock
- DYKE: Dolerite
- CRM: Chromite ore, does not distinguish weathering state

The material can be distributed into three broad groups as follows:

- STRONG :Chrome (lumpy), Dyke ('hard') and HUB
- INTERMEDIATE : Chrome (mixed), SUB and Dyke ('soft')
- WEAK :Chrome (friable) and the limonite-chert (soil)

The country rock in the vicinity of the orebodies at greater depths considered to be taken up for underground mining consists mainly of HUB. In the North and Middle ore bands the material near surface, and in close proximity to the ore at depth, has been preferentially weathered. A photograph to illustrate the progressive weathering from the deep seated HUB to SUB to LIM soil nearer the surface is given above under para 3 in Fig. 3.1.

The HUB is partly or fully altered to serpentinite or talcose schist. The HUB in the vicinity of the North and Middle Ore Bands is generally much less altered than the HUB in the South Band. This is probably related to the presence of the shear zones in the north and south walls of the South Band. A photograph to illustrate the alteration of the HUB is given above in Fig. 3.2.

Dykes:

Dolerite dyke intrusions crosscut the three ore bands and offset them. These dykes vary in character from strong, weathered and structureless, to weak, highly weathered and structured. The dyke widths vary between 15 - 75m and this area is barren. The compartments defined by the dykes vary between 250 - 600m in strike length. These dykes offset the chromite bands in a step-wise fashion on to the south by generally 20 - 30m, upto 130m in the eastern end of the Middle Band. A photograph to illustrate the different dolerite types encountered in the area is given above in Fig.3.3.

Chromite Ore types:

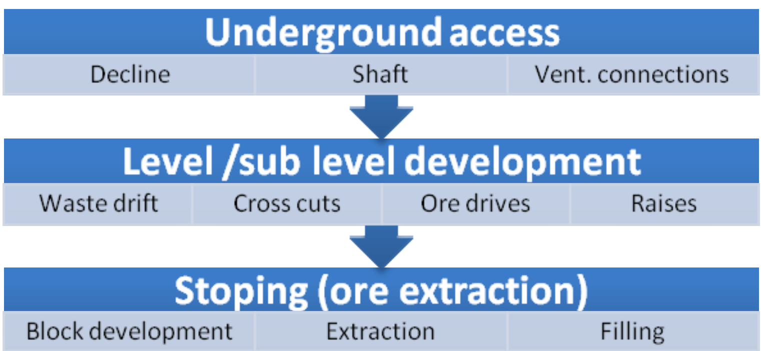
The chromite ore materials can be categorized into three groups, based on weathering state. The materials are described further as “friable”, “mixed” or “lumpy”, and represent progressively less weathered states (and more siliceous) states.

The friable material is a very soft rock (upto 5 MPa intact rock strength) or soil material. It can occur as reddish or black, silty, slightly clayey sand, with minor amounts of cohesion. Such friable ore disintegrates with a single, light hammer blow. The upper chrome orebody in the North & Middle Bands are friable and the distribution with depth is associated with the distinctive weathering cone that extends down to ~ 150 - 300m below ground surface. This material will act as a conduit for ground and surface water infiltration. Developing any type of underground mine excavation within this material will be extremely hazardous and should be avoided, if at all possible.

The mixed material is the transitional horizon between the soft-friable and hard lumpy ore. Core intersections exhibit both characteristics in alternating bands. Underground mining within the mixed should be possible, provided that the support is installed within the mining cycle. The lumpy is slightly weathered to unweathered rock. The lumpy is penetrated by faults and dykes, and some talcose alterations are occasionally present. A photograph to illustrate the three types of chromite ore as mentioned above under para 3 is given in Fig. 3.4.

Method of Mining and Stoping:

Broadly the mining operation can be divided into two categories, i.e. development and stoping. The process flow chart for underground mining is given below.



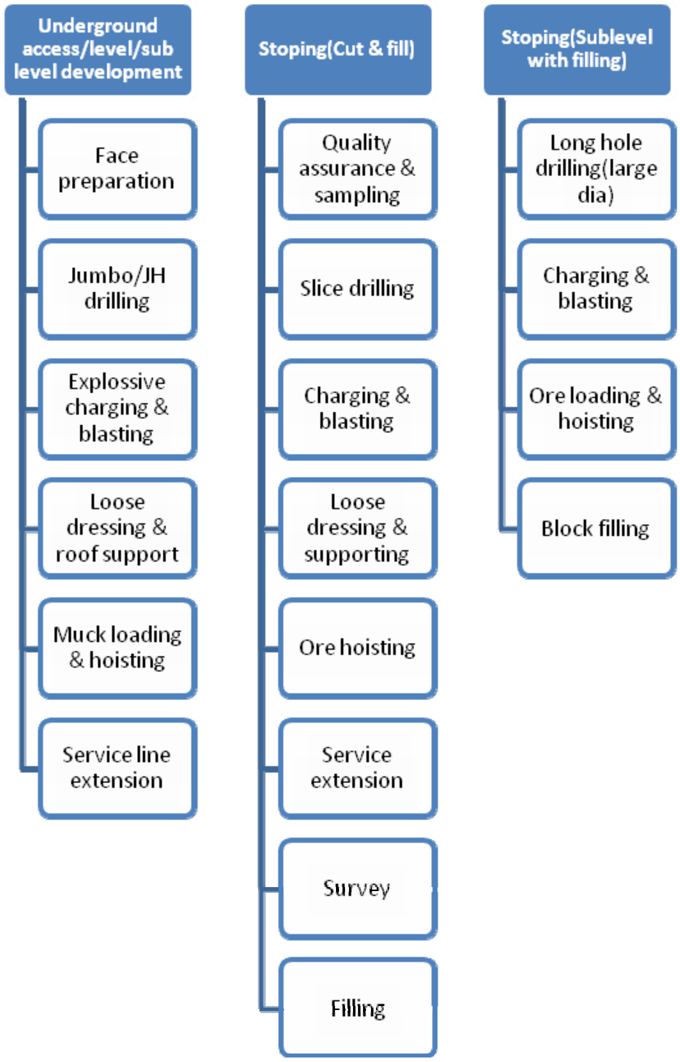
Based on geotechnical considerations as well as numerical analysis the method of stoping that is proposed to be adopted in the respective orebodies are outlined in Table 4.8.19.

Table 4.8.19
Applicable Stoping Method

Orebody	Ore Type	Weathering Class	Practical Width of Ore Band (m)	Corresponding Practical Q' Value	Proposed Mining Method	Preferred Stope Dimensions (m)		
						Width	Stope Strike Length	Stope Height
North Band	Friable	CW	10 – 20	< 0.1	UG Development not suggested			
	Mixed, some lumpy	MW -CW, Some SW,		1 – 4	Transverse Cut & Fill	5	10- 20	5
	Lumpy	MW -SW		5 – 10	Sub Level Open Stoping with backfill, with some areas of Cut & Fill	10	10	20
Middle Band	Friable	CW	4 – 8	< 0.1	UG Development not suggested			
	Mixed	MW - CW		0.4 – 1	Longitudinal Cut & Fill	4	110	4
	Lumpy	UW-MW		40 – 50	Narrow Open Stopes with back fill or pillars	4	10	20
South Band	Lumpy	UW	8 – 10	10 – 15	Sub Level Open Stoping with backfill, with some areas of Cut & Fill	10	10	20

Where: CW = Completely Weathered; MW = Moderately Weathered; SW = Slightly Weathered; UW = Unweathered; Q' = Modified Rock Quality Index = (RQD / J_n) x (J_r / J_a); RQD = Rock Quality Designation; J_n = No. of joint sets; J_r = Roughness of joint surfaces; J_a = Alteration, coating or infilling of joint planes

The performance with the above stope dimensions during initial mining phases shall be monitored and adjustments shall be accordingly made for future stoping areas. The sequence of operations involved in the process of development and stoping is in the chart given below.



Sub-level Open Stoping:

The sub-level open stoping and its many derivatives, is relatively flexible in that it can accommodate varying widths down to a minimum of 2 m. The maximum mineable width is dependent on the rock mass characteristics of the ore body and the surrounding host rock. The typical schematic layout of the Sub-level Open Stoping method is given in Fig. 4.18. Typically the development and stoping sequence in Sub-level stopes is given in Fig. 4.19.

A derivative of the conventional sub-level open stoping method with backfill is proposed to cater for the variable ground conditions in the ore bodies. The modified method is sub-level open stoping with post backfill. The sub-levels are 20m apart and crosscuts are spaced at 110m. There is a sill pillar to protect the stopes every 100m, this gives a stope that is 110m wide and 80m high. The crosscut access to the stope is protected by a 10m rib pillar. Fig. 4.20 shows the layout of the stope blocks bounded by sill pillars, the crosscuts from the footwal access development, the pillar protecting the crosscut into the ore body and the in-stope development. The proposed pillar and block dimensions are given in Fig. 4.21. The pillar dimensions have been proposed based on numerical modeling and analysis.

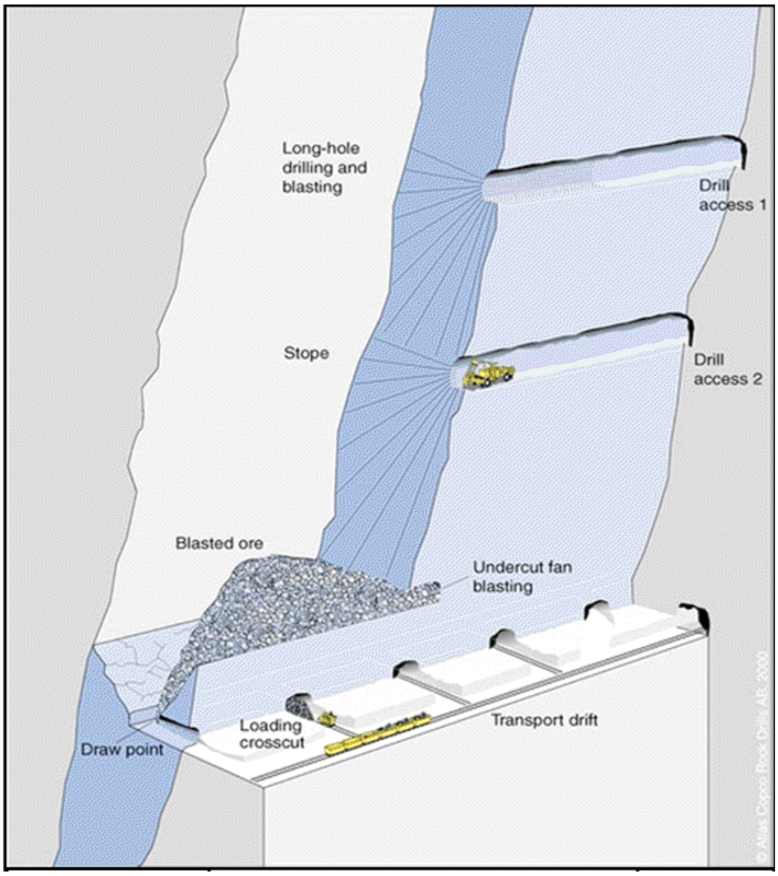


Fig. 4.18: Typical layout for Sub-level Open Stoping Method

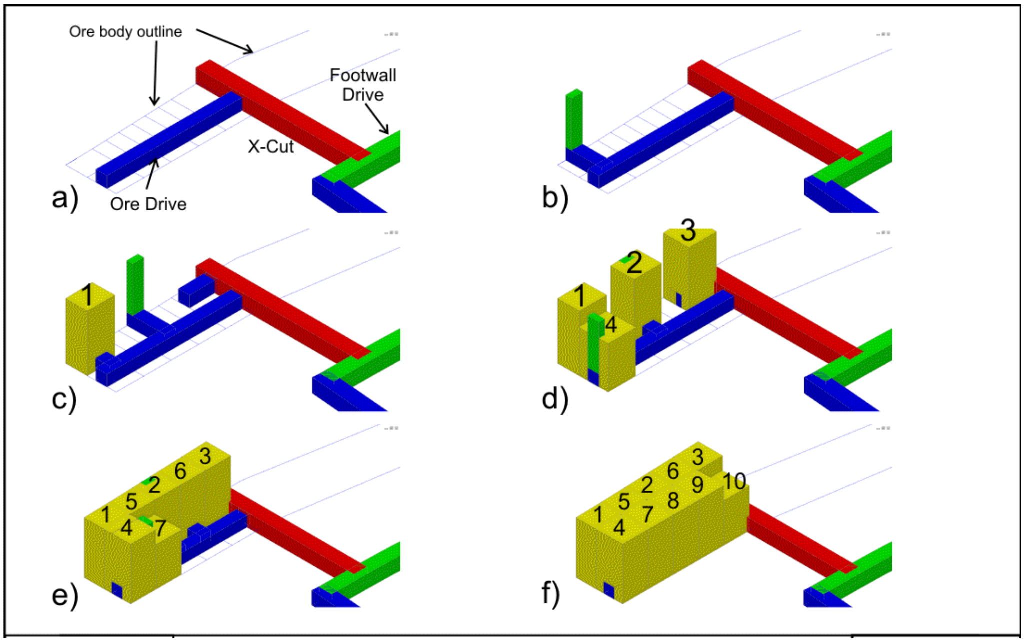


Fig. 4.19: Typical - Development and stoping sequence in Sub level stopes

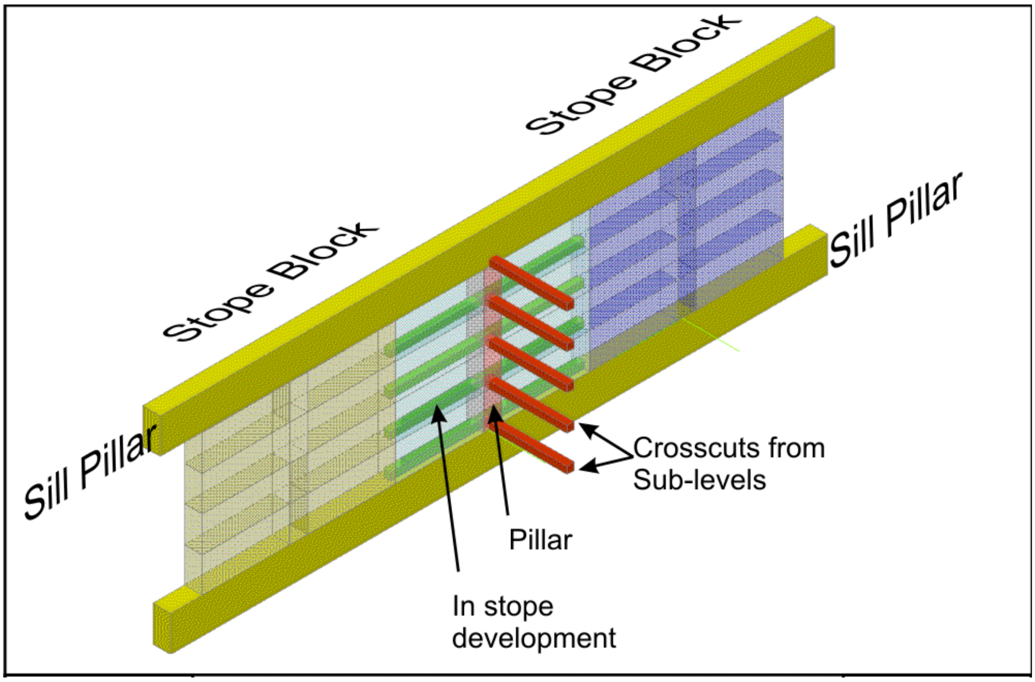


Fig. 4.20: Sublevel stoping with post fill layout with sill pillars

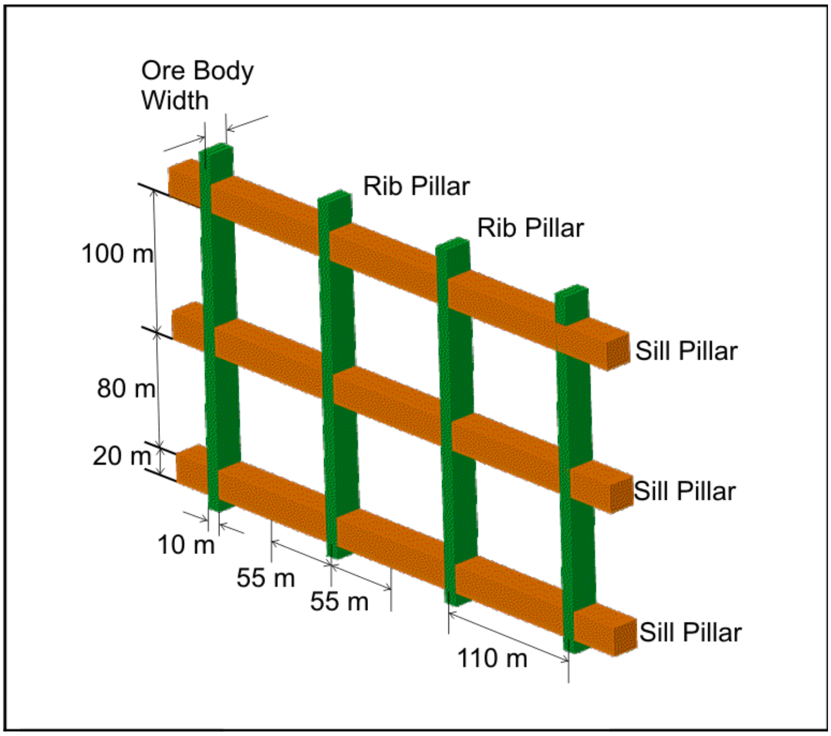


Fig. 4.21: Proposed pillar and block dimensions for sub level open stoping

Sequence of Sub-level Open Stoping with post filling:

The stope block shall be mined out from the “bottom-up”, as a panel between sub-levels is mined and filled; the fill is then used as a platform for the above panel to be mined. Before the panel is mined the in-stope development, on the top and bottom, to the limits of the block must be completed. A raise or slot is developed up from the bottom ore drive to the top ore drive. This slot provides a free-breaking face for the ring blasting and ventilation. Holes shall be drilled from the bottom drive to the top drive so that a 2 m advance is blasted. The broken rock shall be cleaned/ mucked with remote controlled load haul dump (“LHD”) vehicles. The drill and blast operations shall be continued in a retreating fashion until the correct dimensions are reached. Fig. 4.22 shows the mining sequence that shall be followed to start extraction.

The mined out block shall be filled from the top drive, and when the fill has cured a new slot is developed and the operation repeated. In similar operations, a removable polystyrene-type shuttering is left in the fill. Once the fill is cured and the polystyrene removed, the resultant void is used as the slot for the next lift.

The crosscuts shall be centred in the mining block and extraction carried out on either side of the crosscut. This means that two blocks, on either side of the crosscut shall be mined simultaneously.

Fig. 4.23 shows the sequence of extraction that is proposed to be adopted in case of narrow ore bands having width about or less than 10 m. In wider portion of the ore band more parallel stope drives would be taken and strike parallel stope blocks shall be formed and extracted as shown in Fig. 4.24.

A longitudinal section showing the proposed pillar (crown/sill and rib) dimensions, stope sizes and sequence of extraction is shown in the Fig. 4.25.

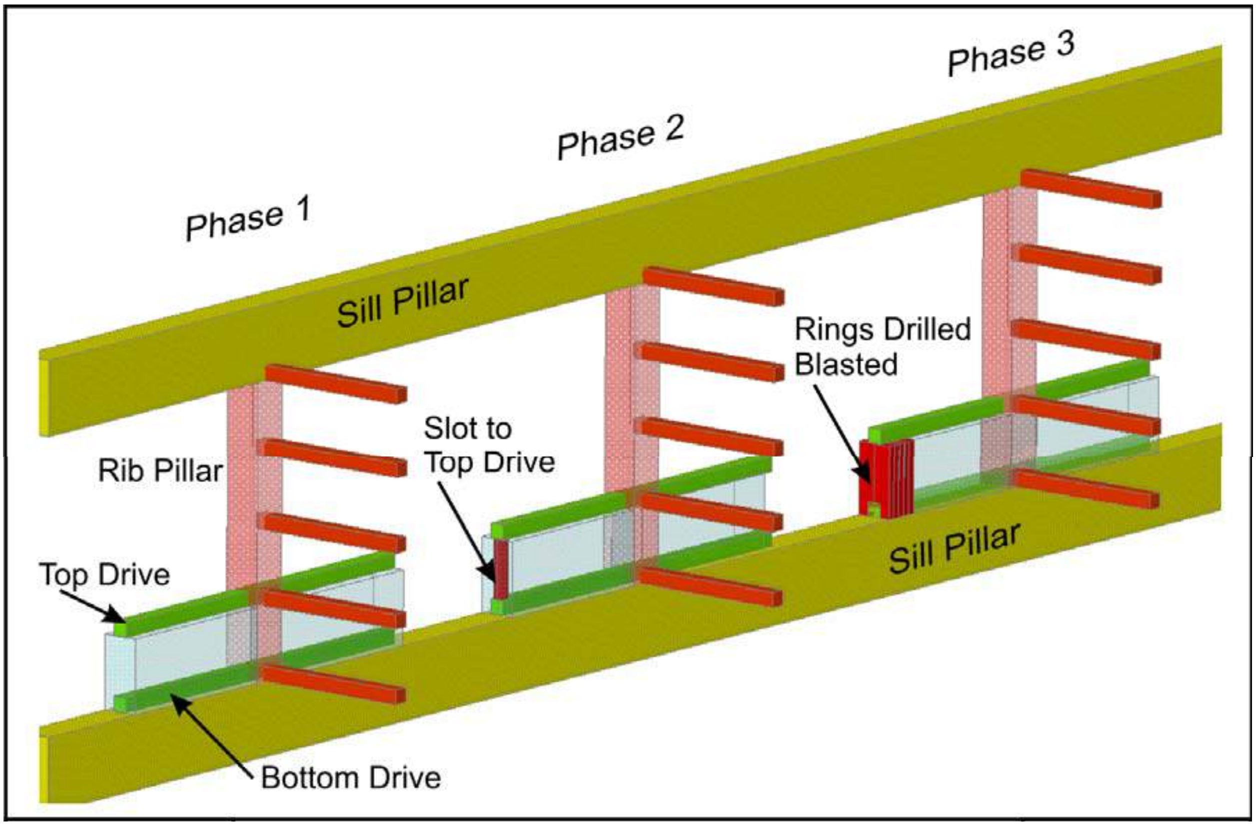


Fig. 4.22: Sublevel open stoping with post fill development and mining sequence

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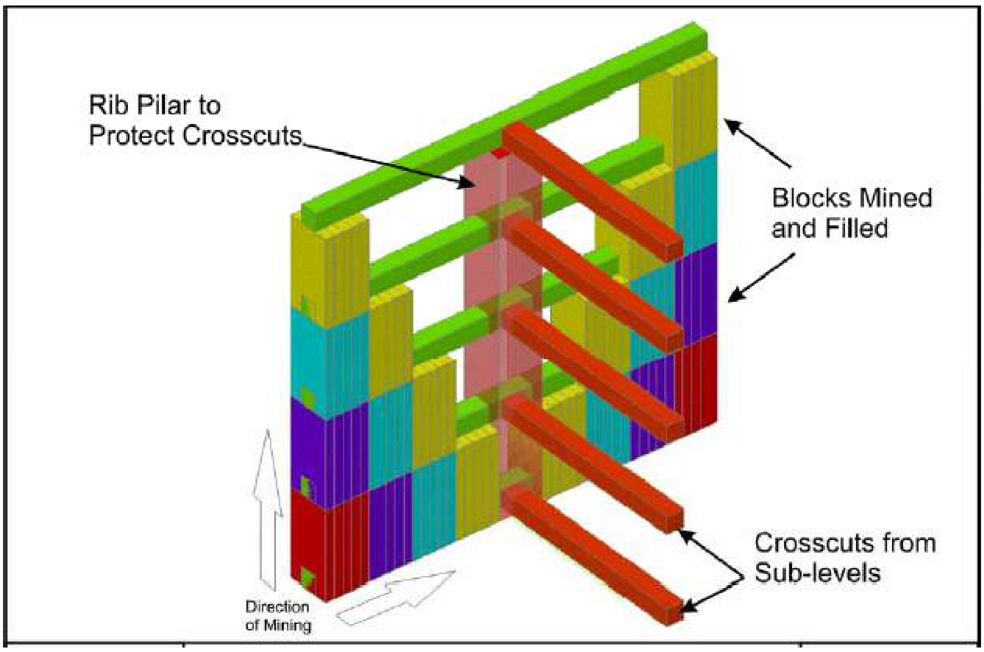

Fig. 4.23: Sub level open stoping sequence in Narrow Ore Body

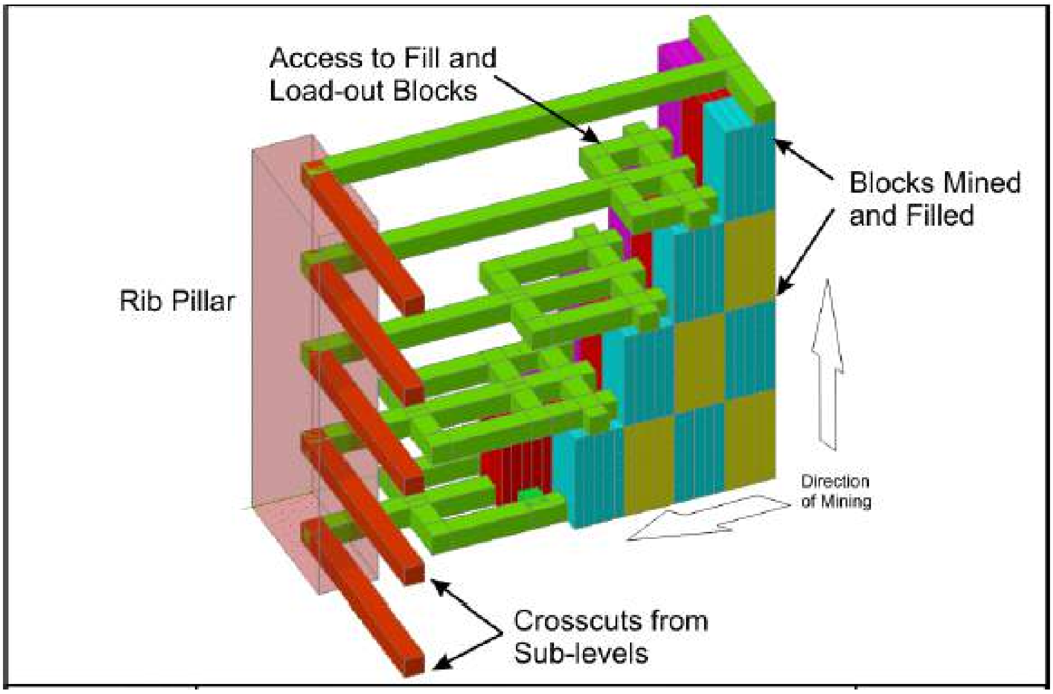

Fig. 4.24: Mining Sequence in Wide Ore Body

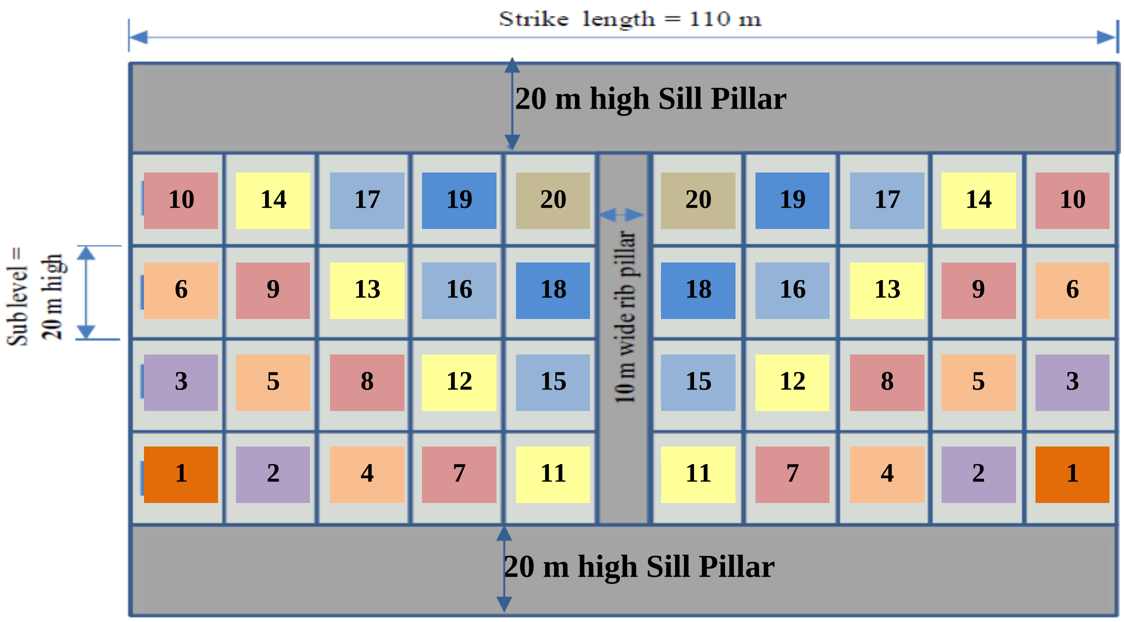

Fig. 4.25: Longitudinal Section of the Sub level stoping sequence in a block

Cut and Fill Method:

The cut and fill mining method is based on longitudinal drifts mined on strike. The drifts are 5 m wide x 5 m high and the length dependent of distance apart of the access crosscuts. Fig. 4.26 shows the transverse section of a typical drift and fill mining with long anchor support for the hanging wall. Access to the ore body is from ramps and the ore is mined in horizontal cuts and carried out in a “bottom-up” sequence. The drift is filled and mining always takes place on top of the fill material.

Drift and fill mining is generally used in rock conditions that do not allow the opening up of the stope to the full width of the ore body. Drift and fill can be mined with hand-held or mechanized equipment.

A variation of Cut and Fill is the Ramp in Stope, which is proposed to be adopted. The mining and filling is carried out in a number of stages and the sequence of operations is shown in Fig. 4.27 and Fig. 4.28.

A sill drive shall be first developed along the strike length of the stope panel with a ventilation raise to the level above (Fig. 4.27 - 1). A ramp is started by building a waste rock pile across the width of the stope. The next lift is started to continue the ramp and then mined horizontally. The lift is completed by filling to allow the next cut to be taken (Fig. 4.27 - 2). This process is repeated until the level above is reached (Fig. 4.27 – 3 & 4). The second phase is then started (Fig. 4.28) with a ventilation raise at the other end of the panel. The broken ore is removed to the top of the panel rather than to the bottom as in the first phase of stoping.

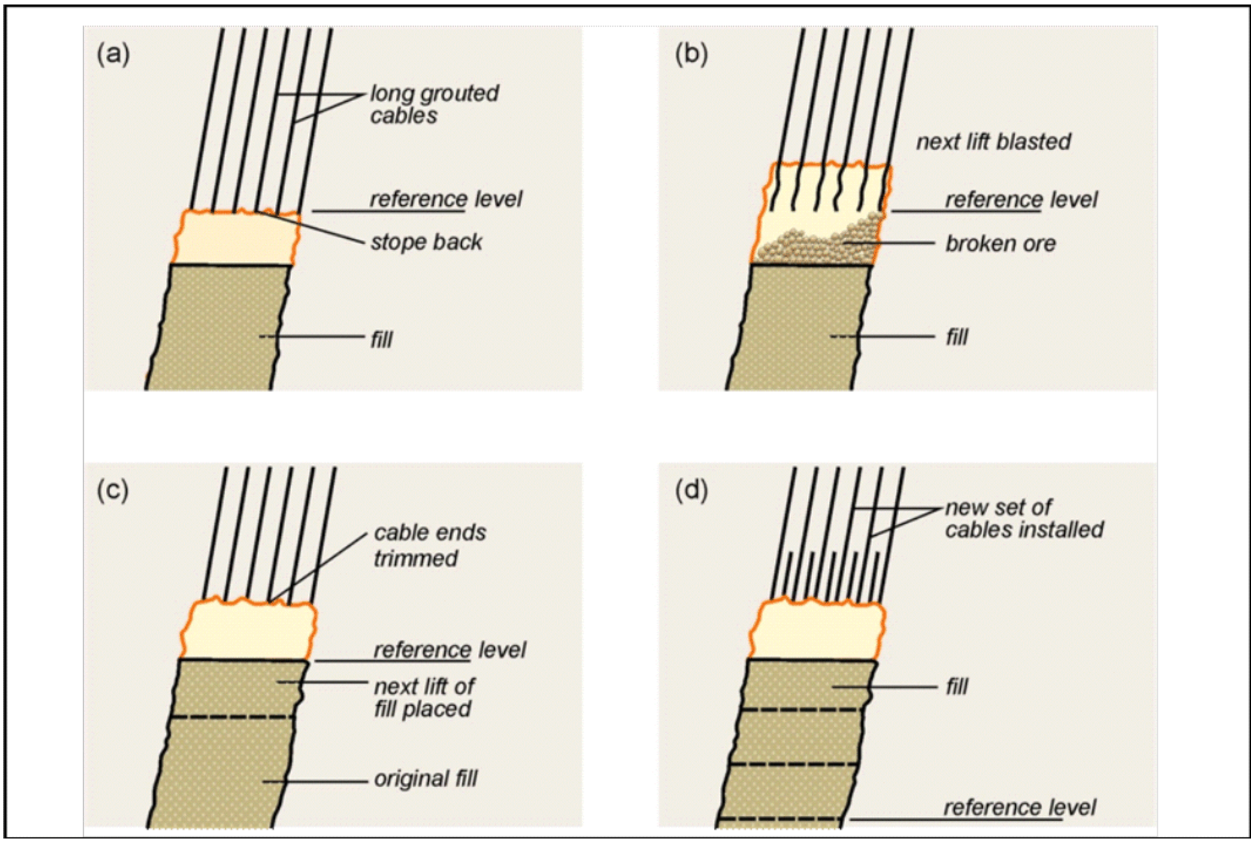

Fig. 4.26: Mining sequence in typical Cut & Fill Method in poor ground

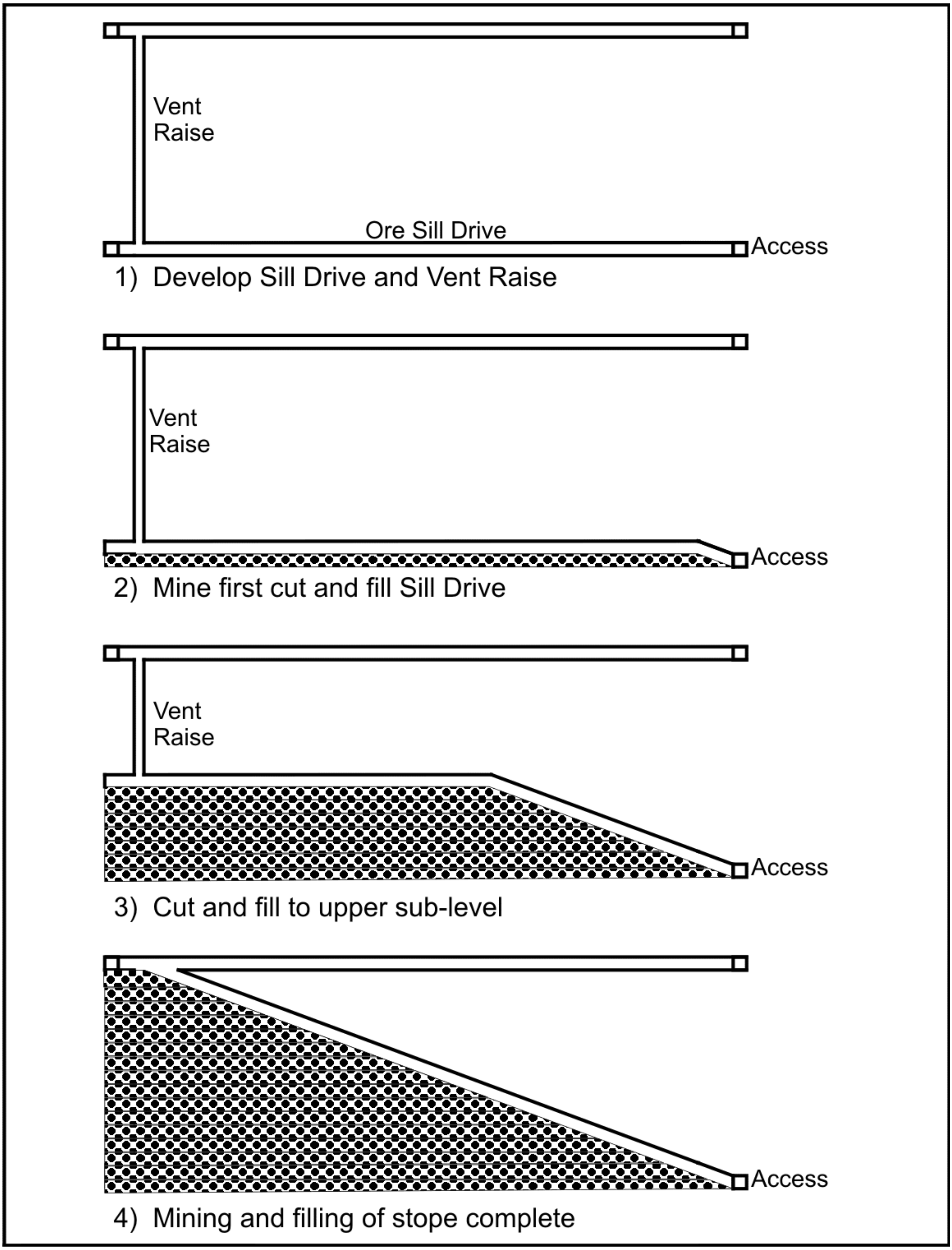

Fig. 4.27: Cut and Fill – Ramp in Stope (sequence of operation): Phase 1

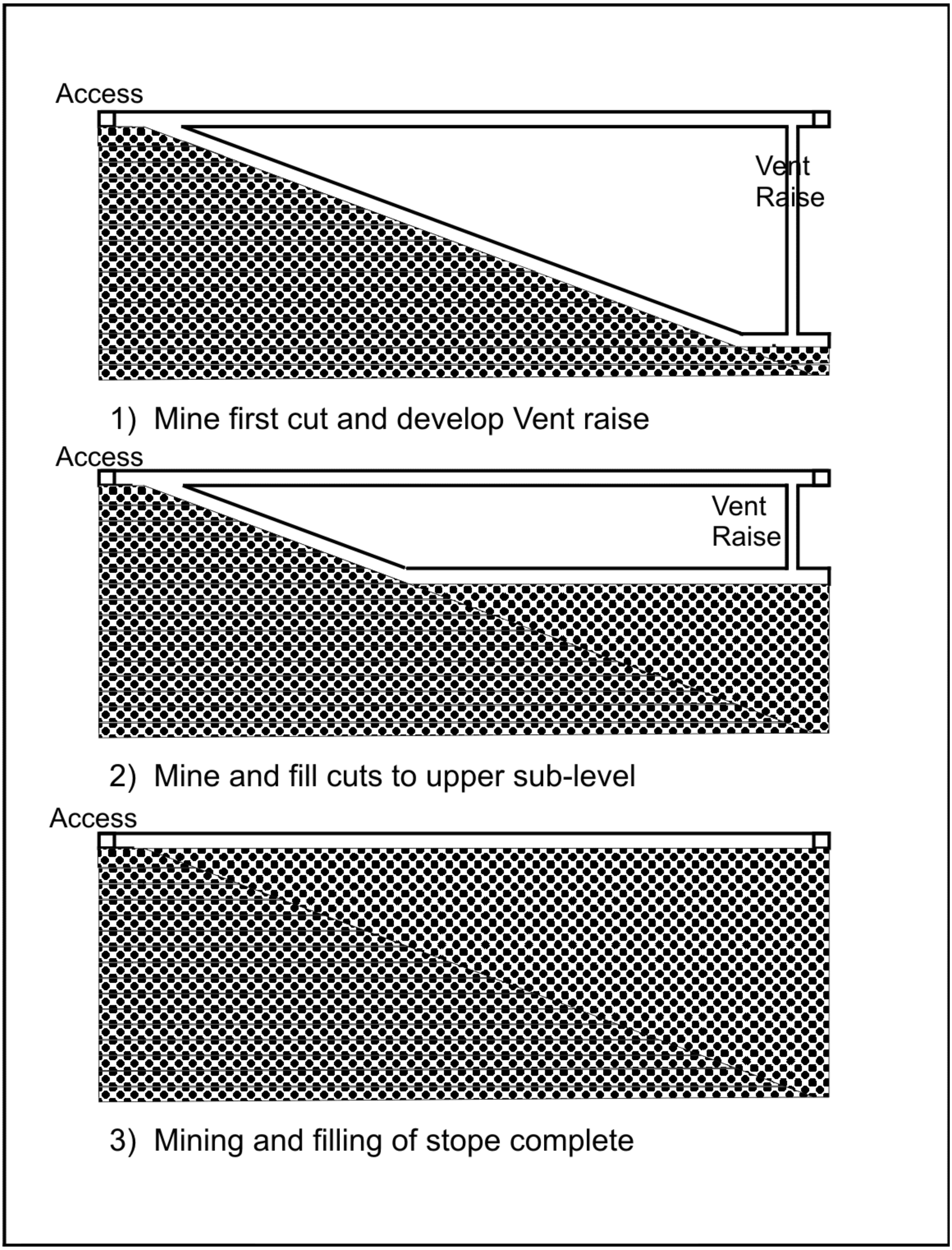

Fig. 4.28: Cut and Fill – Ramp in Stope (sequence of operation): Phase 2

The stoping operations are proposed to commence from -95 mRL and move upwards and along the strike direction in all three orebodies. The detailed band wise stoping program, sublevel wise ore reserve for stoping, number of complete blocks and blocks that can be prepared and need to be extracted is given in the Table no. 4.8.9.

Table No. 4.8.9

Ore band	Level	Proved & probable reserve in tonnes	Estimated Production per block, tonnes	No of blocks	Proposed Stoping Sequence	No of blocks	No. of Stope Blocks		Production from Stopes, tonnes	
							2016-17	2017-18	2016-17	2017-18
Northern band	-95 ML to -75ML	396819	7120	56	Stope preparation	56	25	31	85440	178000
					Stoping	37	12	25		
Middle band	-95ML to -75ML	209899	6500	32	Stope preparation	37	19	18		201500
					Stoping	31		31		
	-75ML to -55ML	167210		26	Stope preparation	29	15	14		
					Stoping					
Southern band	-95ML to -75ML	551138	7000	79	Stope preparation	72	44	28	140000	182000
					Stoping	46	20	26		

Sub-level wise Stoping Sequence & Production

The yearwise ROM chrome ore production from the underground workings includes the ore from development and stoping, the details of which is given in Table 4.8.10 below.

Table No. 4.8.10
Summary of ROM production from the underground workings

Ore band	Production From	Tonnage			
		2014-15	2015-16	2016-17	2017-18
Northern band	Development		19879	35600	80313
	Stoping			85440	178000
	Total	**0**	**19879**	**121040**	**258313**
Middle band	Development		15652	45448	75387
	Stoping				201500
	Total	**0**	**15652**	**45448**	**276887**
Southern band	Development	18704	70056	132160	127176
	Stoping			140000	182000
	Total	**18704**	**70056**	**272160**	**309176**

V) Mine Ventilation:

Northern and Middle bands:

Sufficient air shall be introduced from the ramps into the stope access drives (developed in waste on the main and sub-levels) to dilute the diesel exhaust of vehicles operating from that level. Typically each level (mining from the North and Middle Bands) shall have four ends. Each end shall be supplied with 25 m3/s this shall be sufficient to operate one LHD and one truck. Multiple stopes mining in a drive shall be ventilated in series from the access drive. At each stope entrance (cross cut) a 760 mm duct equipped with a 45 kW fan shall force approximately 14 m^3/s into the stope. From the stope air shall return to the access drive where it shall re-mix with the full quantity (25 m^3/s) introduced from the ramp. Although only 12 m^3/s is required for diesel dilution (of the largest vehicle, the LHD, operating in the stope) the additional air (2 m^3/s) shall cater for the fraction of the 'reused/mixed' air from preceding (series ventilated) stopes. The main and sub level access drives (in waste) shall hole into a series of return raises (10 m2 RBHs or equivalent drop raises) established at the extremities of the drives (typically two raises shall be required at each 'extremity'; these shall be located at the foot of return air raises). In order to prevent air short-circuiting between levels through the slot raises (pre-developed between stopes) it is proposed that the quantity of air introduced into a drive (25 m^3/s) shall be regulated by low pressure fans (from each main and sub level drive) discharging into the return raises (at the extremity of each drive). The proposed ventilation system for the North & Middle Band sections is given below in Fig. 4.29.

The planned air quantity to care of the ventilation of the North and Middle Bands is given in Fig. 4.30.

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South Band:

The amount of air supplied to the South Band far exceeds the minimum required for diesel dilution; however the amount of air is required to provide flexibility for machines to operate in the number of ends indicated by the production schedule. Air shall enter the South Band from the surface decline and through the 2.6 m diameter down cast RBH after which it shall be distributed to the working sub-levels through the ramp system. Some air shall enter the South Band (via regulators) from the hoisting shaft connections. At this stage the sub- level return raises (approximately 10 m^2) located on the east and west extremities shall be in place and air from mining activity shall return via the raises through the return airway to the 3.8 m diameter return raise on Ramp Level. Air from any concurrent ramp development shall continue to return via the adjacent ramp raise to Ramp Level and via a connection to the main return airway.

The planned air quantity to care of the ventilation of the South Band is given in Fig. 4.31.

The drawings showing the ventilation layout of the mine is enclosed as Drawing no.'s SCM/ MP/19/12 & SCM/MP/20/12.

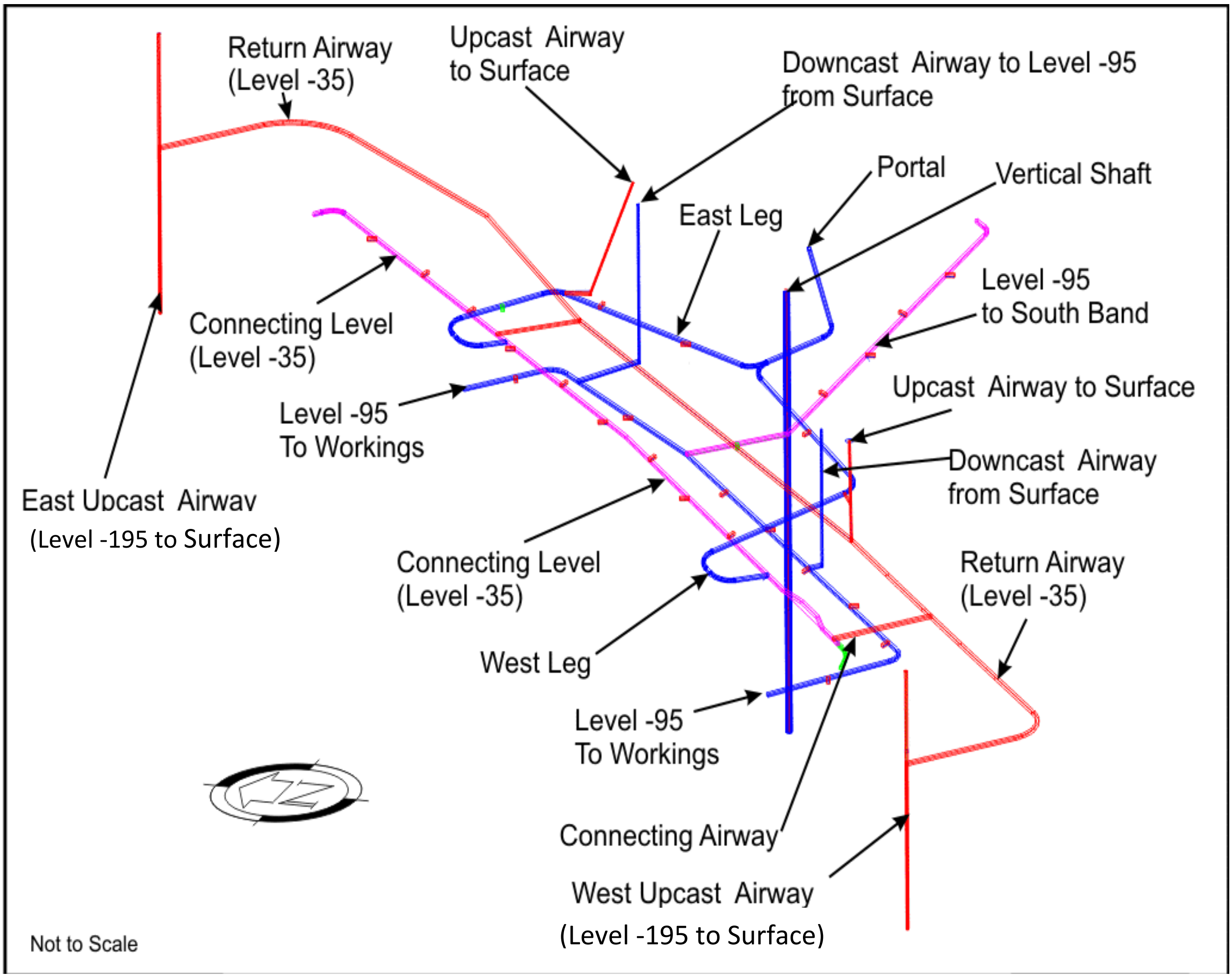

Fig. 4.29: Ventilation System for North & Middle Band sections

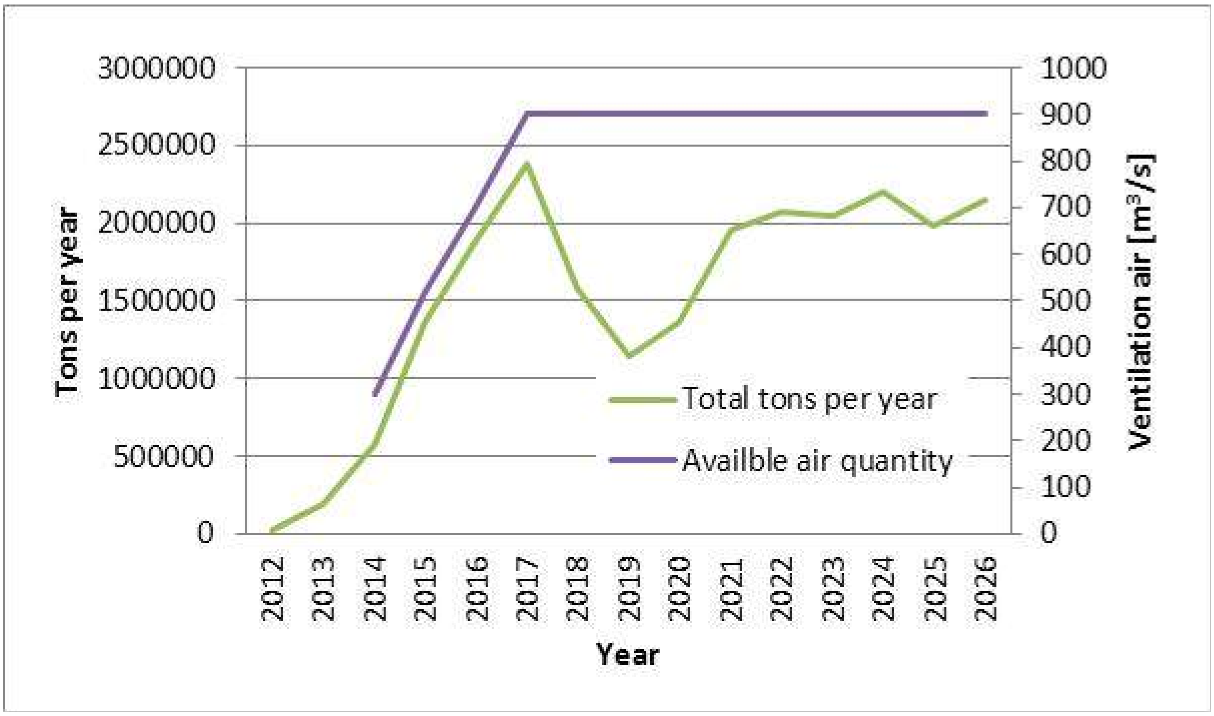

Fig. 4.30: Primary air quantities for Northern and Middle band

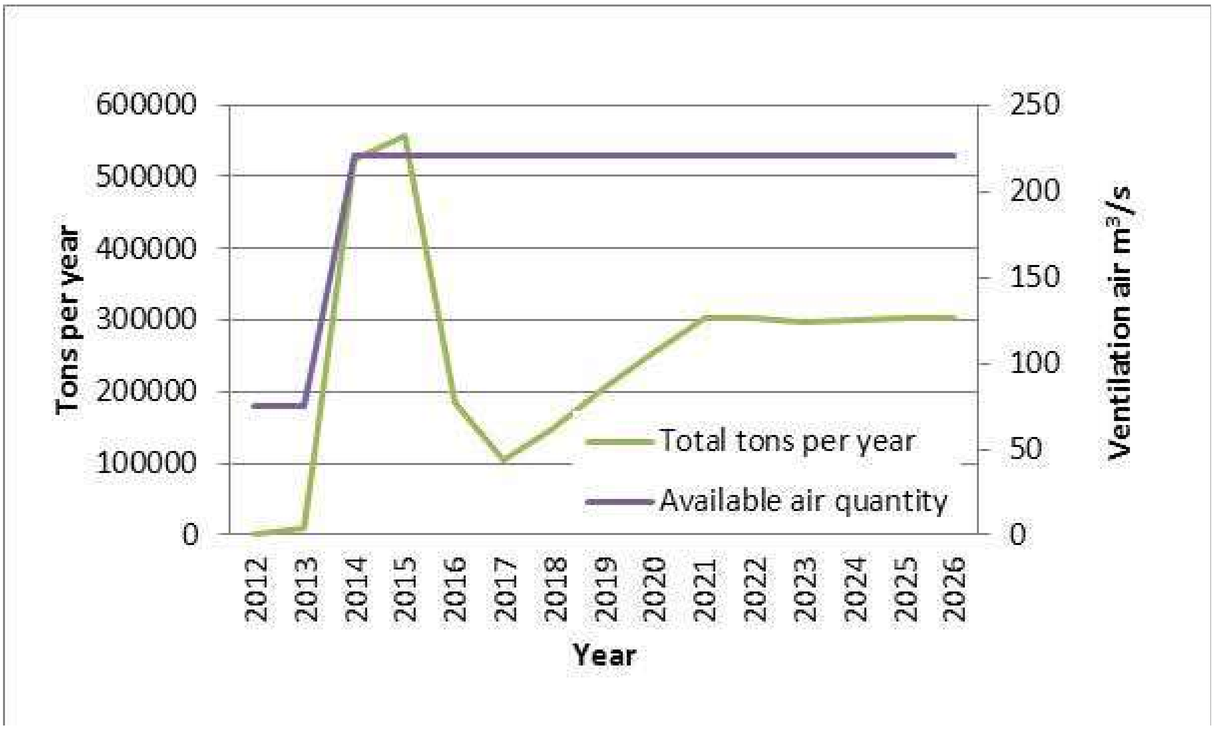

Fig. 4.31: Primary air quantity for South Band

Up cast fan stations

Preliminary ventilation estimates indicate that (mechanised) Sub-Level Open Stoping shall require a total primary air quantity of about 900 m^3/s for the North and Middle Band. An air quantity of around 230 m^3/s shall be required for the South Band. In total there shall be five main fan stations as follows:

- Four fan stations shall be required for the North and Middle Bands. Two of these shall be located east and west of the portal and shall serve return raises A & B; the remaining two shall be located at the East and west extremities and serve return raises C and D. Two fans (bifurcated) shall be installed at each station.
- One identical fan station shall be required for the South Band. The Ventilation shaft shall be of 3.8 m diameter.

Description of the Fan Stations

Each fan station shall comprise two axial fans with associated steel bend, silencer(s) and drift arrangements. To allow for varying operating points (over the life of mine) the fans shall be fitted with hubs to accept additional blades and for varying blade angles and shall operate at efficiencies of about 80%.

The design specification is as follows:

- Flow rate per fan – 115 m^3/s
- Static pressure differential – 2.5 kPa
- Inlet density to fan – 0.98 kg/m^3
- Speed 2 pole – 1500 rpm
- Number of fans – 2 off
- Nominal flow rate for fan station – 230 m^3/s

Main fan station shall include shaft-top bend, 2-way drift assembly, two axial fan-motor sets, inlet and outlet silencers, MCC with switchgear, PLC and instruments.

VI) Extent of mechanization:

The entire mining operations are planned to be executed by trackless diesel equipments. Equipments like electric drills which are to be operated mostly in static position shall have electric mode of operations. Following trackless equipments would be deployed in the mine to carry out mine development and stoping operations. The list of equipments proposed to be deployed in different ore bands are as mentioned in Table 4.8.20 below. The type of equipments remaining same, the make and capacity may change depending upon the availability of the same during actual implementation phase of the project.

Table 4.8.20
List of equipments

Equipment	North	Middle	South	Total
Twin Boom Drill Rig	4	3	2	9
Single Boom Drill Rig	4	3	2	9
Rock Bolter	4	3	2	9
Stope LHD ($5m^3$)	4	3	2	9
Development LHD ($5m^3$)	4	3	2	9
Primary Dump Truck (30t)	6	4	2	12
Longhole Drill Rig	4	3	2	9
Shotcreter	3	2	2	7
ANFO Loaders	4	3	2	9
Service Vehicles	6	4	3	13
Raise Borer	1	1		2
Total	**44**	**32**	**21**	**97**

Adequacy of equipment fleet:

Selection of the mining equipments have been done based on the specifications received from the manufacturers and may undergo change during the actual purchase which shall be in phases as mentioned in the table below. The number of equipments has been estimated based on their performances in the operating mines as given in Table 4.8.21 below. It is seen that every mechanized underground mine and deposit as a whole is unique and follow many unique operating practices which result in different cycle times for these machines.

Table 4.8.21
Adequacy of equipments

Equipment	Total numbers	Ideal equipment shifts/day	Availability in shifts/day (85%)	Average output/ shift/ equipment	Total Output/ day	Correlated Production Capacity /development achieved /day	Target /day	Purchase schedule (Yr-1 to Yr-5)					Remarks
Twin boom drill jumbo	9	27	23	175m	4025m	47m	46m	2	3	3	1		Deployment in development faces
Single boom drill rig	9	27	23	125m	2875m	2400 t	2000t		1	2	3	3	In stopes for production drilling. Actual number of cut & fill stopes would be decided after the level development, additional capacities kept as the numbers may vary.
Rock bolter/Cable bolter	9	27	23	35	805 no.s	161m	46m	2	2	2	2	1	Requirement shall vary as per the number of operating C&F stopes. Numbers are likely to be readjusted after re-assessment of the number of SLOS and C&F stopes during detail design.
Development LHD	9	27	23	150 t	3450 t	58m	46m	2	3	4			Efficiency may vary with face distances.
Stope LHD	9	27	23	200 t	4600 t	4600 t	4500 t			2	3	4	Remote control LHD’s to be deployed in stopes.
Primary dump truck	12	36	31	270 t	8370 t	8370 t	8050 t	2	3	4	2	1	To be deployed in conjunction with development and stope LHDs
Long hole drill rig	9	27	23	50m	1150m	4600 t	4500 t		1	2	3	3	Only be used in Sub level open stopes in ring drilling.
Shotcreter/ Spray mec	7	21	18	50 sq m	900 sq m	900 sq m	680 sq m	2		2	2	1	Only to be used in case of poor rock developments and stope walls. Capacity assumed with 3.5 cum Miller.
ANFO loader	9	27	23	40 holes	920 holes	920 holes	684 holes	2	1	2	2	2	1000kg capacity. To be procured in phases with increase in number of Sub- level stopes.
Service vehicles	13	39	33	Separate service vehicles for Northern, middle and southern band shall be provided.				2	3	4	3	1	Includes service trucks, scissor lifts, passenger carriers and Lub trucks.
Raise borer	2	6	5	4m	20m	20m	20m	1	1				These have limited applications for the ventilation and ore pass raises only.

Equipment details:

General specifications of the equipments proposed are as given below. The specifications suggested are as per the model ranges available. The equipments are proposed to be purchased in phases as the mine developments starts. The specific model to be purchased in subsequent phases would be selected based on performance, safety and operational environment point of view.

1. Twin boom electro-hydraulic drill jumbo – It is a double boom drill jumbo having diesel tramming and electric drilling facility. It shall be used for mainly solid face drilling like Decline, level and sublevel drives & crosscuts etc.

    a. Model /Make – Sandvik/ Atlas Copco – twin boom
    b. Hole dia. – 43-64mm
    c. Reamer dia. – 76-127mm
    d. Standard voltage – 380-690 V, AC
    e. Power: Diesel engine – 110- 145 kW and Electric – 135 -175 kW

2. Load haul dumps or LHDs – It is a wheel mounted loading equipment with a front end bucket and can carry load over short distances. These loading and hauling equipments shall be used from the beginning of the decline to the stope development and stoping as well. During development of the decline, drifts , cross cuts and drives the LHDs shall have a operator in the cabin where as during stope extraction the LHDs shall the remote controlled one, because they shall be working under the undressed stope roof. Loading bays shall be excavated in each cross cut junction to facilitate the ore loading from stopes.

    a. Model/ Make – Sandvik/ Atlas Copco
    b. Bucket capacity – 4-7 cum
    c. Power – 220-285 kW

3. Dump trucks- Articulated dump trucks shall be used for long distance transportation of the ore/waste from the stope and development faces. The LHDs shall load the materials in the trucks which shall have an access to the nearest loading point from where it shall carry the same to the nearest dumping space ore pass or waste yard.

    a. Model/ Make - Sandvik EJC/ Atlas Copco
    b. Dump box capacity – 15 cum - 20 cum
    c. Power output – 350-410 kW

4. Rock bolters – Considering the higher rate of development of the development faces the immediate face supporting has been mechanized. Rock bolting machines shall be used which shall undertake the entire operation from drilling to grouting of the hole with cement capsules.

    a. Model/make – Sandvik/Atlas Copco
    b. Hole dia. – 33-45mm
    c. Bolt length – 5’-10’
    d. Power – Diesel engine – 60-85 kW/Electric motor – 50-80kW

5. Long hole drill rigs – Long hole rigs shall be used during stoping for drilling in sublevel stopes. The holes shall be drilled from the bottom sub level or main level drive upwards in fan cut pattern, so as to cover the entire ore mass in a 10m x 10m x 20 m block.

    a. Model/ make- Sandvik/Atlas Copco
    b. Hole dia. – 85-140mm (3.5”-5.5”)
    c. Hole depth – 30 -60m
    d. Power : Diesel engine – 85-120 kW, Electric – 65-85 kW

6. Cable bolters – Cable bolting would be required to be done in cut & fill stopes where thin ore slices are to be taken by drilling and blasting. This method is proposed in the mixed type of ore where the stope walls and rock is softer than the area where sub level stopes are planned. The cable bolts would be drilled and fixed from the first level as a proactive support measure. Horizontal slices would be taken from the bottom and exposed cables after each slice would be trimmed for next production round as shown in Fig. 29.

    a. Model/Make- Sandvik/Atlas Copco
    b. Hole dia. – 48 to 60mm
    c. Length of hole – 25m
    d. Power: Diesel engine – 110 kW, Electric – 75kW

7. Raise borers – The main purpose of the raise borers shall be to facilitate the ventilation connections in line with the development schedule. There are about 1900 m of raise boring for the ventilation connections and 2600m for the ore pass system. The raise diameter shall vary from 1.6m to 4.1 m.

    a. Model/make- Sandvik/ Atlas Copco
    b. Dia. – 1.6 m-4.1 m

Service equipments:

Apart from the above equipments required for the development and production, the service equipments as listed below would also be deployed.

1. Service truck – Service trucks shall be used mainly for shifting of materials inside and in-between the levels and sublevels. These are flat bedded low profile trucks exclusively made for underground operations

    a. Model /make – Normet/EJC
    b. Power – 60-85 kW

2. Scissor’s lift – These are similar to service trucks having hydraulic lifting arrangements for height applications in stopes and high places like loading bays etc.

    c. Model/make- Normet/EJC
    d. Power – 60-85 kW

3.Shot creter – It is a concrete mixing drum and sprayer with nozzle mounted on a service carrier capacity vehicle. The mixing drum can be loaded from the batching plant in surface and carried to underground workings where the exposed rock surface is to be shotcreted.

4.Charmec – It is also a service carrier capacity vehicle mounted with two tanks to load the ANFO or Slurry explosives to be loaded in the underground stope holes for solid or mass blasting.

5.Lube truck/Crane truck – They have similar engine capacity as the service truck and having facility to load the fuel and lubricants in the Lube truck and a (2 t- Cap.) Crane for underground use.

Haulage within the mining lease hold:

The underground mine trucks shall be utilized for transport inside the underground mine in declines, drives and ramps. A site dump yard for ore and waste shall be designated near both the decline portals within 100-200m from the decline portals where the loaded underground trucks would dump the waste and incidental ores. The ore and waste shall subsequently be transferred by using surface rear dumpers and wheeled pay loaders to waste dump and processing plants respectively by 4.6 cum pay loaders and 35 t surface dumpers.

Development & Production schedule:

The following parameters are taken in the schedule.

- Decline and Ramps are at 8°;
- Development rates;
    1. Decline and Ramps 60 m/month;
    2. Flat Drive 60 m/month;
    3. Cross Cut 40 m/month;
- Stope development 6 days;
- Ring drilling 50 m/shift
- Band size mined 2 000 m³ (10 m x 10 m x 20 m);
- Loading 12 days (200 t/shift);
- Install Bulkhead 7 days;
- Filling 3 days (100 m³/hr);
- Curing 30 days;

The production target for all the three bands has been considered as given below in Table: 4.8.22.

Table: 4.8.22
Production Target (MT)

Ore body	Monthly	Annual
Northern band	91667	1100000
Middle band	83333	1000000
Southern band	25000	300 000
Total	200000	2400000

The total time to mine a 10 m x 10 m x 20 m block, load and fill is approximately 62 days. In the schedule the extraction of a single stope lift, 110 m x 20 m high, is simplified by dividing this panel into its individual 10 m x 10 m blocks to obtain the number of blocks in the panel and then the length of time to mine and fill the blocks is calculated. The average production rate over that period is determined by dividing the total tonnage mined in the panel divided by the total time to mine and fill the panel. The period to mine and fill includes the curing time. The production target for the underground option is 2.4 million tons per annum with the tonnage divided between the three ore bodies as shown in Table 4.8.23. It has been indicated that steady state production should be reached by 2022 to 2023.

Table-4.8.23
Open cast and Underground ROM Production
(All Figures in MT)

Year	Chrome Ore ROM Production from Opencast Mining			Chrome Ore ROM Production from Underground Mining			Total Chrome Ore ROM Production (Opencast+ Underground)			Pyroxenite Ore ROM
	OB-X	OB-II	Total	OB-X	OB-II	Total	OB-X	OB-II	Total	
2011-12	1276863	110352	1387215	0	0	0	1276863	110352	1387215	50000
2012-13	1405443	108751	1514194	0	0	0	1405443	108751	1514194	0
2013-14	1636618	120059	1756677	0	0	0	1636618	120059	1756677	0
2014-15	1743152	129788	1872940	0	18704	18704	1743152	148492	1891644	0
2015-16	1740492	0	1740492	35531	70056	105587	1776023	70056	1846079	0
2016-17	1818882	0	1818882	166488	272106	438594	1985370	272106	2257476	250000
2017-18	1560537	0	1560537	535200	309176	844376	2095737	309176	2404913	500000
2018-19	251800	0	251800	1850000	299388	2149388	2101800	299388	2401188	500000
2019-20	201800	0	201800	1900000	299388	2199388	2101800	299388	2401188	500000
2020-21	141800	0	141800	1960000	299388	2259388	2101800	299388	2401188	500000
2021-22	0	0	0	2101800	299388	2401188	2101800	299388	2401188	500000
2022-33	0	0	0	2101800	299388	2401188	2101800	299388	2401188	500000

IV) Method and sequence of stoping:

The selection of the mining method was based on the recommendations of the geotechnical study in 2009 (SRK Report 218663/8) and a further study in 2010 (Sukinda Underground Feasibility Study: Numerical Modeling and Analysis, August 2010) which undertook the numerical modeling and analysis of the ore bodies. The recommendations of the 2009 study were:

- The friable chrome should not be mined from underground;
- The mixed chrome should be mined using cut and fill methods, and;
- The lumpy chrome should be mined by open stoping methods.

The objectives of the numerical modeling and analysis were to:

- Determine the dimensions of crown pillars;
- Assess caving methods for mining in friable chrome;
- Assess the performance of cut and fill stopes and support requirements;
- Assess the open stoping mining sequence and dimensions of rib and sill pillars;
- Evaluate the performance of open stopes at different depths, using the material properties for each ore body, and;
- Assess the influence of different in-situ stress measurements.

In addition to this, the backfill requirements need to be evaluated for the purpose of determining the backfill plant and operating costs.

The 2D non-linear modeling showed that the crown pillar sizes depend strongly on the assumed stress regime. Assuming the results of the Mesy stress measurements to be correct, the ratio of the required crown pillar size to ore body width is approximately 1:1. If the stress regime postulated by SRK's desktop study is used in the modeling, the ratio of crown pillar height to ore body width becomes approximately 1:2.5. A minimum 60 m crown pillar is required in terms of the regulations. In the North and Middle bands the remainder of the crown pillar shall be made up of friable chrome. In the South band it shall be necessary to leave additional lumpy chrome unmined, in order to meet this requirement.

Support system:

The design criteria adopted in the sub level open stopes is to provide self-supporting stopes with limited localized support, as required.

The stability of the waste walls along strike impose limits to the stope dimensions in MB and OBX (mixed ore). In Northern band, with a chrome band width of more than 10 m, it would not be possible to open up the full width; there the stope width would be restricted to 10m as specified.

For the sub level stoping operations, a bottom up sequence is to be adopted with Primary and Secondary stopes, with backfill for a higher extraction percentage. Sill/crown pillar of 20m width and Rib pillar of 10m width have to be left in the process for overall stability of the blocks. However upon filling of the stope blocks in the same sequence and the stability increases significantly and the rib and sill pillars would be extracted in a second phase of extraction.
Apart from the natural supports the external supports to be provided for support of immediate roof and sides are as below. The supports can be classified in following two categories.

5. BLASTING :

5.1 Broad Blasting Parameters:

5.1.1 Opencast Mining:

The requirement of drilling and blasting at Sukinda Chromite Mine is mostly confined to lumpy ore 9OB-II) where the overburden consists of hard quartzite and also for ore raising in both the quarries. In friable zone (OB-X) the drilling requirement is 20% of total overburden, which is generally soft and amenable to direct excavation by machines.

The ore benches , wherever required are drilled by 100mm diameter blast holes and spacing of 3.0m by 4.0m to 3.5m by 4.5m respectively as per the hardness and strata conditions.

The overburden benches are drilled by 150mm diameter blast holes with a burden and spacing of 3.5m by 4.0m to 5.0m by 5.0m respectively as per hardness and strata conditions. Blasting is done in single or multiple rows. A maximum charge of 264kg/ delay in a round is used for blasting. On an average stemming varies from 2-2.5m for 6-8m holes and 2.5-3.0m for 8m-11m holes. The drilling patterns followed at the mine are given below:

	Burden (m)	Spacing (m)	Depth (m)
OB II overburden	3.5 - 4.5	4.0 - 5.0	9.0 -11.0
OB X overburden	4.0 - 5.0	4.5 - 5.0	8.0 -9.0
Ore body	3.0 - 3.5	4.0 -4.5	5.0 -6.0

In hard formation, sub grade drilling up to 0.1 to 0.2 times the burden is provided to avoid toe formation. On an average 25-30 holes are blasted in a row using conventional explosives. The secondary blasting is confined only to lumpy ore boulders. Presently blasting is carried out every day of the week to meet the proposed excavation targets. On an average the powder factor comes to 7-8 tonnes/ kg of explosives. Now Site Mixed Emulsion Explosive is being used.

Pre-split Blasting:

The main purpose of this controlled blasting technique by pre-splitting is to reduce back break and minimized damage to the pit walls from the production blast, thereby improving high wall stability leading to better safety. Pre-split blasting is one of the controlled blasting techniques for which is successfully practiced in Sukinda Chromite Mine for better slope stability.

In this technique a row of closely spaced parallel holes are drilled along the excavation perimeter or final excavation line and are fired simultaneously prior to the primary blast using decoupled charges or low strength and energy explosives. The production drill hole of single row/multiple row of dimension (burden 3 m, spacing 3.5 m) is marked in square pattern. Depth of the drill holes as per bench height +10% sub grade drilling and dia. of drill holes is 150mm. Pre-split drill hole position are marked along the final excavation line. The pre-split line has a burden of 3 m from the last row of production line and the spacing between the holes is 1.9 m. Priming cartridge of dia. 150 mm and 6.25 kg is tied with detonating fuse to avoid toe formation. Column charge of cartridges 50 mm dia and 1 kg each is tied in 1.5 m intervals with detonating fuse in the pre-split holes.

All the activities above are conducted on the surface. The whole cartridges with detonating fuse are then inserted into the hole with a rope tied with it. Care is taken so that all the cartridges are in the straight line with detonating fuse by tightening the rope on surface.

The stemming length is 2 m from collar of the hole. Gasbag of 150 dia. is used for air decking. Production holes are charged and all the holes are stemmed properly. All the pre-split holes and production holes are connected in rows. The trunk line to the Pre-split drill holes is connected with front row production blast trunk line by 500 ms delay detonator on one side and the production rows by 25 ms delay detonator.

5.1.2 Underground Mining:

Two predominant methods of stoping are to be followed as mentioned above. They are Sub level Stoping and Cut & fill mining. The development pattern and headings shall be same for both types of stoping methods. The development faces like decline, waste drive, ore and sub level drives shall be drilled as per the drilling pattern as shown in the drawing. The holes shall be 45mm dia. which shall be charged with 40mm power gel cartridges. Each round shall be drilled for 3.6m length. The holes shall be charged up to 66% of its length and rest would be stemmed with clay plugs. The charge factor for the solid faces is proposed to be around 0.7 to 1.0kg /tonne of broken rock depending upon the rock type. Burn Cut drilling pattern with a central hole of 200mm as shown below in Fig. 5.1 shall be adopted for blasting in the development headings.

Drilling pattern in Sub level stope blocks (10m x 10m x 20m) shall be of ring pattern as shown below in the Fig. 5.2. In thicker portion of the ore bands more parallel blocks shall be extracted by drilling ad blasting as per Fig.5.3 given below. The ring drilling hole diameter has been planned to be of 58mm and would be charged in ANFO/ SMS with 50mm primer cartridges and boosters at an interval of 2.5m.

The drilling option for these holes may be either an upper fan ring or a lower fan ring. Both the patterns have its advantages and disadvantages. The most suitable method would be selected based on the trial stoping and blasting outcomes.

In the Cut & fill stopes the 1st stope drive shall be taken as a solid face in burn cut drilling and blasting as in case of normal level developments. The ore drive shall be filled up to 3m. Roof of the drive shall be stripped further by horizontal slicing so as to heighten the entire drive to 5 m and the same cycle repeats. Here the holes shall be charged with ANFO.

A full bottom bench drilling pattern and blasting is proposed to be adopted with three boom hydraulic shaft drill rigs. A 4.2 m round would be taken in each blast. The explosive to be used here is 50mm cartridges of Powergel which have excellent water resistant properties. The shaft drilling pattern is as shown in Fig. 5.4.

The further details of the drilling and blasting pattern, no. of holes/ round, charge per delay and charge density etc. for the development headings and shaft is given in Table 5.1.1 below.

Table 5.1.1
Blasting and face details

Area/ face, Size	Blasting pattern	No. of holes/ face/ ring	Max. charge /delay/ hole in kg	Max. no. of holes/ round	Charge density (Kg/m)	Type of Explosives	Powder Factor (Kg/Cum)
Decline, 5m x 5m	Burn cut	44*	4.4	37	1.2-1.8	Powergel-40mm	0.6-0.8
Drive/drift, 5m x 5m	Burn cut	44*	4.4	37	1.2-1.8	Powergel-40mm	0.6-0.8
Drive, 4m x 4m	Burn cut	33*	4.4	31	1.2-1.8	Powergel-40mm	0.6-0.8
Vertical shaft, Dia. 6.6m	Wedge cut	81*	5.5	68	1.9-2.4	Powergel-50mm	0.7-1.0
Sub level stope, 10m x 10m x 20m	Ring pattern	19	68	76	3.8-4.4	ANFO	0.3-0.5
Drift & fill stope 5mX4m	Horizontal slicing	18	3.2	18	0.8-1.2	ANFO	0.3-0.6

*Excludes reamer hole

Secondary blasting:

If the boulder size in the stopes exceeds a size of 2’ x 2’ size these are to be blasted in the stopes or the ore pass grizzlies. Maximum attempts would be made to sort the oversized boulders during the LHD mucking process in the stopes, which would be blasted there itself. Oversize pieces if any that get carried away to the grizzly shall not pass through it and therefore have to be blasted there.

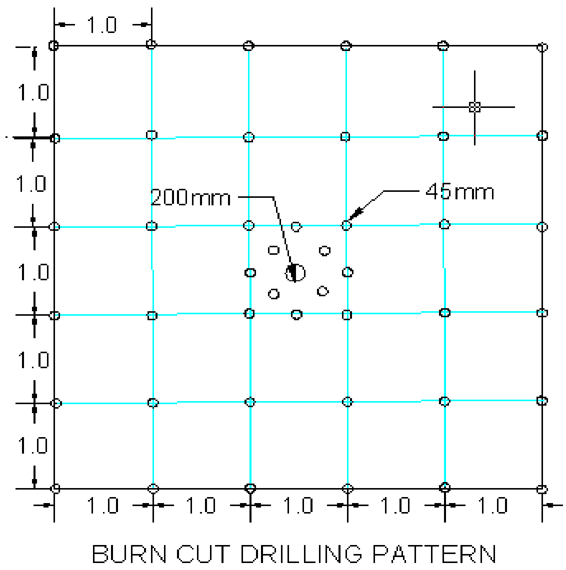

Fig. 5.1: Burn Cut Drilling Pattern

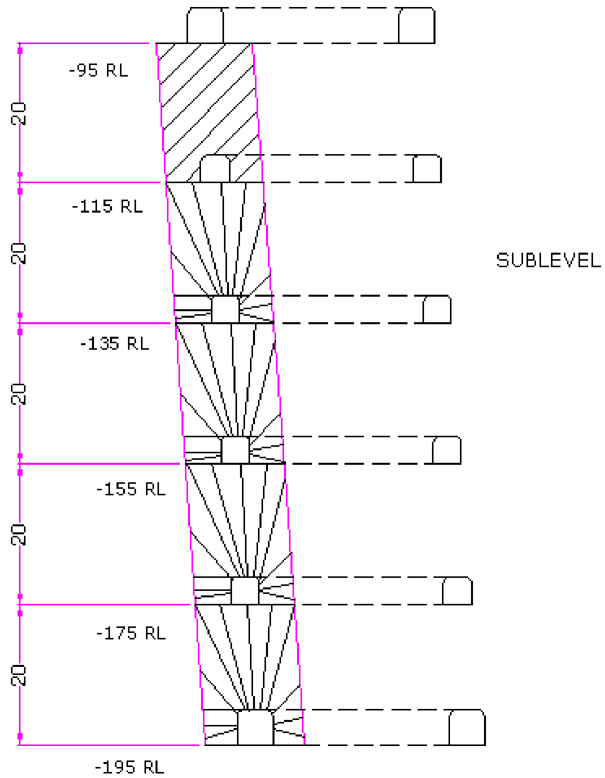

Fig. 5.2: Ring cut drilling pattern for Sub level stopes

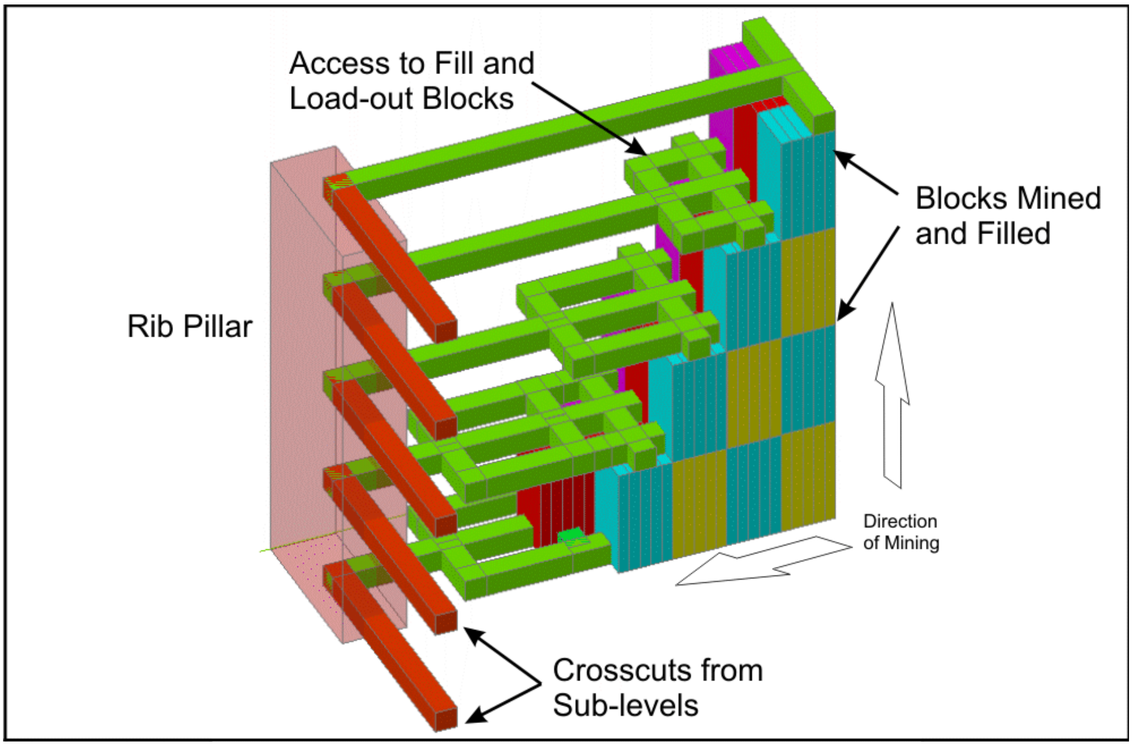

Fig. 5.3: Sub level blasting and extraction sequence in wider part of ore bands

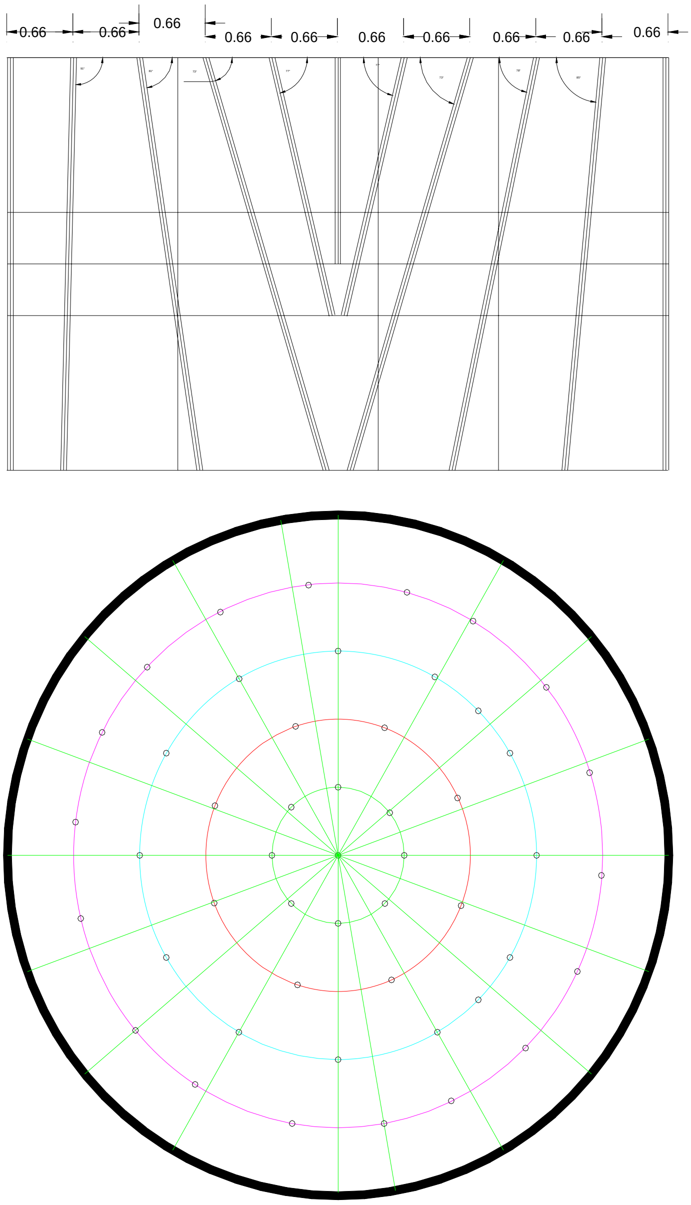

Fig. 5.4: Shaft Drilling Pattern

5.2 Type of Explosives used:

5.2.1 Opencast Working:

The base charge is mainly of Indo Prime, OCG, PG1, PG2 with Primex and the column charge consists of Indogel, GN1, and ANFO etc. Recently the use of SME (Site Mixed Emulsion) has started for better safety, lesser ground vibration and better fragmentation.

5.2.2 Underground Working:

The mine development activities include solid blasting of the waste and ore faces using Power gel(40mm) which is a high strength, robust detonator sensitive emulsion explosive and non electric shock tube detonators. Higher dia (50mm) Power gel will also be used in the shaft sinking along with non electric shock tube detonators. However during the mass blasting operations in the underground stopes ANFO will be extensively used with Powergel primer cartridges.

5.3 Storage of Explosives:

An explosives magazine exists for storage of explosives having license no. E/ HQ/OR/ 22/ 178 (E23229). For taking shelter during blasting, total 6 nos. of portable type blaster's shelters are provided. For transporting explosives from magazine to site 2nos of explosive vans of approved type have been provided (OR04-5840&OAU- 4078) having license no. E/EC/OR/22/57(E19284) and E/EC/OR/25/50/ (E19210) respectively.

The mine has a magazine with a storage capacity of 20 tonnes explosive and 44000 no.s detonators as per License no. E/HQ/OR/22/178(E23229). The details of explosive quantities permitted vide this license is as per the table below.

Table 5.3.1
Explosive Magazine capacity

Explosive type	Quantity/no.
Class-2	19 tonnes
Class-3-II	1 tonne
Detonators	44000 no.s

6. MINE DRAINAGE :

Opencast:

It has been observed from the nearby wells and water bodies that the water table varies between 4m to 9m from ground level on the valley side between monsoon and dry seasons respectively.
To continue the mine working below water table by using heavy mobile equipments, quarry dewatering continues throughout the year. With the mine working going deeper the water inrush due to seepage is likely to go up. The average discharge of water per day for the period 2009-10 from various working quarries is given below.

Quarry Name	Mine Discharge (KL/ Day)
OB-X	13618
OB-II	5820
Total	19438

The water from the quarry benches is channelized through bench drain to the sump from where it is pumped to the reservoir for initial settling of suspended solids through pontoon pumps. Then the overflow water goes to the main ETP as inlet for treatment. Garland drain is provided at the top of the quarry to channelize the surface runoff to the ETP located at the lease boundary end for treatment before discharge out of leasehold. The mine water is reused in COB Plant, dust suppression and service facilities such as workshop; civil construction etc. the excess water after above use is discharged outside the leasehold area after treatment. The mine dewatering network is shown in the drawing no. SCM/MP/09/12.

The company had engaged M/s NEERI (National Environmental Engineering and Research Institute, Nagpur) for undertaking a comprehensive study to provide the necessary technical-knowhow and design of a suitable plant for treatment of quarry discharge water. The company was first in the valley to treat the mine discharge water for removal of hexavalent chromium. It has already installed one ETP having roughing filters to arrest suspended solids from mine discharge.

Underground:

The average regional ground water table in the area varies from 4m to 9m below the surface. The present open pit has already crossed the water table much earlier. The 1st underground level, i.e. the ventilation drift level shall be at -35 RL which is about 150m below the ground water table. The levels and sub levels have been divided in to blocks based on the position of ramps. Each block shall have a sump to which all the water from ore drives and waste drives shall flow down. The levels and sublevel portion in each block would be developed at mild gradient (1 in 100) from the ramp so as to allow the water to flow in to the sump near by the spiral. The schematic line diagram showing sump in each level is given below in Fig.6.1.

The pumping arrangement in each of the declines and pump specifications are given in Table 6.1.1 & 6.1.2 respectively.

Table 6.1.1
Pumping arrangement in each decline

Decline	Ramp Serviced	Upper pump station elevation	No. of pumps to be installed
North	West ramp	-95 level	3
North	Central ramp	-95 level	3
North	East ramp	-115 level	3
South	South ramp	-95 level	3

Table 6.1.2
Pump specifications

Pump Station	No. of pumps in series	Quantity pumped (l/s)	Total Operating Head, m	Pump Type	Motor Size (kW)	Approx.' speed (rpm)	Daily running hours
West upper	3	130	227	C5	185	1350	8
West lower	2	130	200	C5	185	1525	8
East upper	3	130	247	C5	185	1350	8
East lower	2	130	200	C5	185	1525	8
Central upper	3	130	227	C5	185	1350	8
Central lower	2	130	200	C5	185	1525	8
South upper	3	75	220	C5	185	1325	6-8
South lower	2	75	200	C5	185	1525	6-8

The pumping system in the open pits OBX and OBII shall eventually be used to supply the new underground mining operations in the Northern, Middle and Southern Bands. It shall still supply the Colony gardens, the pyroxenite operations and the effluent treatment plant, as it does now. Each of the underground access Decline Ramps (North and South) shall have its own dedicated surface settler.

The surface settler at the North Decline Portal shall service the West, Central and East ramps. The South Decline Ramp settler shall service the South ramp. The settlers shall handle the dirty return water from the underground operations and make up water from the open cast quarry pump house, as required. The settlers shall be positioned close to the portal entrance and they shall be engineered to enable easy cleaning out of the settler sections by LHD mobile machinery. The settler shall have a running and standby section, to allow one section to be cleaned out, whilst the other section is operating. Water from the level development ends and production areas shall be pumped back to the decline and dropped via inter-level drain holes to the main pump stations situated at the lowest levels mined in the areas serviced by each decline. The dam design shall be of sufficient capacity to store 16 hours of mine drainage water; approximate size of the dams shall be 5m x 5m x 30m length, the dam shall have two legs, one operational leg and one standby leg. This is to allow one dam section to be cleaned out whilst the other leg is operational. The dam cleaning shall be done by LHD and dump truck, with the slurry being classified as ore and transported to a drying area at the plant ore stockpile on surface. When an operating dam leg is taken off line, the fines are allowed to settle for a period of around 1 week, after which time a small submersible pump shall be used to decant the clear water and pump this into the operational dam leg. Once the water has been pumped off, the slurry can then be loaded out and transported to the surface beneficiation plant stockpile area.

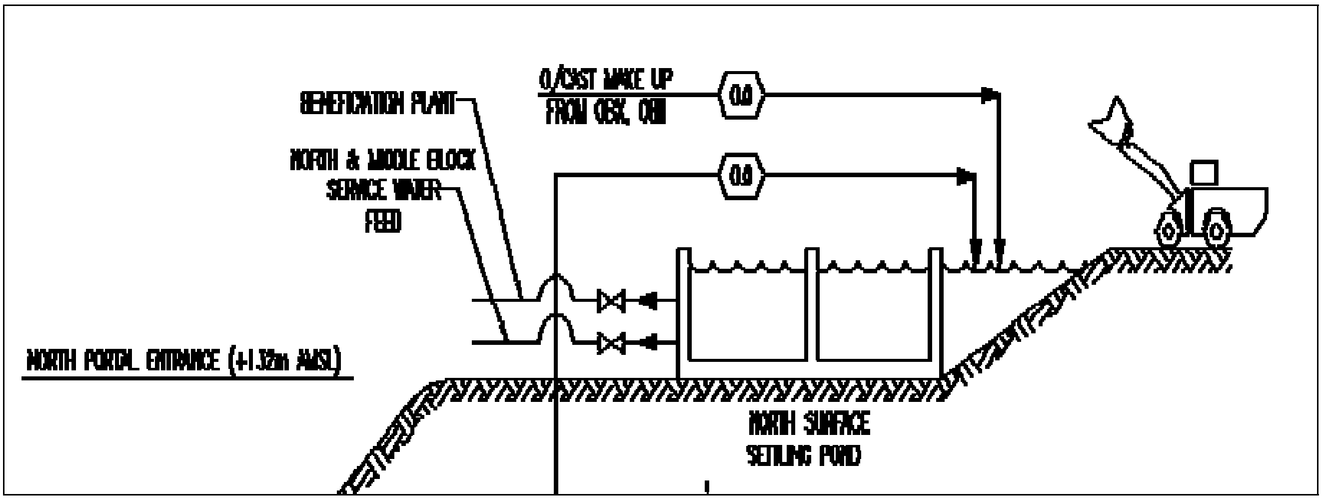

Fig. 6.1: Layout of portal settling pond

7. STACKING OF MINERAL REJECTS AND DISPOSAL OF WASTE :-

a) Nature Of Waste & Mineral Rejects:

At Sukinda Chromite Mine, the term “Waste” includes the overburden generated viz. nickeliferous limonite, serpentinite and quartzite.

Considering the recent research work being done at RRL, Bhubaneswar on the nickeliferous limonite there is a possibility of the same to be a potential source for nickel in the future. Similarly, certain parties have indicated their desire to utilize the hard serpentinite as a flux. Thus, these materials may also now be treated as “associated industrial mineral”. Efforts shall be made to stack such material separately, and develop markets for the same. The approximate grade/ quality of various associated mineral are given below in Table no. 7.1.1.

Table. 7.1.1

Associated Mineral/ OB	Quality Range (%)						
	MgO	SiO_2	Cr_2O_3	CaO	Al_2O_3	Fe	Ni
Serpentinite	24.0-40.0	28.0-45.0	2.0-8.0	0.05-1.5	1.0-5.0	5.0-12.0	
Quartzite	0.0-1.5	89.0-95.5	0.3-3.0	0.12-0.4	1.0-3.0	2.0-2.5	
Ni- Limonite	0.5-1.6	10-32	4-10	0-0.2	2.5-7	25-44	0.39- 0.50

Considering, the high cost involved in the downstream agglomeration process, the metallurgical plants in our country are no longer interested in the chrome ore fines (-10 mm). Such chrome ore fines generated after processing (dressing, sorting, crushing and screening) are therefore to be treated as “mineral rejects” and shall be stacked separately and shall be processed in the beneficiation plant to the extent possible.

Similarly, the pyroxenite fines (-10 mm) generated after processing shall be treated as “mineral rejects” and shall be stacked separately. Efforts shall be made to utilize the same to the extent possible.

The yearly generation of overburden / associated mineral from the respective quarries proposed from 12.01.2013 to 2017-18 through both opencast and underground mining operation is given in Table 7.1.2 & Table 7.1.3 below. As shown in the Table 7.1.2 given below, the likely generation of nickeliferous limonite for next five years shall be 313 Lakh Cum with an annual average of 62 Lakh Cum approximately. The yearly generation may vary due to changes in production requirements, which is largely governed by fluctuations in the market demand. However, efforts shall be made to maintain the total quantity for the 5 -year period in order to ensure proper development.

Table 7.1.2

OB-X Quarry

YEAR	OPENCAST			UNDERGROUND		TOTAL PROPOSED OB+ WASTE (CuM)
	TOTAL ROM (MT)	STR. RATIO	PROPOSED OB (CuM)	TOTAL ROM (MT)	PROPOSED WASTE (CuM)	
2012-13*	351361	3.70	1300000	0	0	1300000
2013-14	1636618	3.30	5400000	0	12165	5412165
2014-15	1743152	3.38	5900000	0	48617	5948617
2015-16	1740492	3.73	6500000	35531	136140	6636140
2016-17	1818882	3.57	6500000	166488	193637	6693637
2017-18	1560537	4.17	6500000	535200	267433	6767433
TOTAL	8851042	3.63	32100000	737219	657992	32757992

*OB generation figure for 2012-13 is from 12.01.2013 till 31.03.2013.

Table 7.1.3

OB-II Quarry

YEAR	OPENCAST			UNDERGROUND		TOTAL PROPOSED OB (CuM)
	TOTAL ROM (MT)	STR. RATIO	PROPOSED OB (CuM)		PROPOSED OB (CuM)	
2012-13*	27188	6.90	187500	0	2500	190000
2013-14	120059	7.08	850000	0	12984	862984
2014-15	129788	6.55	850000	18704	44653	894653
2015-16	0		0	70056	91494	91494
2016-17	0		0	272106	95546	95546
2017-18	0		0	309176	103929	103929
TOTAL	277035	6.81	1887500	670042	351106	2238606

*OB generation figure for 2012-13 is from 12.01.2013 till 31.03.2013.

Year wise generation of Associated Minerals / overburden from the different quarries and build up of dumps during both opencast and underground mining are given below in table 7.1.4.

Table 7.1.4

Associated Minerals / Overburden generation (Cum)

Opencast & Underground Mining

Year	Generation of Associated Minerals / OB (Lakh CuM)		Total (Lakh CuM)
	OB X Quarry (Ni- Limonite)	OB II Quarry (Serpentinite & Quartzite)	
2012-13*	13.00	1.90	14.90
2013-14	54.12	8.63	62.75
2014-15	59.49	8.95	68.43
2015-16	66.36	0.91	67.28
2016-17	66.94	0.96	67.89
2017-18	67.67	1.04	68.71
Total	327.58	22.39	349.96

* OB generation figure for 2012-13 is from 12.01.2013 till 31.03.2013.

Similarly the year wise generation of mineral rejects from OB-X & OB-II quarry and generation of tailing after beneficiation is given below in Table 7.1.5

Table 7.1.5
Year wise Generation of Mineral Rejects & Tailings

Year	Generation of Mineral Rejects (MT)			Chrome Tailings (Dry MT)
	OB X Quarry (>10% Cr_2O_3)	OB II Quarry (-10mm fines)	Proxenite Dust (-10mm fines)	
2012-13*	6920	11506	0	117704
2013-14	30274	14349	0	471065
2014-15	32444	5806	0	471065
2015-16	31854	26011	0	427296
2016-17	42322	30518	25000	427296
2017-18	47840	11506	50000	427296
Total	191654	99696	75000	2341722

* Figures for 2012-13 is from 12.01.2013 till 31.03.2013.

Low grade lumpy ore produced during mining shall be stacked over old dumps (1A & 1B) over backfilled area. The low grade ore, mineral rejects and chrome tailing shall be stored at designated place as shown in the surface plan having drawing no. SCM/MP/02/12. Efforts shall be made to consume the OB-X quarry mineral rejects in the COB Plant to the maximum possible extent.

b) Selection Of Site:

The following points are taken into consideration for selecting site for the disposal of waste.

i). Area of disposal should be barren.
ii). Area should be in proximity to the places of work to avoid long hauls/ lead.
iii). The area so selected should not be over land earmarked for township/ other ancillary facilities connected with the mining or proposed to come up in near future.

While selecting the dumping space for waste disposal all the above points had been considered, now, within the reduced leasehold area, no virgin barren area is available for disposal of waste. In conformity with the observation of the Sharma Committee, the Central Govt. vide order no. 5(22)/95-MIV dated 17.08.95 has directed state Govt. to provide additional area for waste dumping outside the leasehold area of erstwhile leasehold area of 1261.476 ha held by the company. In view of the exhausted space within the leasehold area for disposal of waste, Company had applied for a land lease of 289 ha for stacking of the nickeliferrous limonite and overburden / waste disposal. Subsequently, DPF diverted in favour of M/s TISCO Ltd. by MoEF vide letter no. F. No. 8-28/2004-FC, dated 11.05.2005 (Annexure-21). Assumption was drawn that the permission for additional area of 100 ha. will be obtained by 2004-05, however permission for dumping was obtained at a later date and dumping in the new area started in June’05. There was delay in making road to the additional area, tree felling due to ongoing monsoon at that point of time. Further to above phased distribution of 100 ha in 25 ha. each restricted the dumping of overburden. To facilitate systematic dumping company has approached the State Govt. to allow rest of 75 ha. at one go vide letter no. SCM/GM (Ore)/1521/07, dated 10.10.2007. Copy of the above letter is attached as Annexure-29.

c) Buildup of Dumps:

Height and spread of the dumps will depend on the quantity of waste & the deign parameters of dumps based upon guidelines issued from various statutory agencies like Director Mines Safety, Indian Bureau of Mines and Ministry of Environment & Forests. As the present dumping area is supported by hill at one side and few places at both sides; to accommodate maximum quantity of overburden due to paucity of space and without affecting the stability, the dumps are designed to have maximum height of 90m in 4-9 stages of 10-25m each. Width of individual terrace is 10-15m. Further, suitable plantation will be done along the inactive dumps in concurrent manner to help in enhancing the stability of dumps.

The yearly build up of dumps from year to year has been shown in the form of plans and sections in drawing no. SCM/MP/08/12. The method of dumping has already been mentioned above.

The position of the associated mineral stacks within the lease hold area of 406 ha. has been shown in the Surface Plan (Drawing no.: SCM/MP/02/12). The details of such stacks is given in the table 7.3.1

Table 7.3.1

Dump Number	Dump type OB / SGM	Dumped on (mineralized / non-mineralized land)	Co-Ordinates		Area Sq m	Height	Angle	Quantity (Lakh Cum)
			X	Y				
1A & 1B	OB (Quartzite)	Non-mineralized	2870	1670	155000	74	28	38.01
	OB (Serpentinite)							19.19
2A	OB (Quartzite)	Non-mineralized	4000	2000	292500	75	28	1.00
	OB (Serpentinite)							57.71
	OB (Ni- Limonite)							53.59
2B	OB (Ni- Limonite)	Non-mineralized	2100	2100	75000	52	34	79.00
3	OB (Ni- Limonite)	Non-mineralized	3750	2740	30000	26	34	5.67
4	Sub-Grade Mineral	Non-mineralized	3150	1500	37500	32	30	8.07
5	Sub-Grade Mineral	Non-mineralized	3000	2100	55000	32	28	3.85
6	Sub-Grade Mineral	Non-mineralized	3850	2700	27500	17	29	0.25
1D	OB (Ni- Limonite)	Non-mineralized	3165	2030	10000	18	28	1.00
2-BY	Sub-Grade Mineral	Non-mineralized	3400	2000	29700	4	25	2.55
7	Mineral Reject / Fines	Non- Mineralized	2200	3000	17500	3.2	28	0.32
8	Mineral Reject / Fines	Non- Mineralized	2000	3050	2100	8	28	0.17
9	Mineral Reject / Fines	Non- Mineralized	1900	3025	4500	2.5	28	0.11

Year-wise generation of associated mineral/ overburden from both the quarry till 2017-18 has been mentioned in Table 7.1.2, 7.1.3 & 7.1.4. The manner in which associated mineral/ overburden to be stacked on the dumps is given below.

a) Dumping inside the lease area for back-filling of 272 L CuM will be on over an approx. area of 42 ha in OB II Quarry as shown in drawing no: SCM/MP/08/12 along A-A',B-B', C-C', D-D' & E-E' section. The stages of dumping at back-filled area in OB II quarry will also facilitate widening of main haul road leading to additional lease area.

b) Total quantity to be accumulated inside the additional area and to be backfilled in OB-II quarry during the plan period is given below in Table 7.3.2. The manner in which the associated mineral/ overburden to be stacked till 2017-18 is given in drawing no: SCM/MP/08/12.

Table 7.3.2
Year wise Overburden Generation & Management

YEAR	OB Generation (Cum)	At Kakudia Dump (Cum)	Backfilled in OB-II Quarry (Cum)	Remarks
2012-13*	1490000	1490000	0	Concurrent reclamation & rehabilitation measures to be taken up as soon as it reaches its holding capacity
2013-14	6275149	6275149	0	
2014-15	6843270		6843270	
2015-16	6727634		6727634	
2016-17	6789183		6789183	
2017-18	6871362		6871362	
TOTAL	34996598	7765149	27231449	

Associated mineral/ overburden generated in 2012-13 till part of 2014-15 shall be stacked in the additional area of 100ha till the space is exhausted. The manner in which overburden shall be stacked in the additional area of 100 ha is given in drawing no. SCM/MP/08/12 and sections along A-A', B-B', C-C'. D-D' & E-E'. Subsequently the overburden generated during part of 2014-15 till 2017-18 shall be backfilled in OB II quarry in a manner as shown in the drawing no SCM/MP/08/12. However if problem is faced to accommodate more overburden in the backfilled area of OB-II beyond planned period, company may seek for additional area out of applied area of 289 Ha for disposal of associated mineral/ overburden generated during 2018-19 onwards.

Protective Measures:

It is proposed that rehabilitation of the associated mineral stacks shall be taken up. As mentioned all the so-called overburden/ wastes of the past are essentially associated minerals as defined in the mining plan and there is every likely hood that these stacks of associated minerals shall be utilized in the not too distant future.

These stacks have already attained heights of almost 80 m at places due to dearth of space for accommodating associated mineral in the past several years after the reduction of the leasehold area. As may be seen from the surface plan, these stacks have been made in lifts of 10- 30 m each, leaving a berm of 10-15 m between the two lifts. These stacks are quite stable as per studies conducted by CMRI, Dhanbad and IIT, Kharagpur. The possibility of flattening the overall slopes of these dumps further with a view to increase their stability was explored but the same cannot be flattened any further since these stacks are located just beyond the ultimate pit limits of the quarries and any further widening of the berms would mean that the stacks would encroach into the mineralized area which shall be taken up for mining in the near future. Under the circumstances it is proposed that the following special measures shall be taken up for the rehabilitation of these associated mineral stacks:

1. In order to prevent the siltation and spreading of the solids by water erosion it is proposed that toe walls of at least 1.0 m height made by stacking of boulders shall be constructed at locations where the topography and drainage pattern so permits along the lease boundary. The year wise toe wall construction proposed is as shown in the table 7.3.3 and is shown in the drawing no. SCM/MP/10/12.

Table 7.3.3
Construction of Retaining Wall

Year	Additional Area of 100 ha.	Applied Area of 73.685ha.	Total	Cost	Breadth (m)	Height (m)
	Length (m)	Length (m)	Length (m)	(Rs Lakh)		
2011-12	1000	0	1000	15	1.00	1.00
2014-15	0	3500	3500	53		
Total	1000	3500	4500	68		

2. In order to further prevent the solids (particularly the finer particles from the nickeliferous limonite stacks) from flowing into the adjoining area it is proposed to provide garland drains at the toe of all such stacks wherever the topography so permits. As the garland drains exist to prevent the flow of solid waste to the adjoining leases, the construction of garland drains close to the lease boundary is not envisaged in the next five years. These garland drains shall be provided with stone partitions at intervals for arresting the flow of the solid particles. The drains shall ultimately lead to settling pits followed by an effluent treatment plant before the surface run off from these stacks is ultimately discharged out from the leasehold.

3. In order to prevent the formation of gullies along the slopes of these stacks it is proposed to cover these slopes with coir matting. Coir matting shall be done in the Ni-Limonite dumps were the chances of wash off is more. Coir matting over the OB-II quarry overburden material is not possible due to the very hard nature and sharp edges of overburden material.

4. It is further proposed that the slopes of all the associated mineral stacks shall be afforested by planting different varieties of grass and bushy plants along these slopes to further arrest the solids from wind erosion. The year wise stacks with the lease and additional area of 100 Ha. which have saturated shall be covered for bush plantation. The year-wise proposal for plantation and expenditure to be incurred in the additional area of 100 Ha, with in Mining Leasehold area of 406ha. and in applied area of 73.685 ha along the slopes of associated mineral stacks and in vacant land is as shown in drawing no. SCM/MP/10/12 and is given below in Table 7.3.4.

Table 7.3.45
Year wise proposed Afforestation and Expenditure

Year	Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total	
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)
2013-14	2.5	6250	8.00	20000	3.00	7500	13.50	33750
2014-15	1.6	4000	7.95	19875	3.15	7875	12.70	31750
2015-16	3.0	7500	8.05	20125	1.25	3125	12.30	30750
2016-17	3.0	7500	7.85	19625	1.05	2625	11.90	29750
2017-18	3.4	8500	8.05	20125			11.45	28625
TOTAL	13.5	33750	39.9	99750	8.5	21125	61.85	154625

The coir matting shall be done in the Ni- limonite overburden dump of OB-X quarry where the chances of wash off is more but may not be possible in overburden dump of OB-II quarry due to its very hard nature and sharp edges of overburden material.

Justification For Applied Area of 73.685 ha for Tailing Disposal:

An area of 70 ha was allotted within the existing mining lease area of 406 ha for the disposal of waste during the second renewal of the lease. 10ha out of the 70ha area for waste disposal was earmarked for tailing pond. This 10 ha area is now full and no further tailings can be accommodated within this area since the tailing pond has acquired the maximum safe height. The State Govt. at the time of takeover of the Bamnipal Plant had assured to recommend to Govt. of India for an additional area of 180 ha for the disposal of waste. So far an additional area of 100 ha only have been allotted for waste disposal. The existing mining lease area of 406 ha is surrounded in the north, east and west directions by other lessees. The applied area of 73.685 ha consisting of mostly non-forest land is located further south of the additional area of 100 ha and is therefore the only suitable area available in close proximity to the mine for the operations to be economically viable. The applied area is further south of the Mahagiri Hill range and outside the limits of the Sukinda Ultramafic Complex and hence devoid of any mineral. The requirement of land for tailing disposal has been estimated based on the space required to store the quantum of dry tailing to be generated over the life of the beneficiation plant, runoff management arrangements, storm water handling and water recirculation facilities.

8. USE OF MINERAL:

a) END USE OF MINERAL:

The year-wise use of mineral expected during the next five years is given in Table 8.1.1 below. The forecast given may change as the mine is catering to a number of industries and is dependent on the market conditions which changes very often within very short intervals of time. As shown in the table the requirement of chrome ore ROM is expected to remain close to 2.4 Million MT during the end of 2017-18. Similarly there would be production of chrome concentrate @6.5 Lakh tonnes/ annum during the end of 2017-18 from the chrome ore beneficiation plant.

Besides chrome, the mine shall be producing maximum @ 5 lakh MT / annum of pyroxenite to mainly cater to the needs of our Jamshedpur Steel Plant, Ferro Alloy Plant at Bamnipal and other plants under construction. The company has initiated process to set up additional furnace at FAP, Bamnipal and at its Rawmet plant. Besides company is looking to increase the value addition through FeCr production by having more no of conversion agreements. .

In addition, the Company is also setting up additional FeCr manufacturing capacities at Gopalpur & Nayagarh in Orissa and Stainless Steel units in India, provided a level playing field is provided by way of power availability at globally competitive rates.

Table: 8.1.1:
Requirement of Chrome ore
(All figures in lakh MT)

Purpose	Destination	Type of Ore	2013-14	2014-15	2015-16	2016-17	2017-18	Total
Captive	FAP, Bamnipal	Friable + Conc	0.87	1.48	1.45	3.21	3.58	10.58
		Lumpy	0.22	0.45	0.45	1.02	1.08	3.22
	FAP, Joda	Friable + Conc	0.25	0.21	0.21	0.21	0.22	1.10
		Lumpy	0.08	0.08	0.08	0.08	0.08	0.38
	TS Alloys	Friable + Conc	0.87	1.48	1.45	1.41	1.48	6.69
		Lumpy	0.26	0.53	0.53	0.53	0.53	2.38
	Gopalpur & Nayagarh	Friable + Conc	1.74	1.48	1.45	1.41	2.50	8.57
		Lumpy	0.53	0.53	0.53	0.53	0.89	3.00
	Conversion	Friable + Conc	1.81	1.34	1.31	1.28	1.35	7.10
		Lumpy	0.47	0.41	0.41	0.41	0.41	2.10
	Beneficiation Plant	Friable	10.31	10.94	11.39	11.39	11.39	55.42
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
	Jamshedpur Works and Joda	Friable	0.00	0.00	0.00	0.00	0.00	0.00
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
	Refractory Plants	Friable	0.51	0.51	0.51	0.51	0.51	2.53
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
	TSKZN, South Africa	Friable	0.00	0.00	0.00	0.00	0.00	0.00
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
Domestic Commitments		Friable	0.00	0.00	0.00	0.00	0.00	0.00
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
		Concentrate	0.00	0.00	0.00	0.00	0.00	0.01
Export Commitments		Friable	0.00	0.00	0.00	0.00	0.00	0.00
		Lumpy	0.00	0.00	0.00	0.00	0.00	0.00
		Concentrate	1.10	1.10	1.10	1.10	0.60	5.00
Total		**Total Friable**	**16.37**	**17.43**	**17.76**	**19.41**	**21.02**	**91.98**
		Total Lumpy	**1.56**	**1.99**	**1.99**	**2.56**	**2.98**	**11.08**
		Total Chrome Ore	**17.92**	**19.42**	**19.75**	**21.97**	**24.00**	**103.06**
		Total Concentrate	**5.39**	**5.72**	**5.95**	**5.95**	**5.95**	**28.95**
Grand Total Finished Prod. (Friable and			**7.18**	**7.98**	**7.15**	**10.85**	**12.39**	**45.56**

The company since long has also been registered as a primary mining company under para 3 (b) of the Memorandum and Articles of Association. Accordingly, the company besides meeting its own captive requirements had been in the business of sale of chrome ore and concentrates since long both for other domestic consumers as well as export. However, the company over the years has increased its captive consumption manifold and continues to do so by progressively adding and modernising both its downstream plant capacities and port handling facilities. The company has also been doing further value addition of its chrome ore produce through other plants under conversion agreements and also provides other raw material requirements such as coke for the same. Thus, while the company aspires to achieve the future end use of the chrome ore and concentrates it produces from this mine as given above under Table 8.1.1, it may under certain very special circumstances - while still remaining within the legal framework and rights, resort to the sale and export of chrome ore and concentrates of certain quantities due to either prevalent force majeure conditions, unforeseen market forces or technical difficulties; in the interest of business continuity/ sustenance and overall benefit to the economy and country. Some of such conditions may be:

- Severe recession in the market leading to the non-viability and closure of the company's captive plants and other plants under conversion agreement – leading to huge build-up of stocks of ore and concentrates;
- Directives from the Government for the sale of ore and concentrates to other consumers in the event of non-availability of the mineral from other domestic sources and for the overall benefit of the country - like defence purposes, or meeting such other economic and livelihood interests of the local community;
- Delays in the commissioning of the new downstream captive plants due to non-availability of land, power, water or other resources - leading to build up of huge stocks of mineral at the mine and stockyards;
- Stoppage of any of the captive downstream plants or plants under conversion agreement, over extended periods of time due to certain force majeure conditions (like strike, lockout, earthquake, flood, transport bottlenecks, blockades, other natural and accidental calamities, shortfall in electrical power, major breakdowns & non-availability of spares, etc.) – leading to build up of huge stocks of ore and concentrates at the mine and stockyards;
- Past long-term commitments of the company with other consuming plants in India & abroad that have been set up with technology that was developed based on specific quality parameters (both physical and chemical) of the ore and concentrates produced and available only from this mine;
- Absolute non-availability of ore at a reasonable price for operating the plants owned by the company in other countries, which may lead to closure of such plant;
- Build-up in stocks of certain specific grades and quality of chrome ore and concentrates which cannot be consumed by the company in its own captive plants due to certain technical limitations in the plant & process capabilities and/ economic non-viability.

b) PHYSICAL & CHEMICAL SPECIFICATION:

The user industries have of late become more demanding with respect to both chemical and physical specifications of their requirements. Besides, Cr_2O_3 content, the underlying needs of the user industries in the metallurgical sector are more with respect to the Cr/Fe ratio. With practically no charge chrome production in our country, the demand for charge-chrome grade ore (+38 % - 44 % Cr_2O_3) as defined in the approved mining plan has virtually become negligible. The charge-chrome grade ore cannot be used by blending with higher grades also due to very low Cr/Fe ratio. Part of the salable ROM is screened and friable ore lump of +25-75mm size is segregated which is directly fed to the furnace by ferro chrome users to improve the reduction process. The changed specification of ferro chrome grade chrome ore are as follows:

Ore Type	Cr_2O_3 content:	Cr / Fe Ratio	Size range :
Friable ore - (OB X Quarry)	+50 – 52%	min. 2.7	+10 – 75 mm. (preferred)/ -25 mm / +0 – 75 mm.
Friable lumpy ore (OB X Quarry)	+48 – 50%	min. 2.7	+25 – 75 mm.
Hard lumpy ore (OB II Quarry)	+33 – 38%	min. 2.9	+10 – 100 mm/ +40 – 100 mm/ +10 – 40 mm.

c) BLENDING:

Efforts are made to meet the customer requirement both internal as well as external for physical and chemical specification as agreed through memorandum of understanding. While going deeper the chrome ore grade has been improving which may not be suitable for ferro chrome making directly. So high grade ore is suitably blended with lean ore to get requisite grade. Geologist using the information obtained from blast hole analysis, channel sampling coordinate the ore raising in such a manner that customer requirements are met.

9. OTHER :

a) SITE SERVICES:

To facilitate the work of mechanized mine a number if site services are provided in the mine. These are as follows:

i).Workshop:

The mine is provided with a covered workshop consisting of the following shops;

1) Machine shop
2) General working
3) Motor vehicle garage
4) Tool room, office
5) Electrical section
6) Engine overhauling section
7) Quarry dewatering

Apart from above the central workshop as described above, the mine has a field workshop know as O.K Line to take care of all the repairs and maintenance of heavy mobile equipments. Additionally mobile maintenance van takes care of the maintenance problem of heavy earth moving equipments scattered over various locations. The above facility shall continue to be used to maintain the HEMM to be deployed for underground mining.

ii).Power Supply:

At present electric supply comes to Sukinda Chromite Mine from CESU (Central Elcricity, Service & Utilities) through Kaliapani substation at 33 KV. In order to operate COB Plant, industry supply is also being supplied by Kalipani substation. Power supply is stepped down from 33 KV and distributed to 3.3 KV to different sections in the beneficiation plant. The power at 3.3 KV is stepped down to 440 V, 3 phase system and used for medium tension drivers and load. The supply to the colony area is taken at 3.3 KV from 33/3.3 KV substation and stepped down to 400V, 3 phase at two substations in the colony for distribution to avoid voltage drop.

CESCO shall be approached to enhance power supply to match underground power requirements. With underground mining operation at place the power requirement shall increase to 14.35 MVA. The breakup of energy requirement in future shall be as per Table-9.1.1 given below. More DG sets of 1000 KVA capacity each is proposed to be installed to take care of the emergency supply to the underground mine.

Table-9.1.1

Requirement of Electricity (MVA/ annum)

Sl No.	Consuming Head	Existing Requirement (MVA/ Annum)	Future Requirement (MVA/ Annum)
1.	Open cast mining & Plants	3.3	2.0
2.	Colony	1.3	1.3
3.	Shaft winder (Rock)	0.0	2.8
4.	Man winding	0.0	2.1
5.	Ventilation Fans	0.0	3.6
6.	Underground ventilation	0.0	0.5
7.	Underground Drills	0.0	0.45
8.	Dewatering	0.0	1.6
9.	Total	4.6	14.35

iii).Water Supply:

The supply of water to the camp residence is from Damsala nallah outside our lease at a distance of 1.5 km from the camp. Using an intake well, water is lifted by a diesel pump set through a 6 inch diameter pipeline system to the settling tank of 40,000 gallons capacity, located in the camp. From the settling tank water is processed through a filtration plant of 10,000 gallons/ hour capacity. The filtration plant operations include.

1) Alum mixture by flocculator
2) Settling
3) Slow sand gravity filtration
4) Chlorination

The filtered water is stored in reservoirs of 15,000 gallons and 8,000 gallons capacity respectively. The filtered water so stored is distributed to the camp residence through a network of pipeline. A standby pump of similar capacity is also provided in the pump house. The same system shall be extended to service the underground workings.

iv).Office:

The mine has got a centrally air-conditioned modern office to accommodate various section such as Administrative Block, Civil Engineering section, Computerized Mine planning, Surveying & Environment Cell, Geological, Personnel, Accounts, Time Office and cash office also has a conference room to take care of various meetings. Apart from above there are various site offices located at COB Plant, Mining complex, Purchase and stores, Engineering complex to take care of the site specific issues. Underground pit office shall come up in near future with installation of underground infrastructure.

v).Vocational Training Centre:

The mine has a Vocational Training Centre for imparting training to its employees and contract labours in various trades. It has got 3 nos. of lecture hall, one model room, trade test workshop and other training aids like, TV, CDs, Slide Projector, LCD Projector, Computers, 166mm film projector etc. Regular classes for basic, refresher, special training, fire fighting, quality management, leadership skill etc are conducted. In addition to seminars visit talks, quality circle presentations and environmental awareness programme are conducted. The same facility shall be used for underground related trainings.

The vocational training centre is also equipped with a “Knowledge Center” containing more than 10,000 books and journals on various subjects for the benefits of employees and their wards.

vi).Hospital:

There is a 32 bedded modern hospital having an ICU. It has also got X-ray facility and pathological laboratory to take care of medical needs of camp residents free of charge. Two ambulances are provided to carry patients from the camp or worksite to the hospital or to send the patients to better hospitals in case of emergencies. Medicinal facility is also extended to community people at very nominal charges.

The mine is also provided with adequate number of first aid stations as per mines rules at various worksites.

vii).Canteen:

To cater the requirement of employees, the mine is provided with a modern canteen which serves all the employees. Beneficiation plant has its own canteen and supplies snack to COB Plant workers during all the three shifts. The main canteen provides hot meal during the lunch time. A Canteen Managing Committee constituted as per the provisions of Mine Rule, looks after the management of canteen facility and sorts out various day to day problems.

viii).Roads:

Company has renovated the 20KM stretch of Tomka- Mangalpur road at the cost of Rs 11.5 Crores. Well-developed roads within and to the lease hold already exist. All the roads within the colony have been blacktopped or concreted. Other mining roads shall be developed in due course, for which adequate and suitable mining equipments (viz. dozer, grader etc.) are already there.

b) EMPLOYMENT POTENTIAL:

The details of present manpower and as envisaged during the end of the planned period is given in Table 9.2.1.

Table 9.2.1
Details of Manpower

Category of Personnel	No.of Heads (Present-Opencast)	No.of Heads (Future- Underground)
Supervision	112	136
Highly Skilled	235	342
Skilled	185	166
Semi Skilled	30	463
Un Skilled	6	587
Total	568	1694

Labour supply and skill: Presently around 568 persons are employed directly by the company at Sukinda Chromite Mine. Around 1960 persons are employed indirectly to provide various essential services like transportation, loading, security etc. A significant proportion of the employees under indirect employment are attending to their duty from the local villages. The colony has the infrastructure to cater to the needs of around 5000 people and presently around 3200 persons reside within the colony. With the introduction of underground mining, there may be a temporary increase in the population within the colony to around 5000 people which after development is likely to again come down to around 4000 persons.

10. MINERAL PROCESSING/ BENEFICIATION :

The ROM ore from the various quarries is required to be either processed manually or in the various crushing and screening plants for meeting the physical specifications or upgraded in the Chrome Ore Beneficiation Plant in order to meet the requirements of both physical and chemical specifications of the various customers.

In course of winning medium & high grade ore, the mine used to generate certain quantity of beneficiable grade ore having +10% Cr_2O_3. In order to conserve the scare mineral, it was felt necessary to beneficiate the lean ore and accordingly the company established a Chrome Ore Beneficiation Plant at Sukinda in the year 1990 with an installed capacity of 1.10 lakh tonnes of chrome concentrate per annum. Subsequently lot of modification was done and lot of process improvements were done to enhance the capacity to 7 lakh tonnes of chrome concentrate per annum with a feed grade of 39% Cr_2O_3. The process flow chart showing the mass balance is given as figure 10.1.

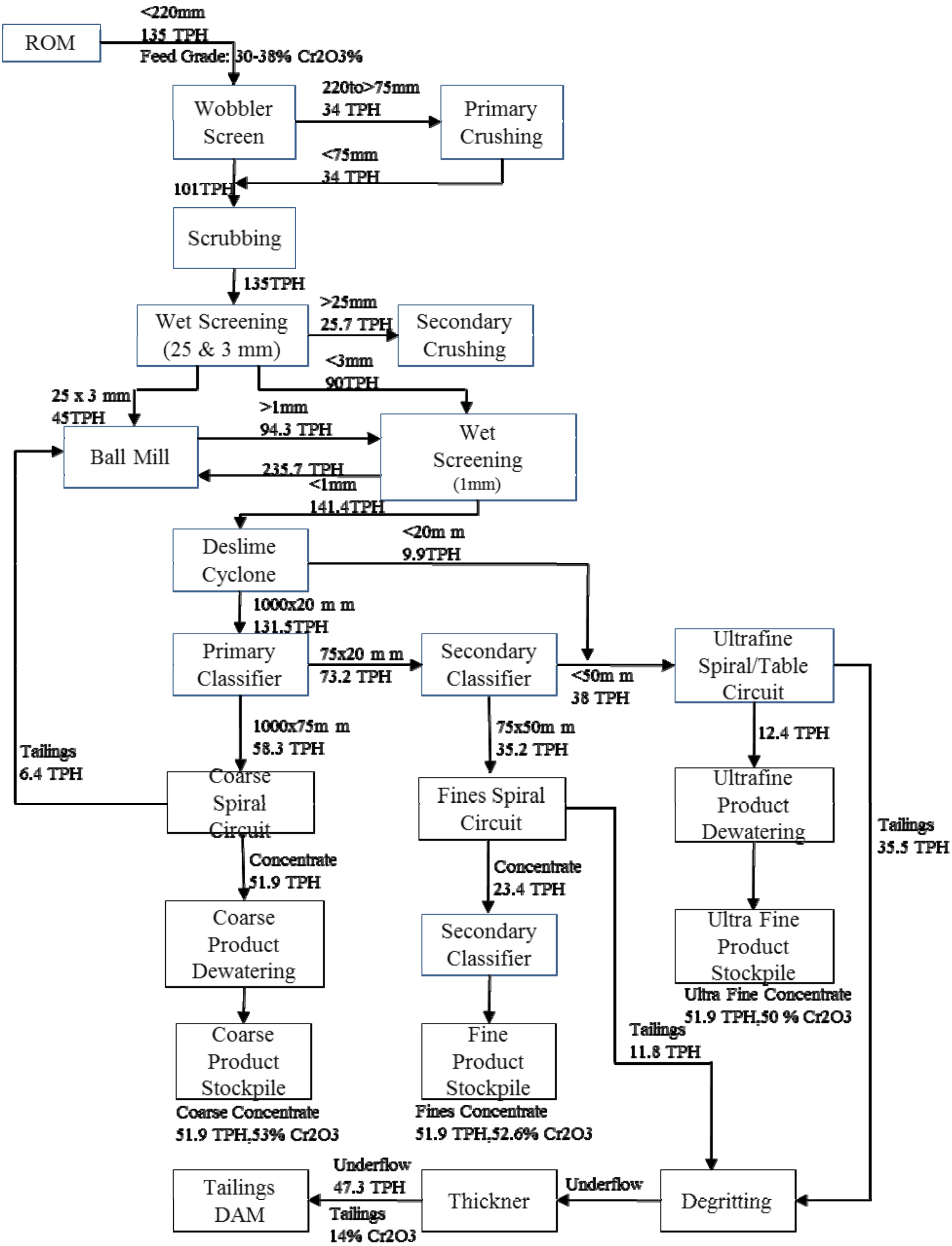

Figure: 10.1: Process Flow chart of Chrome Ore Beneficiation Plant showing mass balance

10.1 Grade, Tonnage of Feed & Product:

Screening and crushing of the friable ore is now done at Lumpy ore Processing Plant. The Lumpy Ore Processing Plant has a capacity of 60 MT / Hr. this is basically a screening and crushing plant and the recovery of –75 +10 mm product is around 30%. Part of the fines (–25 mm) of marketable grade is despatched as a product depending on customer requirements, while part of it that is mixed with the nickeliferrous limonites from the contact zone and having inferior grade is sent to the beneficiable grade stacks. ROM ore from OB X quarry which is extremely fragile in nature and having +46% Cr_2O_3 is proposed to be either manually sized, dressed and sorted in the nearby stack yard or fed to the Lumpy Ore Processing Plant depending upon the customer requirements with respect to size specifications. The recovery after manual processing is about 95% for the –75+0 mm product. The ROM of lower grades or beneficiable grade of friable ore is sent to the low grade stacks / blending yard from where it is fed to the Chrome Ore Beneficiation Plant having an average feed grade of 30% to 38% Cr_2O_3.

The ROM ore from the OB II quarry (Lumpy) having a grade of +30% Cr_2O_3 shall be fed to the Lumpy Ore Processing Plant. The plant is located adjacent to the Chrome Ore Beneficiation Plant and has a capacity of 60 MT/Hr. The process involves crushing and screening. Wet screening is done to wash the clayey material adhering to the ore. The recovery of +40 –100 mm and +10 –40 mm (finished product) put together is around 85%, while that of fines (-10 mm+100 mesh) is around 10%.

The remaining 5% (-100 mesh) material goes as slime loss to the tailings thickener for the Chrome Ore Beneficiation Plant. The ROM lumpy ore with –30% Cr_2O_3 incidental to the above shall be stacked in the low grade stacks. Some amount of +33% Cr_2O_3 shall be produced manually from such low grade stacks by manual dressing & sorting which has been referred in the text as “segregation”. The flow chart showing the flowage of various grades of mineral and mineral rejects and waste, , their source and consumption point are shown in the Fig. 10. 2.

Other than chrome ore being mined from the two quarries viz. OB II & OB X quarry, a quarry is being developed for producing pyroxenite. The pyroxenite quarry is part of the OB II quarry. Pyroxenite ore forms as a part of OB to the existing OB II quarry. Shovel-dumper combination along with deep hole blasting shall be done for raising the pyroxenite ore. The ROM shall be crushed and screened in the crushing and screening plant. The plant has a capacity of 40 MT/hr. The recovery of the finished pyroxenite product of –75+10 mm is around 88%. The remaining fines are stacked as mineral rejects.

Apart from the fresh generation of beneficiable grade ore from the quarries the ultimate feed to the plant is achieved by suitably blending the same along with the mineral rejects and available old stocks of beneficiable grade ores. In view of the proposed change in the specifications of beneficiable grade ore, the average feed grade to the beneficiation plant is estimated to be 30% to 38% Cr_2O_3.

The expected feed quantities of beneficiable grade ore to the Chrome Ore Beneficiation Plant (COBP) from various sources and expected concentrate production is as given in Table 10.1.1 below. The final concentrate production is given in dry basis. However the despatchable concentrate in wet basis may be estimated adding 12% moisture in it.

Table 10.1.1
Beneficiable grade (BG) ore feed for Concentrate production
(All figures in MT)

Year	BG ROM FROM QUARRY	MINERAL REJECTS	FEED FROM OLD BG STOCK	TOTAL FEED TO COBP (Wet)	MOIS TURE (%)	TOTAL FEED TO COBP (Dry)	PROD. CONC. (Dry)	PROD. CONC. (Wet)
2012-13*	212963	10000	25000	247963	12	218207	113468	127084
2013-14	1031145	0	0	1031145	12	907407	480926	538637
2014-15	1094276	0	0	1094276	12	962963	510370	571615
2015-16	1138937	0	0	1138937	12	1002264	531200	594944
2016-17	1138937	0	0	1138937	12	1002264	531200	594944
2017-18	1138937	0	0	1138937	12	1002264	531200	594944
TOTAL	5755195	10000	25000	5790195	12	5095369	2698364	3022168

*The figure for 2012-13 is from 12.01.2013 till 31.03.2013.

The yearly quantities and average grades of feed, concentrates & tailings, yield / % recovery of concentrate by weight expected by virtue of proposed changes / modifications & improved operational efficiencies is given below in Table 10.1.2. The feed grade and concentrate grade may vary depending upon the customer's requirements. The year wise proposed figures are from 2012-13 (From 12.01.2013 to 31.03.2013) till 2017-18.

Table 10.1.2
Year wise COB feed, Concentrate production & Tailings grade

Year	Feed Grade % Cr_2O_3	Feed Qty DMT	Concentrate Grade % Cr_2O_3	Concentrate Production DMT	Tailing Grade % Cr_2O_3	Tailing Qty DMT	Yeild % By Wt
2012-13*	30-38	218207	50-54	113468	14 - 16	117704	44-58
2013-14	30-38	907407	50-54	480926	14 - 16	426481	44-58
2014-15	30-38	962963	50-54	510370	14 - 16	452593	44-58
2015-16	30-38	1002265	50-54	531200	14 - 16	471065	44-58
2016-17	30-38	1002265	50-54	531200	14 - 16	471065	44-58
2017-18	30-38	1002265	50-54	531200	14 - 16	471065	44-58
TOTAL	30-38	5445410	50-54	2886067	14 - 16	2559343	44-58

*The figure for 2012-13 is from 12.01.2013 till 31.03.2013.

It is proposed that the tailings from the COBP shall be further re-treated alongwith the old tailings that have been stacked within the old tailings pond. The details of such re-treatment facility to be commissioned from 2015-16 has been furnished later under Para 10.3 for Proposed Change in Technology.

Table 10.1.2.1
Average feed grade & quantity for tailing re-treatment

Year	Tailings Grade From COB % Cr_2O_3	Tailings Qty. From COB DMT	Old Tailings Grade % Cr_2O_3	Tailings From Old Tailings Pond DMT	Avg. Feed Grade To Retreatment Plant, % Cr_2O_3	Total Feed Qty. For Retreatment, DMT
2015-16	14 - 16	471065	18-22	75000	15-17	546065
2016-17	14 - 16	471065	18-22	75000	15-17	546065
2017-18	14 - 16	471065	18-22	75000	15-17	546065

Table 10.1.2.2
Year wise Concentrate Production from Tailings Re-treatment Plant & Final Tailings Grade

Year	Avg. Feed Grade To Retreatment Plant, % Cr_2O_3	Total Feed Qty. For Retreatment, DMT	Concentrate Grade % Cr_2O_3	Concentrate Production From Tailing Retreatment, DMT	Final Tailings Grade After Tailings Retreatment, % Cr_2O_3	Final Tailings Qty., DMT
2015-16	15-17	546065	38-42	118769	8-10	427296
2016-17	15-17	546065	38-42	118769	8-10	427296
2017-18	15-17	546065	38-42	118769	8-10	427296

Table 10.1.2.3
Year wise net concentrate production

Year	Concentrate Production From Existing Plant, DMT	Concentrate Production From Tailing Retreatment, DMT	Total Concentrate Production, DMT
2012-13*	113468	0	113468
2013-14	480926	0	480926
2014-15	510370	0	510370
2015-16	531200	118769	649969
2016-17	531200	118769	649969
2017-18	531200	118769	649969
TOTAL	2886067	356307	3242374

*The figure for 2012-13 is from 12.01.2013 till 31.03.2013.

Capacity of the COB Plant:

The capacity of COB Plant was validated by the IBM after the modernization of the beneficiation circuit in 2005-06 to be 7.0 lakh tonnes/ annum. This report on capacity validation studies of the chrome ore beneficiation plant of October, 2005 (Report no. IBM/NGP/R.I. N.O. 1681) states that the plant can produce about 7 lakh tonnes/ year of chrome concentrate (dry basis) assaying about 50-52% Cr_2O_3 with weight % yield of 60-65%, by treating a ROM ore assaying 38-40% Cr_2O_3 and plant availability of 90% i.e. 7800 hours/ year, with a plant feed rate of 170 tph (wet basis) and 150 tph (dry basis).

Presently though, the ROM ore feed to the plant is assaying 33-35% Cr_2O_3. With this lower grade of ROM feed to the plant the feed rate is estimated to be around ~ 125-130 tph (dry basis). Further due to the lower feed grade, the weight % yield being achieved presently is ~ 53-55%.

Thus, considering the above, the plant capacity for the given conditions = 7800 x 127 x 0.54 = 5, 34,924 tonnes/ year, i.e. ~ 5.3 lakh tonnes/ annum of chrome concentrate (dry basis) assaying about 50-52% Cr2O3.

The COB Plant capacity with the commissioning of the proposed re-treatment of tailings facility from 2015-16 is expected to be around 6.5 lakh tonnes/ annum as per estimates made under Tables 10.1.2.1, 10.1.2.2 & 10.1.2.3.

The quantities of friable ore feed to the Mechanised Processing Plant and that for manual dressing & sorting has been given in Table 5.3.2. The quantities of various products (friable ore and lumpy ore) expected on the basis of the recoveries are given in Table 10.0.3 & 10.0.4 below. The figures in the Table 10.1.3 & 10.1.4 for the year 2012-13 are from the date 12.01.2013 to 31.03.2013.

Table 10.1.3
Material Balance for OB-X Ore
(All figures in MT)

Year	ROM to Friable Ore Processing Plant (+42%)	ROM to Stackyard for Manual Dressing & Sorting (+42%)	PROD. MANUAL PROCESS		PRODN. MECH. PROCESS			TOTAL FINISHED ORE
			-75 mm +0 mm	MINERAL REJECT	-75 mm + 10 mm	- 25 mm	MINERAL REJECT	
2012-13	135630	2768	2713	55	108504	27126	0	138343
2013-14	593364	12109	11867	242	474691	118673	0	605231
2014-15	635898	12978	12718	260	508719	127180	0	648616
2015-16	624345	12742	12487	255	499476	124869	0	636831
2016-17	829505	16929	16590	339	663604	165901	0	846095
2017-18	937664	19136	18753	383	750131	187533	0	956417
TOTAL	3756406	76662	75128	1534	3005125	751282	0	3831533

The yearly quantity of feed to Lumpy Ore Processing Plant (LOPP) and the total marketable lumpy ore production including the production through manual segregation from the old low-grade ore stacks based on the recovery is given in Table 10.1.4 below.

Table 10.1.4
Material Balance for OB-II Ore
(All figures in Lakh MT)

Year	FEED TO LOPP +30% Cr_2O_3	PROD.FROM PLANT			SEGRE GATION	TOTAL LUMPY FINISHED ORE
		+10 mm	MINERAL REJCT -10 mm	SLIME LOSS		
2012-13*	25438	21622	2544	1272	5000	26622
2013-14	115059	97800	11506	5753	45000	142800
2014-15	143492	121968	14349	7175	60000	181968
2015-16	58056	49348	5806	2903	60000	109348
2016-17	260106	221090	26011	13005	60000	281090
2017-18	305176	259400	30518	15259	71000	330400
TOTAL	907327	771228	90734	45367	301000	1072228

*The figure for 2012-13 is from 12.01.2013 to 31.03.2013.

10.2 Disposal of Tailings:

The amount of tailing generated and grade is as given in Table 10.1.2. The tailing used to be stored in the tailing pond located inside the leasehold. However, there is a dirth of space for disposal of tailings in the existing tailing pond. With the reduction of the mining lease area in 1996 during its second renewal, the tailing pond went outside the granted area of 406 ha. A new tailing pond was established within the reduced lease covering an area of 10 ha. The tailing dam was being heightened every year to accommodate the tailing generated from the COB plant.

IIT, Kharagpur was engaged to assess the holding capacity of the tailing pond and the stability of the dam. In their report, IIT, Kharagpur has recommended for the tailing dam heightening to be done to a maximum height of 215 mRL. Presently the dam has already reached a height of 212 mRL and the operation of above tailing has now been discontinued.

The present tailing dam is not having sufficient capacity to cater for the requirement till the end of planned period. Company has therefore has identified and applied for an additional area of 73.685ha having 65.315 ha of non-forest Government land as shown in the drawing no. SCM/ MP/05/12 for future storage of the tailing which is under process. It may take a further period of at least two years to acquire this additional land.

Further we also have a proposal to install parallel plate filters in the Beneficiation Plant in order to separate out the water from the tailings slurry/ slimes within the plant premises. With the introduction of these parallel plate filters within a year, the tails shall be in cake/ solid form which can be transported in dumpers to the proposed additional area to be acquired for the purpose. Pilot scale test for these press filters have already been successfully conducted. It is expected that this new method of solid disposal shall substantially reduce the threats associated with the stability of the conventional tailing ponds/ embankments.

Presently the slimes/ tailings is being stored in the pyroxenite section and the water is being re-circulated back to the plant for its reuse.

During the planned period the dry tailing shall be stored in the applied area of 73.685ha. As per the earlier modification to the Scheme of Mining it was envisaged that the area would be used till the end of the Scheme period. However, it may take still some more time for acquiring the area.

Therefore as a contingency measure, it is proposed that the solid tailings obtained after filtration as mentioned above shall be stored in the same area for another two years upto 2014-15. Thereafter it is proposed to rehandle the tailings from this area to the applied area of 73.685 ha during 2015-16. The manner in which the tails shall be stored is shown in Drawing no. SCM/MP/11/12.

Presently one study is going on detailing the future tailing generation, its management, storage at the applied area, runoff management, confinement and mitigation measures to be followed while transportation of dry tailings. Details are also being working out about management of tailing after the closure of plant with method of stabilization, rehabilitation. R&D study is also going on exploring the possibility of reuse of tailings.

10.3 Beneficiation Process:

In order to conserve the scarce mineral, the company has set up the largest Chrome Ore Beneficiation Plant within the lease in the year 1990. A totally wet gravity separation process is used for the beneficiation of the ore. Apart from the fresh generation of beneficiable grade ore from the quarries the ultimate feed to the plant is achieved by suitably blending the same along with the mineral rejects and available old stocks of beneficiable grade ores. With the underground mining the contact zone material/ mixed ore shall be feed to COB Plant for beneficiation. The average feed grade to the beneficiation plant is estimated to be 30% to 38% Cr_2O_3.

The manner in which the beneficiation is being carried out at chrome ore beneficiation plant is described below. There are mainly three circuits in COB Plant. The flow sheet of the beneficiation process is given in the diagram in Fig: 10.1.

1. Feed Preparation Circuit
2. Feed Classification Circuit
3. Beneficiation & Dewatering Circuit

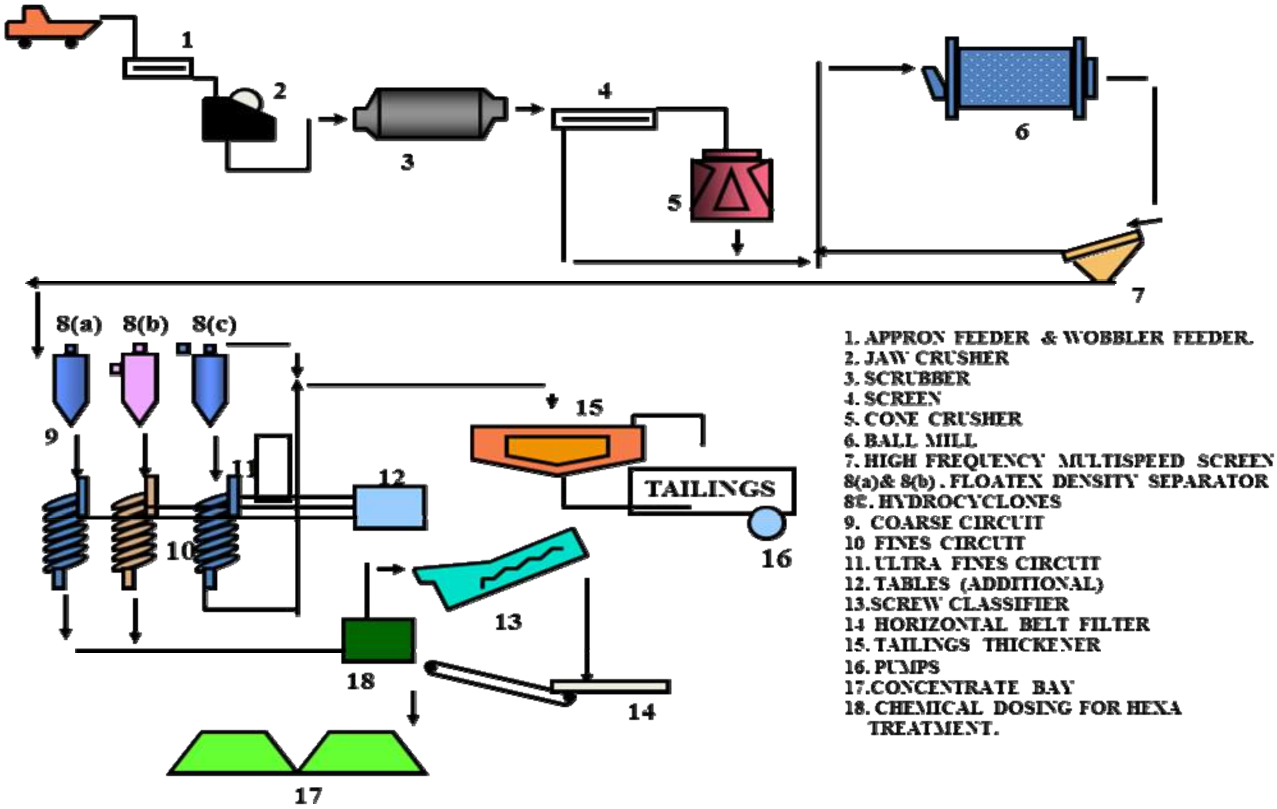

Fig. 10.1: Flow Chart: Beneficiation Process

Feed Preparation Circuit

The material from COB stackyard (also known as blending yard) is fed on to Stationary Grizzly, with the opening of 220 mm. The oversize (> 220 mm) material is broken with rock breaker to make it <220 mm, and along with undersize (< 220 mm) is fed into primary hopper. Then the material is carried by primary apron feeder to C1 A conveyor & then to secondary hopper. Secondary apron feeder feeds the material on to wobbler feeder with opening of 75 mm. The oversize material (>75 mm) is sent to DTJ crusher for crushing to < 75 mm size & then fed this material along with undersize (-75 mm) material of wobbler feeder to a rotary scrubber for washing of the ore. The scrubber discharges the material on to the double deck screen with top deck of 25 mm & bottom deck of 3 mm.

The > 25 mm material is sent to cone crusher for further crushing & again the product of crusher is sent to the screen. The <25 mm and > 3 mm particles are sent to ball mill for grinding to < 1 mm. The ball mill product along with <3 mm particle of screen is send to High Frequency Screen of 1 mm aperture. The oversize (> 1 mm) is sent to ball mill & undersize (<1 mm) is sent to hydrocyclone no 1 (Feed to Classification Circuit).

Feed Classification Circuit

The overflow of hydrocyclone is sent to ultrafines beneficiation circuit & underflow is sent to primary floatex. The overflow of primary floatex is sent to secondary floatex & underflow is sent to coarse beneficiation circuit. The overflow of secondary floatex is sent to ultrafines beneficiation circuit & underflow to fines beneficiation circuit.

Beneficiation & Dewatering Circuit

Spiral Concentrators are used for the beneficiation of coarse, fines & ultrafines material. The dewatering of the coarse & fines concentrate, obtained from spirals, is done in screw classifier. The dewatering of ultra fines concentrate takes place in vacuum belt filter. Thus, three products i.e. Coarse, Fines & Ultra Fines Concentrate are obtained. The reject material i.e. tailing is send to thickener for dewatering & thus thickened material is pumped to the tailing pond & the recovered water is circulated back to the beneficiation circuit.

The plant utilizes the low grade chrome ore from mines, and beneficiate it to produce chrome concentrate of different grades as per customer requirement. The plant produces different grades of chrome concentrate by adjusting / controlling various process parameters in the circuit, such as operating single stage or double stage spiral cleaning, adjusting the circuit spirals settings, adjusting / controlling floatex density separator process parameters, etc,.

Proposed change in technology:

In order to further bring down the tailings losses from the current level of 14-16% $Cr_2O_3$to the targeted level of < 10% Cr_2O_3, the Company has undertaken pilot scale studies for tailings beneficiation in collaboration with IIMT, Bhubaneswar; DRA and Mintek, South Africa together with our own R&D Dept. at Jamshedpur. Based on such studies the Company has a proposal to set up the tailings retreatment facility by 2014-15 having a feed capacity of 70 tonnes/ hour.

With this it shall be possible to upgrade the Cr_2O_3 % in the tails from a level of 14-18% to 38-42% in the final concentrate with tailings of <10 % Cr2O3.

The block diagram of the forthcoming tailings retreatment plant circuit after the tailings thickener in the present plant is given in Fig: 10 below.

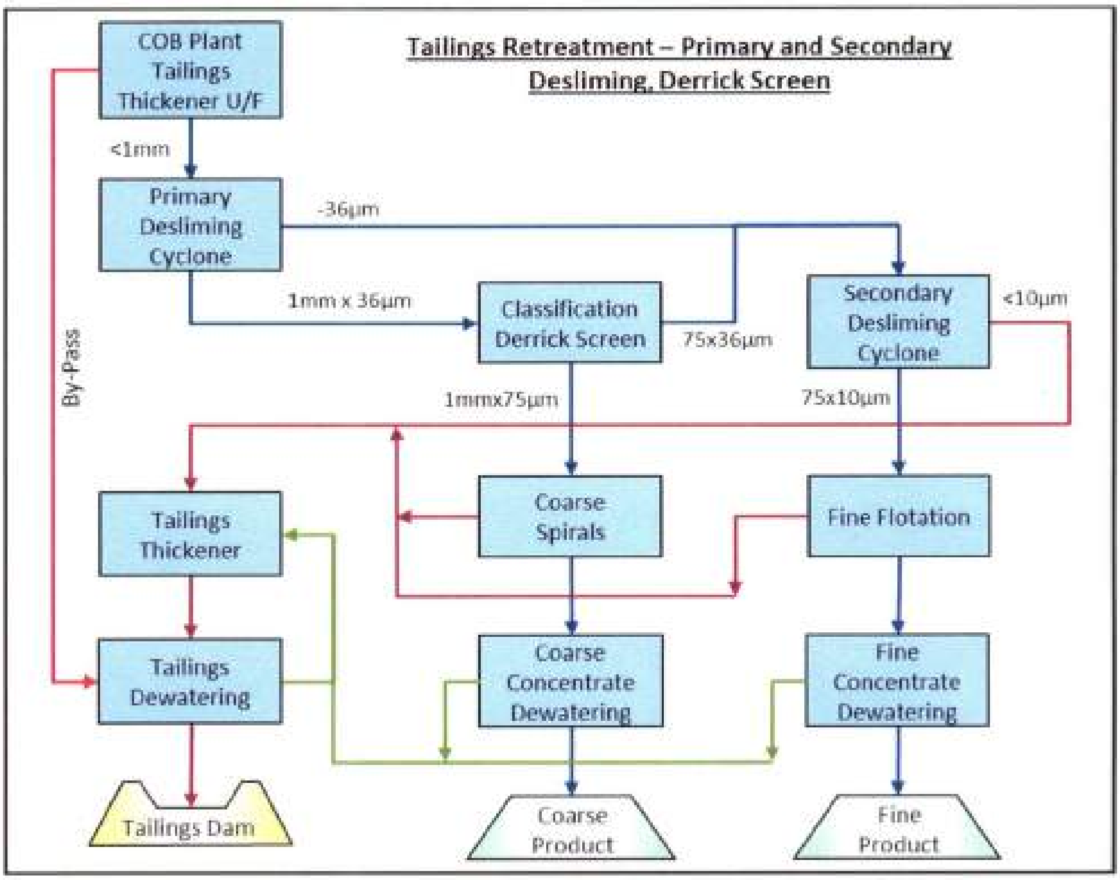

Figure 10: Primary and Secondary Desliming, Classification Derrick Screen

The water from the tailings is further proposed to be filtered by the introduction of parallel plate press filters. 3 such units having a capacity of 27 t/ hr each is proposed to be installed. The tails after dewatering shall then be in cake form and shall then be transported in dumpers to the proposed disposal site. Fig: 10.3

The schematic layout of the manner in which these parallel plate press filters shall be installed within the Chrome Ore Beneficiation Plant is given in Fig: 10.4.

10.4 Requirement of Chemical & Water:

The beneficiation process is totally based on gravity separation method. No chemical is being used during the beneficiation process. For the better and faster settling of tailings in the thickener, Indifloc-27 Flocculent is being used. The annual consumption rate is 6 tonnes. Similarly Nalco based organic reagent is being used in the screw classifier to treat the hexa chrome present in the final concentrate. The annual consumption rate of the above chemical is 20 tonnes. Similarly around 200 tonnes of ferrous sulphate is also being used at the chrome concentrate stackyard to treat the hexavalent chromium present if any in the final product. There would be a 20% increase approximately in the above chemicals on event of increase in production of chrome concentrate @ 6.5 lakh MT/ annum at the end of plan period.

The COB Plant is taken various in process measures to reduce the water consumption during beneficiation. The overflow water from the tailing thickener is being reused in the plant. Similarly the clean water from the tailing dam is also recycled back and reused in the COB Plant. Presently 150 Cum/hr of make water is being used in the COB Plant. The complete requirement is met from the water being pumped out from the quarries.

Fig. 10.3: Photograph of the parallel plate press filter

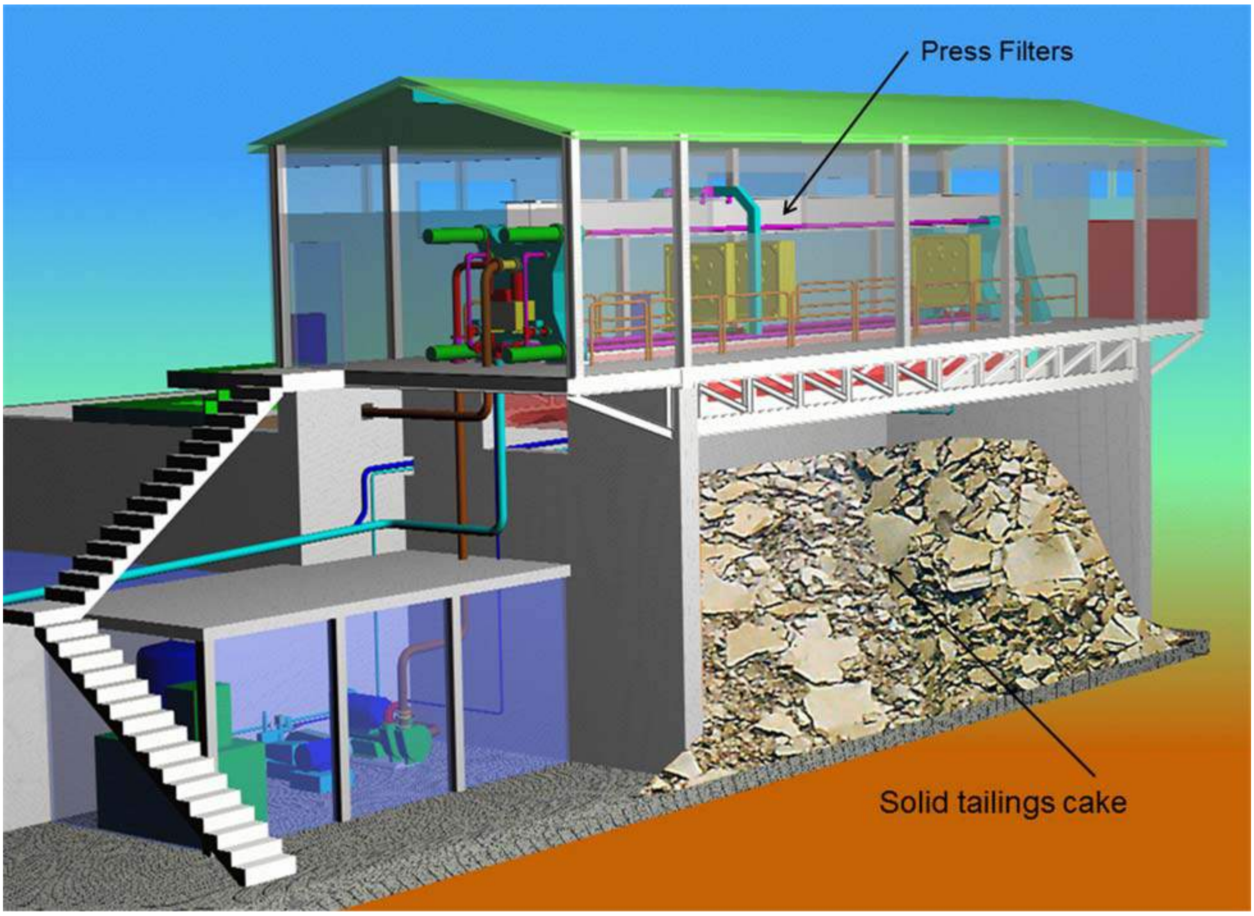

Fig. 10.4: Schematic layout of the solid tailing disposal system

10.5 Treatment of Chrome Concentrate:

Chrome concentrates is preferred by the customers only if it is hexavalent chromium free. The common remediation method (reducing toxic hexavalent chromium, Cr^{+6} to nontoxic trivalent chromium, Cr^{+3}) is by the use of $FeSO_4$, which affects the plant machinery. Hence there is a need to address this issue by an alternative environmental friendly, cost effective way and down the line metallurgical compatible process in terms of impurities. Unlike removing hexavalent chromium from water is a well-researched topic not much literature exists on removing hexavalent chromium from chrome ore. Hence, developing newer reductant especially with solids is a challenging task. Tata Steel in collaboration with CLRI has developed a process to reduce the hexavalent chromium to trace levels in concentrates by using an organic reductant.

The Organic reductant which is obtained by grinding a naturally occurring forests plant nut which contains around 40-60% tannin. For the first time, the Organic reductant is successfully used for reduction of hexavalent chromium in chrome concentrates to trace levels.

Lab scale studies were conducted to get the response of the Organic reductant powder in reduction of hexavalent chromium in chrome concentrates. For this, chrome concentrate were treated with Organic reductant at different dosages and different time.

Process at the plant

At the plant, solution is prepared in a SS tank ((1) of Fig-10.5) which is then pumped to another tank for storage ((2) of Fig-10.5). From the storage tank the organic solution is pumped to coarse classifier for mixing of solution with process water ((3) of Fig-10.5). In organic solution after mixing with the coarse concentrate make it hexa free. The above process has been patented by Tata Steel.

The flow diagram of Herbal Treatment Plant is as shown in Fig-10.5.

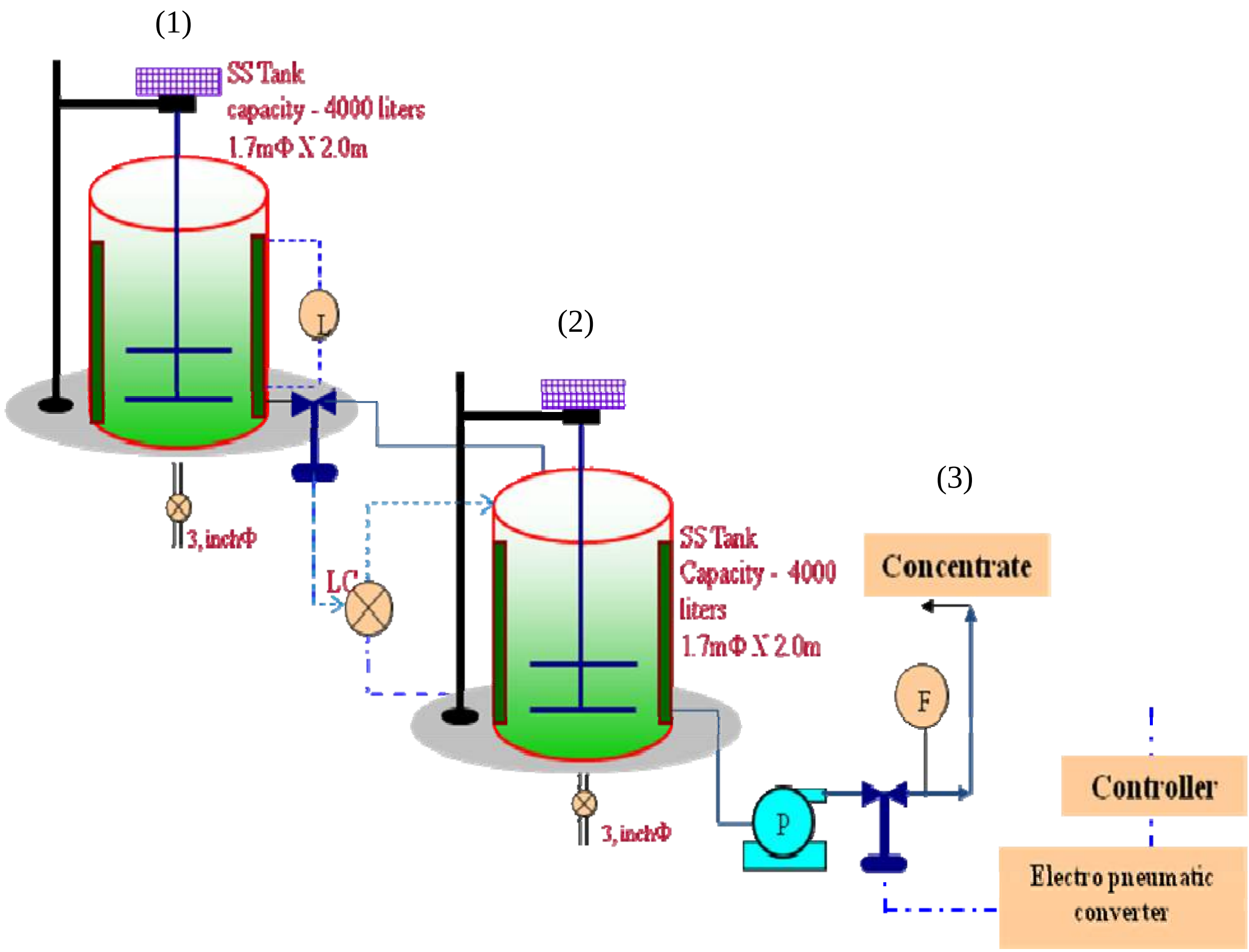

Fig-10.5: Flow diagram of Herbal Treatment Plant

PART- B

11. ENVIRONMENTAL MANAGEMENT PLAN

Sukinda Chromite Mine is a fully mechanized mine equipped for producing chrome ore, chrome concentrate and pyroxenite. Any developmental activity brings in some adverse environmental impacts, which are being dealt with scientifically for a sustainable pattern of mining. The salient environmental measures as presented in the mining plan are discussed herewith with proposal for next five years. The relevant data generated for air, water, and noise level are also enclosed. It may be observed that no negative impacts on water quality and air quality is envisaged since adequate protection measures have already been undertaken which will continued to be maintained. The mine is already certified to ISO-14001 for its better environment management practices and commitment to continual improvement of environmental performance & prevention of pollution. Besides company has plan to switch over to underground operations which is more environment friendly.

The company is having an Environmental Management Department with personnel from various backgrounds to take care of all environmental aspects. The department arranges to monitor all the environmental parameters and suggest to the management the necessary pollution control measures, as well as to execute all reclamation and afforestation programmes.

a) BASE LINE INFORMATION

Environmental study was conducted in an area covering 10km radius, with centre as the mining lease. The study area has been broadly divided into two imaginary zones –the Core zone & Buffer Zone. Core zone is the area where mining related activity concentrated. This constitutes the mining leasehold area of 406 ha, additional area of 100ha allotted for overburden dumping, contiguous to southern lease boundary and applied area of 73.685 ha for tailing disposal located further south east of the additional area of 100ha. The details as required under Rule No. 28(5) of MCDR, 1988 is shown in Key Plan (Drawing No. SCM/MP/01/12). The environment parameter monitoring data with respect to ambient air quality and water quality are enclosed as Annexure-9.

i). Existing Land use pattern:

The location and extent of lease area, the type of lease area (forest, non-forest etc) and the present land use pattern is given in the Table no. 11.1.1 and 11.1.2 respectively. The presently land use pattern is shown in the drawing no. SCM/MP/12/12. The activity wise breakup of the land is given below in Table 11.1.2

Table: 11.1.1

Area Type	Total Area (Ha.)	Forest Block No-27 Area (Ha)	Khesra Forest (Ha.)	DLC Area (Ha.)	Total Forest Area (Ha.)	Non-Forest Area (Ha.)	Remarks
ML Area	406.000	73.612	0.085	0.000	73.697	332.303	Mining Lease Area
Additional area	100.000	95.450	4.550	0.000	100.000	0.000	Allotted for dumping of Overburden
Applied Area	73.685	1.940	6.430	0.000	8.370	65.315	Applied for disposal of Tailing

Table 11.1.2

Present Land Use

Sl No	Activity	Present Land use (Ha.)
1	Area to be excavated	199.77
2	Storage of Top Soil	0.00
3	Over Burden Dump	37.47
4	Mineral Storage	37.08
5	Infrastructure (Workshop, magazine etc)	3.27
6	Roads (Present L= 7812m, W= 25m)	19.05
7	Railways	0.00
8	Greenbelt	38.53
9	Tailing Pond	15.95**
10	Effluent Treatment Plant	0.32
11	Mineral Separation Plant	18.38
12	Township Area*	36.18
13	Others (to be specified)	0.00
14	Area which will remain untouched	0.00
	Total	406.00

Remarks:

* - includes roads within the colony area

** - includes area of pyroxenite section for temporary storage of tailings

ii). Water regime :

The drainage system in the core as well as the buffer zones is well developed due to hill topography. Drainage pattern in the study area has been shown in drawing no.. SCM/MP/09/12. Damsala Nallah, which is perennial stream, forms the main drainage system in the study area. Damsala Nallah is flowing from the north-east towards the south through the central part of the study circle, at a distance of 0.8 km away from the leasehold. Patna Nallah and Nadibarna Nallah flowing from the north side hilly region join Damsala Nallah at eastern and centrals parts respectively. Damsala Canal runs from the central part towards north through a comparatively flat region (60-100 mRL) to serve the irrigation needs of the agricultural land. Apart from these, Paruajara Nallah is flowing from west to north near to northwest periphery of the study circle. Damsala Nallah and Paruajara nallah ultimately join Brahmani River flowing at a distance of about 15 km from Sukinda mine, beyond buffer zone study area.

iii). Flora and Fauna :

The vegetation of Sukinda Mining area and adjoining region belong to dry-deciduous type of vegetation. Similarly no such endangered species live in and around the mining leasehold area. The list of flora and fauna duly certified by DFO, Cuttack is enclosed as Annexure-30.

iv). Quality of Air, Ambient Noise level and Water :

Quality of Air:

During the operational stage the air quality monitoring is been done at different work zone locations like in the open pit faces, processing plants, waste tailing and reject dumps, transport roads, in mining colonies and adjacent villages for three dry seasons and during monsoon season, as per IBM/MoEF norms; to draw on climatic data coupled with emission data. The pollutants to be monitored are respirable particulate matter & suspended particulate matter (now PM_{10} & $PM_{2.5}$) containing significant silica/ silicates, toxic elements and other gases like CO, SOx & NOx.

To monitor the effectiveness of dust prevention and control actions, it is necessary to compare background levels of airborne dust with conditions down wind, and back this up with adequate meteorological equipments. At present air quality monitoring on core zone is carried out by six separate dust samplers complete with particle size partitioning and independent power generators provide sufficient data. The equipment selected is capable of collection of Sulphur dioxide and Nitrogen oxides samples. The method of deployment of respirable dust samples are down wind condition at the mine site and in nearby villages where there is chance of dust nuisance. Air quality analysis is carried out twice in a week all around the year.

The baseline status of the ambient air quality has been established through a scientifically designed ambient air quality monitoring network and was based on following considerations:

- Meteorological condition on synoptic scale.
- Topography of the study area
- Representatives of regional background air quality for obtaining baseline status
- Location of residential areas representing different activities
- In absence of any stack, the ambient air quality is expected to be affected in and around the mining areas up to a limited distance (within 5 km). Keeping this in view, air quality of core and nearby buffer zone mainly has been monitored.

To assess the ambient air quality level, 16 (Sixteen) monitoring stations are set up in the study area. Of these 16 stations, four stations are within the core zone of the mining area, one in sensitive area and one in colony area while the rest were outside the mining lease in buffer zone. Importance was given to measure existing air quality in core zone and neighboring buffer zone up to 10 km distance.

The results of Ambient Air Quality are enclosed as Annexure-9.

Ambient Noise Level:

In order to have an idea about the existing noise levels of the study area, noise monitoring has been carried out at various locations. Measurements have been carried out during the study at all the monitoring stations. At each noise monitoring station, Leq. Noise level has been recorded at hourly intervals for 24 hours continuously by operating the noise recording instrument for fifteen (15) minutes during each hour.

Quality of Water:

There is no perennial source of water like river, stream or nallah running through the lease area. Damasala nallah is flowing from east towards the south and is at a distance of 1.6 KM due north from the mining lease. Brahmani river is running at a distance about 15 KM from the lease area. The Sukinda valley experiences annually about 110 cm to 180 cm total rainfall. Eighty percent of total rainfall is during monsoon i.e. June to September. The major source of effluent generated within the lease area

- Mine discharge from dewatering
- Surface runoff during monsoon.
- Insignificant quantity of domestic effluent.
- Waste water generated during vehicle wash.
- Waste water generated from COB Plant.

There is an ETP with ferro sulphate dosing facility and horizontal roughing filters to treat the mine discharge and two ETPs located near old Geological camp and near Jagannath temple to treat the surface runoff. While the mine discharge water and surface runoff during monsoon is treated in effluent treatment plant, company has provided one modern sewage treatment plant to treat the domestic effluent of the colony. Similarly there are two oil and grease separation pit to treat the waste water generated during vehicle wash. The water so generated is recycled and reused again. The waste water generated during the beneficiation process is reused the plant itself. The water pumped along with the tailing is also recycled back from the tailing pond and reused in COB Plant.

The results of analysis of water at the lease boundary end are shown in Annexure- 9.

Climatic Condition:

Sukinda Chromite Mine enjoys four different seasons in a year. During summer from March to middle of June, the maximum temperature varies from 24 degree to 47 degree. During the rains i.e from mid-June to mid-October, south west rains cover the area with precipitation varying between 1200-1400mm. the area experiences cyclonic weather during monsoon period. The month wise rain fall for last five years is as given in Table:11.1.2.

Table:11.1.2
Month wise rainfall (mm)

Month/ Year	2010-11	2009-10	2008-09	2007-08	2006-07
Apr	64.77	0.00	3.60	32.40	0.90
May	134.62	66.04	16.20	37.40	188.70
June	104.14	86.86	288.33	278.20	148.80
July	242.57	421.25	276.74	204.40	198.20
Aug	196.18	366.09	265.89	263.30	382.00
Sept	171.18	200.77	410.40	416.10	411.60
Oct	97.88	119.38	8.89	56.10	12.40
Nov	11.43	10.16	8.89	6.30	4.00
Dec	48.26	0.00	0.00	1.10	0.00
Jan	0.00	2.54	0.00	17.70	4.70
Feb	2.84	0.00	0.00	43.60	9.70
Mar	24.58	19.05	3.81	52.80	18.90
TOTAL	1098.45	1292.14	1282.75	1409.40	1379.90

Autumn lasts from mid-November with pleasant weather having temperature between 18 to 33 degree. The winter starts from mid November and continues till end of February. The temperature drops to 8 degree during pick winter. Day time is quite warm but pleasant during winter.

The area is humid, during monsoon and the relative humidity goes up to 90%. However, the annual average relative humidity is 65%. The wind speed is generally light to moderate. High speed wind occurs in summer and monsoon seasons. Predominant wind directions in the study area for all seasons are SWS, SSW whereas in the post monsoon it is NNE.

v). Human Settlement :

The area has primarily forest and fallow land. Most of the smaller villages and hamlets consist of a single tribe or caste population. The major scheduled tribes are Munda, Khadia, Ganda, Oroan, etc. however the scheduled caste communities are Hadi, Pana, Doma, Mochi, Chamara etc. The two communities together constitute approximately 40% of the total population of the Tahsil.

The people are mainly forest dwellers with agriculture as their main activity. Other than this, a large number of people are employed as labourers in the various mining companies.Due to the poor soil fertility, the yield is extremely poor. Consequently, most of the people have to supplement agriculture with horticulture, foraging, hunting and other forms of occupation.

Trading as the main activity is practiced primarily by the non tribals, more so by some comparatively recent immigrants i.e people who have come along with the development of mining in these area. The little trade conducted by the tribals, is in the form of barter or small stalls to sell vegetables or forest produce.

Industrialization and the consequent urbanization have brought some changes in the settlement patterns as well as the social make up of the people of this area. A number of villages has increased in size, primarily those nearer to mining areas or on road sides. This increase in the population size of the villages was more due to the immigrants who came searching for the economic succor. This has changed the character of the villages from homogeneity of a single tribe or caste to heterogenisation.

b) ENVIRONMENTAL IMPACT ASSESMENT :

The environmental impacts due to mining and its allied activities largely depend upon the size of operation; stripping ratio and extent of mechanization. The matrix for impact identification is shown in Table 11.2.1.

The environmental attributes that may be affected are air quality, water quality and quantity, soil quality, noise level, flora and fauna (ecology), land-use, socio-economic environment and infrastructural development, health etc. Various activities causing impacts have been considered under various stages namely “Siting”. “Operational” (mine operation and secondary activities)” and “Mine closure”.

The activities have been arranged in columns and environmental attributes in rows in the matrix. A preliminary scrutiny has been done and the cells, which fall at the junction of “activity” and “attribute”, that have possible interaction with each other, have been crossed.

Thus the matrix identifies the environmental attributes likely to be affected and the activities responsible for this. The impacts may be beneficial or adverse. These will be analyzed during evaluation of the impacts.

Fig. 11.2.1: Impact Identification Matrix

Aspects	Siting	Operational stage															Mine Closure	
		Mine Operation											Secondary activities					
Environmental attributes	Site clearance	Blasting	Material loading & un loading	Material transport	Dump formation	HEMM operation	Mine water discharge	Workshop	DG set operation	Ore beneficiation	Employment	Town ship	Green belt development	Urbanisation	Infrastructure development	Industrialization	Afforestation	unemployment
Air Quality	x	x	x	x	x	x			x	x			•	x		x	•	
Water Quality					x		x	x					•	x		x	•	
Water resources							x			x		x	•	x	x	x	•	
Noise	x	x	x	x		x		x	x				•	x	x	x	•	
Soil quality	x				x								•				•	
Flora & Fauna	x				x							•	•	x		x	•	
Land use	x				x							•	•	•	@		•	
Health & Safety		x						x				•		•	•	x		
Recreation											•	•		•	•	•		
Socio-economic environment											•		•	•	•			x
Aesthetics	x				x							•	•	@			•	

X Adverse impact
• Beneficial impact
@ May be both beneficial and adverse impacts

The main operation at Sukinda Chromite Mine involves drilling, blasting, overburden, ore loading, overburden & ore transportation and ore beneficiation both in opencast as well as underground mining.

The identified impacts due to mining and allied activities related to certain areas are:

- Topography and landscape
- Drainage System
- Climate
- Land use Pattern
- Water Regime
- Noise Level
- Blast induced vibration, noise fly rock
- Flora & Fauna
- Soil & Agriculture

The impact on all these areas due to the projected opencast and underground excavation for coming years, have been assessed thus:

i). Impact on Topography & Landscape:

The lease area will undergo topographical change due to digging, widening of quarries and extension of overburden dumps. The topographical changes due to opencast mining during the planned period up to 2018 and at end of life period till 2033 have been shown in drawing no. SCM/MP/13A/12 & SCM/MP/05/12. However the change is confined only to the core zone which contributes only 2% approximately to the total study area. There would no topographical change envisaged due to proposed underground mining.

ii). Impact on Drainage System:

In buffer zone, no change in the drainage system is anticipated due to mining activity within the leasehold. In core zone there will be no impact on the existing drainage system.

iii). Impact on Climate:

No effect on the local climate is anticipated. Rather, some improvement is local climate is expected after implementation or massive plantation scheme in operation. No significant impact on overall climate of this region, due to mining activities, is noted or is expected as mining activities are limited to a very small area compared to the study area.

iv). Impact on Land use:

The impact on land use in buffer zone due to mining activities within the leasehold is insignificant. The agricultural land use pattern is not expected to be affected considering the agro-climatic condition of the area

The present land use patterns of the entire lease hold area of 406 ha. is as given in the para 1.d. Presently around 4.03 ha of mined out land in the extreme western part of OB II Quarry has been back filled since the ore body is not existent at greater depth. As mentioned in conceptual plan, back filling of quarries and its subsequent rehabilitation by way of plantation shall be taken up after the economic pit bottom depths are reached along the entire strike of the deposit. The conceptual Mining Plan showing the areas that shall be so taken up for back filling have been furnished as shown in Drawing Nos. SCM/MP/05/12. The land use pattern as envisaged till 2017-18 and end of the life of mine over the entire lease hold area of 406 ha. is given in Table 11.2.2

Table. 11.2.2

Sl.No	Type of Land Use	As at the beginning plann period (12.01.2013) (in ha.)	As at the end of Plan period (2017-18) (in ha.)
1	Area to be excavated	201.50***	175.49
2	Storage of Top Soil	0.00	0.00
3	Overburden Dump	37.47	76.11
4	Mineral Storage	30.70	32.86
5	Infrastructure (Workshop, Magazine etc	7.92	7.92
6	Roads (Present L=7812m, W-25m)	19.05	19.05
7	Railways	0.00	0.00
8	Greenbelt	38.53	29.69
9	Tailing Pond	15.95**	10.00
10	Effluent Treatment Plant	0.32	0.32
11	Mineral Separation Plant	18.38	18.38
12	Township area *	36.18	36.18
13	Others (to be specified)	0.00	0.00
14	Area which will remain untouched	0.00	0.00
Total		**406.00**	**406.00**

* The township area includes the roads with in the township

** The excavated area includes the pyroxenite section

v). Impact on Water Regime:

In the core zone i.e. within 406 ha existing leasehold area, 100ha of allotted area for overburden dumping and 73.685 ha of applied area for tailing disposal, the streams flowing down from lease area etc. shall not be affected in the coming year.

In buffer zone there has been no change in the drainage system due to this project and no further change is anticipated.

vi). Impact on Water Quality:

Contamination of Surface Water in the Core zone could occur due to:

- Discharge of water from mine, COB Plant, workshop and garage.
- Rain water overflowing from stacks.

Water percolation in different quarries varies from 150 gpm to 250 gpm in the post monsoon season. Percolation rate increases considerably in monsoon season. In future years higher water accumulation is anticipated with increase in depth of excavation. similar to the present water utilization plan, in future also mine discharge water will be utilized in various mining and industrial activities, e.g. haul road spraying, HEMM washing, plantation, COB plant and the remaining water will be discharged to fallow land after treatment.

Water analysis results show that in general the concentration of hexavalent chromium in the mine discharge water is above the limits. In order to mitigate the harmful effects due to hexavalent chromium in effluent water, company had engaged M/s National Environment Engineering & Research Institute (NEERI), Nagpur for conducting feasibility studies on removal of hexavalent chromium. Based on the study report three effluent treatment plants have been. One ETP is exclusively installed to treat the mine discharge water and rest two treat the surface runoff during monsoon. Analysis of the treated water shows that hexavalent chromium concentration is much below the permissible limit.

Discharge water from workshop and water used for cleaning HEMM is handled through oil and grease trap and arrangement for suspended solids collection. The water is recycled back for vehicle wash. From the COB plant there is no discharge of water outside the leasehold area as the plant is provided with re-circulation facilities. The tailings are disposed off to a tailing pond. The water from the tailing pond is recycled back and reused in the COB plant.

Rainwater, while overflowing from OB stacks may get mixed with fine rock particles. These rock particles, e.g. Serpentinite, Quartzite, Nickeliferrous limonite etc. contain chromite in trivalent form which is not soluble in water. Water loaded with these particles are thus, not toxic in nature. However, settling pits have been made for treatment of suspended solids before leading this water to the natural drainage system. The water along with dump wash off is also treated at the ETPs located at old Geological camp and near Jagannath temple.

vii). Impact on Air Quality:

The predominant wind directions are ENE in winter season. Average wind speed is 2.53 kmph and readings for air pollutants in the immediate buffer zone are almost same as distant zone readings, besides being much less than the CPCB standards. Air pollution level due to existing mining activities is very low and confined to a limited area, in and around the quarries within the core zone only. PM_{10}, $PM_{2.5}$, SO_2, NOx, CO, NH_3, Pb and HC levels in work zone atmosphere also are well within norms. Mining and transport machinery generate smoke and fumes, which consist of SO_2, NOx, CO, Pb and HC causing air pollution. Much of the fuel emissions are checked due to high standard of maintenance and regular emission level monitoring. Blasting is carried out by using controlled blasting techniques to minimize generation of various gases. The same methods will be followed in the coming years also.

Mining operations such as drilling, blasting, movement of dumpers, dumping of associated mineral, excavation and loading/unloading etc. generate air borne dust. Generation of air borne dust due to excavations is not very high as the rocks are hard and have high specific gravity. Water sprinkling on haul roads, water spraying before and after blasting, wet drilling, is a regular routine phenomenon.

Dust problem in COB Plant is insignificant as wet process is practiced.

Effect on air environment due to proposed activity is mainly due to fugitive dust emissions. The excavation and loading & unloading activities are isolated activities. Contribution due to these sources is minimum. Automatic stationary dust suppression system has been established in the main haul road for dust suppression in addition to mobile water sprinklers.

Further to above as the project is witching over from opencast to underground mining operation during the conceptual period the ambient air pollution is envisaged to be minimum in future.

viii). Impact on Flora & Fauna:

In a mining project, all activities from acquiring of land up to reclamation and post mining cause impact on vegetation and fauna directly or indirectly and of varying duration (i.e. short term or long term). Removal of natural vegetation affects eco-system causing reduction of native plants and animals, disturbing habitat, breaking food chain etc. In the core zone, mining had started in 1960. Vegetation lost since then in meager as in the pre-mining period tree growth was scarce and the vegetation was mostly shrubs and bushes. However, some impact in the coming years in anticipated as 100 ha. more forest land is under utilisation for other overburden dumping for next three to four years.

In order to restore the green cover a massive afforestation scheme has been undertaken. Company has already initiated the afforestation work in a planned manner. The details of plantation are given below in Table No. 11.2.3

Table: 11.2.3

Year wise Plantation details within Lease Area (406ha)

Year	Area of Plantation (ha)	Nos. of sapling planted	Plant Density (nos./ ha)
1998-99	1.70	4000	2353
1999-2K	4.00	18000	4500
2000-01	2.00	28342	14171
2001-02	0.50	15000	30000
2002-03	1.50	22000	14667
2003-04	1.50	45500	30333
2004-05	1.00	48000	48000
2005-06	2.50	75000	30000
2006-07	5.75	129500	22522
2007-08	4.42	94000	21267
2008-09	2.94	85250	28997
2009-10	3.90	28000	7179
2010-11	2.00	25000	12500
Total	33.71	617592	18321

ix). Socio-economic impact:

With the increase of mining the following socio economic changes are expected to take place especially within 4- 5 km from the leasehold area.

- The mining operation is not going to cause any damage to existing agricultural situation which is scanty, characteristically low yield and seasonal in this area. Rather the mine operation is likely help to improve agriculture by way of providing supplementary income to the farmers which is likely to increase investment in agriculture and consequently agricultural production.
- The mine management is going to have positive employment and income effects, on account of indirect employment associated with allied activities.
- The mine operation is going to have positive impact on consumption behavior by way of raising average consumption and income through multiplier effect.
- The mine management is likely to speed up the growing view on importance of education among the people of the area.
- The mine management is going to push up the demand for water for drinking and other purposes in the region.

Health facilities have improved several folds since the mining projects have come up. Company has a full-fledged 32 bedded hospital at Sukinda for employees and local people. They also operate family welfare services, health camps on specific diseases, eye operation camps, etc. Life line express (Hospital on Wheels) has also helped the community people to treat many of the costly and incurable diseases. These amenities will be continued in the years to come to continuously upgrade the health status of this area.

Increasing literacy and education is one of the important parameters of social development. Concern for education is increasing due to setting up of extensive educational facilities, a major part of which has been shared by the company.

Due to coming up of the mining projects, communication facilities have improved several fold. Tata net, E-mail, LAN, Internet etc are now available. The road communication has also improved. Company has now renovated existing Tomka- Managlpur road from Mangalpur to Mines at the cost of approximately Rs 11 crore.

Tata Steel Rural Development Society (TSRDS) is playing a very important role in uplifting the socio-economic status of the inhabitants. The activities taken by TSRDS have been discussed below:

i) TSRDS encourages various village clubs e.g. Jubak Sangha, Mahila Samiti etc. to raise sapling in nursery, which are bought by Tata Steel.

ii) TSRDS aids local people for making leaf plates and also brass materials, which are later brought back by company.

iii) TSRDS assists in agricultural sector also. Crops like paddy, wheat, potato, onion etc. are grown by families in nearby villages of Benagadia, Kendubari, Baragazi, Bambilo, Kaduabandi etc.

iv) TSRDS has arranged for lift irrigation by providing pump sets to the farmers of nearby villages.

v) TSRDS has formed a weaver’s society of village ladies in Pimpudia village.

vi) TSRDS constantly involves itself in digging tube wells, open wells, ponds and also in minor irrigation dams.

vii) TSRDS provides regular medical services to the surrounding villages. Medical services include one mobile hospital with doctor and nursing staff who visit the villages every day. TSRDS arranges one ambulance especially for emergency patients. Apart from these, special programmes are arranged from time to time, These area:

- Malaria eradication programme
- Medical checkup of school children
- Eradication of night blindness
- Health worker's training

viii) TSRDS organizes cultural activities such as Pala, Das kathia competition and also village sports and football tournaments etc.

ix) TSRDS organizes 'Literacy mission programme' through various clubs and societies.

x) Massive plantation work at a number of villages has been undertaken by TSRDS.

c) ENVIRONMENTAL MANAGEMENT PLAN :

Sukinda Chromite Mine of Tata Steel Limited is the oldest chromite mine operating in the valley. This mine was started way back in 1960, much before the enactment of laws for environmental protection.

In order to depict the present environmental scenario, various environmental factors were classified and analyzed by the company's in house facility as well as by consultants., which were engaged by the company for monitor these factors. The company shall continue to monitor all the relevant environmental parameters. The monitoring stations with respect to air quality monitoring, noise level and water quality monitoring have been indicated in drawing no. SCM/ MP/09/12.

To maintain ecological balance and to check the harmful effects due to mining and allied operations at Sukinda Chromite Mines, environmental control measures have been integrated into the process of mine planning.

Many of the areas in environmental management planning are of multidisciplinary dimension. Therefore, the measures envisaged in the report are to be regarded as a guide and depending upon the continuing advice to be taken from experts of relevant fields like forestry, soil chemistry, ground water etc., the suggested schemes are to be detailed and, if necessary, modified. The changes warranted as per specific site conditions are to be accounted for, during actual implementation phase. In this chapter all technical, biological and socio-economic control measures have been envisaged and these pertain to:

- Solid Waste management
- Land use Planning
- Afforestation and landscape development
- Air Pollution Control
- Hydrography and Water Pollution Control

- Noise Control
- Measures against ground vibration, fly rock and air blast
- Occupational health and safety
- Socio-economic measures

i. Solid Waste Management:

Sukinda Chromite Mine has been in operation since last five decades. The area already exposed for mining is devoid of top soil as there is insignificant lateral expansion of quarries. However whatever excavated top soil shall be stored separately before the commencement of mining activities in the proposed area and staked separately at the earmarked area in 1m height top soil dump. Top soil will be preserved by planting grass on slopes and top of the dump. All the overburden waste dumps will have a maximum slope of 28^0. At the end of mining all the inactive dumps will be biologically stabilized with multi species afforestation. The year wise generation of overburden for planned period of five year is as given in Table:11.3.1

Table:11.3.1

Year wise overburden generation (Lakh Cum)

YEAR	Overburden Generation (Cum)	To be disposed at Kakudia Dump (Cum)	To be Backfilled in OB-II Quarry (Cum)	Remarks
2012-13*	1490000	1490000	0	Concurrent reclamation & rehabilitation measures to be taken up as soon as reaches its capacity
2013-14	6275149	6275149	0	
2014-15	6843270		6843270	
2015-16	6727634		6727634	
2016-17	6789183		6789183	
2017-18	6871362		6871362	
TOTAL	34996598	7765149	27231449	

The excavated quarries shall be progressively backfilled and plantation shall be carried out on backfilled area and also over the dumps. A landscape with predominance of vegetated land shall emerge. The external dump also shall be terraced and afforestated.

However, parapet walls will be constructed for some of the active dumps during the mining. Year wise construction of retaining wall and area (plan area) to be afforestated for the next five year planned period is given below in Table 11.3.2, 11.3.3 & 11.3.4 respectively and is shown in drawing no. SCM/MP/10/12.

Table 11.3.2

Construction of Retaining Wall

Year	Additional Area of 100 ha.	Applied Area of 73.685ha.	Total	Cost	Width (m)	Height (m)
	Length (m)	Length (m)	Length (m)	(Rs Lakh)		
2011-12	1000	0	1000	15	1.00	1.00
2014-15	0	3500	3500	53		
Total	1000	3500	4500	68		

The year-wise proposal for plantation and expenditure to be incurred in the additional area of 100 Ha, with in Mining Leasehold area of 406ha. and in applied area of 73.685 ha along the slopes of associated mineral stacks and in vacant land is as shown in drawing no. SCM/MP/10/12 and is given below in Table 11.3.3

Table 11.3.3
Year wise proposed Afforestation and Expenditure

Year	Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total Plantation		
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)
2013-14	2.5	6250	8.00	20000	3.00	7500	13.50	33750	27.0
2014-15	1.6	4000	7.95	19875	3.15	7875	12.70	31750	25.4
2015-16	3.0	7500	8.05	20125	1.25	3125	12.30	30750	24.6
2016-17	3.0	7500	7.85	19625	1.05	2625	11.90	29750	23.8
2017-18	3.4	8500	8.05	20125			11.45	28625	22.9
TOTAL	13.5	33750	39.9	99750	8.45	21125	61.85	154625	123.7

The coir matting shall be done in the Ni- limonite overburden dump of OB-X quarry where the chances of wash off is more but may not be possible in overburden dump of OB-II quarry due to its very hard nature and sharp edges of overburden material.

ii. Air Pollution Control:

The existing level of air pollution in the project area is, in general, low. Due to no increase in mining activities in future, the level of air pollutants are expected to remain same. Further to above after underground mining the air pollution is expected to go further low. The following measures most of which are practiced at present, shall be continued to avoid air pollution.

Fugitive Dust

Fugitive dust is generated in drilling, blasting, shoveling (loading) and unloading and in haul roads due to movement of HEMMs. Fugitive dust is also generated in COB Plant.
Control of dust to Mining operations

While complete elimination of dust due to mining operation is not possible, control measures as indicated below shall be continued to reduce dust nuisance.

Drilling

Wet drilling is practiced i.e. a jet of water is continuously directed at the cutting edge. Correct water pressure is maintained. The drill operators are provided with dust respirators.

Blasting

Generation of dust as a result of blasting cannot be avoided. However, its spread is somewhat checked by the use of various controlled blasting techniques like presplit blasting, use of NONEL and SME (Site Mixed Emulsion) and avoiding over charging of blast holes.

Loading and unloading

By water sprinkling during loading and unloading of associated mineral, dust is prevented from become air borne, to some extent. Dust proof air conditioned cabins have been provided in the shovels and dumpers.

Storing of associated mineral

The volume of dust raised from associated mineral stack areas, by action of wind will significantly be reduced by planting of grasses and creeper species in the associated mineral stack soon after formation. Dust proof air conditioned cabins have been provided in the haulpak dumpers.

Haul Road

Haul roads are the major sources of dust in a mine. To overcome the problem of dust generation in mine haul roads it is suggested to:

- Construct metal roads, which are permanent in nature like road to workshop and garage, colony etc.
- Automatic stationary dust suppression system installed at the main haul road will prevent the air borne dust due to ore transportation through trucks.
- Leaves of trees along these roads (avenue plantation) will act as a sink of dust.
- Water other (temporary) haul roads. These haul roads will be made compact and water spraying will be done in a number of runs by mobile water sprinklers using only a small quantity of water each time just sufficient to wet surfaces.
- Ensure transport equipment to be leak proof.
- Prevent over loading of the transport equipment in order to stop spillage.

Ore Stack Yard

Water sprinkling will be done round the clock in ore stack yard to bring down the airborne dust. Peripheral plantation around the stack yard will screen dust.

Control of dust generated in COB Plant

Since COB Plant operation involves wet process, dust generation level will be very low. However, pressurize water jets have been provided in the crusher plant to suppress air born dust.

Control of pollution due to fumes from mining machinery

Exhaust fumes in the internal combustion engines used in excavators, dampers, dozers and other machinery can be minimized by ensuring vigorous maintenance and stringent overhaul schedules. The repair workshop and maintenance garage equipped with all necessary facilities will ensure upkeep and maintenance of engines. Regular emission checking will continue to be carried out as per the rule to maintain good health of the engines.

iii. Water Pollution Control:

Prevention of surface water contamination

After supplying makeup water to COB plant and also for various mining, dust suppression and plantation work, the remaining quantity of mine discharge will be pumped to the effluent treatment plants. The quarry water, in general, contains Cr+6, which is a toxic. Cr+6 Treatment units have been set up to treat this mine water before discharge. Hill water running through the associated mineral stacks and quarries (in garland drains) are expected to contain high amounts of suspended solids, which will be settled in the setting tanks followed by treatment at ETP located at old Geological camp and Jagannath temple. Water management scheme suggested for this project is as follows:

- Construction garland drains of 1.5 m X 1 m depth (minimum) in cross section) around quarry, associated mineral stacks and other infrastructures viz. Workshop, Garage etc.
- Settling pits at regular interval to arrest the wash off..
- Provision of small stone barriers across the drains at intervals to check the water current and to arrest solid particles.
- Stone pitching and concreting will be made at suitable locations on the drain sides to regulate the water which otherwise may flow here and there.
- Drains to be cleaned periodically.
- Water is treated in ETP for removal of Cr^{+6} , if any.

Treatment procedure

Hexavalent chromium is a strong oxidizing agent Cr+6 and can readily be reduced to Cr+3 in presence of a reductant like ferrous sulphate. Company had engaged National Environmental Engineering Research Institute (NERRI), Nagpur to suggest a suitable treatment procedure for Cr+6 in mine water of Sukinda Mine. It has been proposed by NEERI that from among the other reductants like Sodium bisulphate (NaHSO4), Sodium metabisulphate (Na2S2O5) and sodium dithionate (Na2S2O4), FeSO4 is ideally suited for the removal of Cr+6 from Sukinda Mine water. A dose of 40 mg/l commercial ferrous sulphate (FeSO4), 7H2O) is used for removing 1.0 mg/l of Cr+6. During the process ferrous sulphate is oxidized and Cr+6 is reduced to Cr+3 and is precipitated out of the medium. The treatment units essentially consist of solution tanks where FeSO4 solution is prepared and stored. The consistency of the solution is being maintained by intermittent stirring with the help of electrical stirrer mounted on the tanks. The solution so prepared then added to the contaminated water and is done automatically with the help of a mechanically operated doser. The water is stirred in chamber to ensure proper treatment, although the reaction is instantaneous, and is passed through a bank of horizontal roughing filters for removal of suspended solids. Lime dosing arrangement is provided at the end to ensure correct pH, in case it is not. The horizontal roughing filters are cleaned/ desludged from time to time with the help of sludge drain valves provided for the purpose. Details pertaining to the size of treatment tanks etc. have been given Table 11.3.5

Table: 11.3.5
Design details of Treatment System Components

Design Flow (cum/hr)	Solution Tank		Horizontal Roughing filter			
	No.	Cap. (cum)	No.	Length (m)	Breadth (m)	Depth(m)
715	2	5.0 each	4	25	10.25	1.75

However efforts are going on to augment the existing ETP to an automated one having higher capacity of 3000-4000 Kl/ hr to treat mine discharge water of the underground operation as well. In the meantime IIT, Khargpur was engaged by State Pollution Control Board, Odisha to study the feasibility of common effluent treatment plant at Sukinda Valley. In their final recommendation they have proposed a conceptual design to be followed to renovate the existing ETP. The conceptual design of the ETP is as given in Figure 11.1.

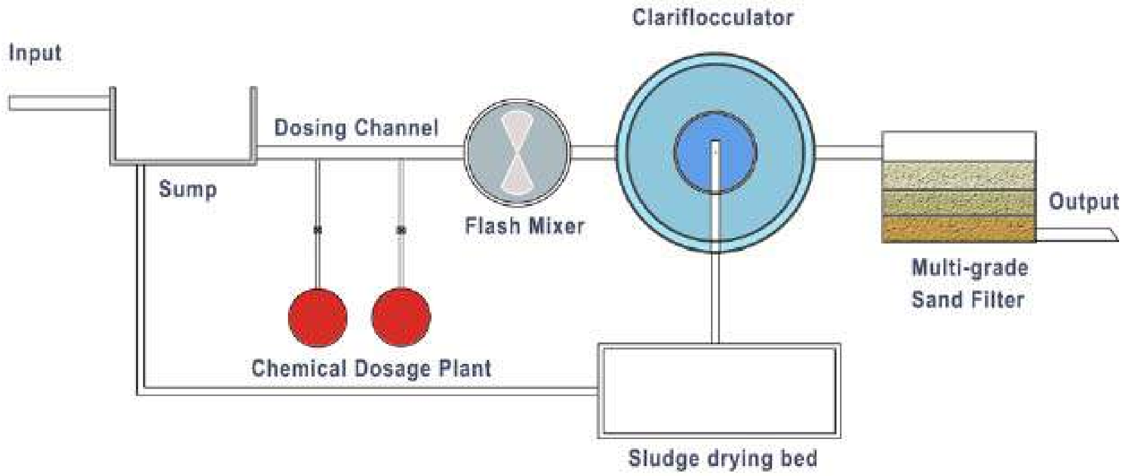

Figure 11.1: Conceptual Design of proposed ETP

Surface runoff

The surface runoff from garland drains is led to another two effluent treatment plants located at old Geological camp and Jagannath temple where dosing of ferrous sulphate solution is done to treat hexavalent chromium, if any. The consistency of the solution is maintained by electrical stirrers. The water passes through a series of settling pits constructed before and after effluent treatment plants for settling of suspended solids. The settling chamber provided after ferrous sulphate has baffle walls to allow resident time to ensure treatment. The capacity of the plants is to treat 3000 Kl/ hr of water. This is now in operation during rainy season only after commission of above mentioned full-fledged effluent treatment plant to take care of mine discharge. The sludge is removed from the settling pits both by mechanically/ manually depending on the approachability of the area. Above two ETPs shall also be renovated in future as per above conceptual design.

The sludge generated from all the above systems are put in sludge impervious drying beds constructed at the Geological ETP and the dried sludge is disposed as per Hazardous Waste (Management, Handling and Transboundary) Rules, 2008.

Effluent water from COB Plant and Workshop, Garage etc.

The clear water from tailing pond is re-circulated to the COB plant and hence there will be no waste water discharge from COB Plant to the surrounding garland drain and natural drainage system.

Workshop and Garage wash water is taken to oil & grease separation system. The plant has the provision of screens to arrest the floating physical materials like dry leaves etc. The baffle wall of the plant then arrests the clods. The overflow water passes through oil and grease separation system gravitationally wherein the oil & grease is collected in a catch pit and clear water goes out to the final chamber. The treated water is recycled back leading to no discharge out of the lease. The plant is covered with shed. The oil and grease soaked material collected from different maintenance and operation activities are stored in a pit constructed nearby. The oily Sludge and the oil soaked jute and filter material is disposed in CHWTSDF as per Hazardous Waste (Management, Handling and Transboundary) Rules, 2008.

Effluent from colony

Sewage from the colony quarters is collected in different septic tanks in different places and is biologically treated in the septic tanks and soak pits under anaerobic conditions. In addition a sewage treatment plant with facilities of grit chamber, aeration network, setting tank, pressure filter, activated carbon filter, chlorination, desludging with sludge drying bed has been provided. The STP is satisfactorily in operation to meet all the water quality norms prescribed by the Board. The treated water is used for plantation.

Drinking water arrangement

As described earlier the principal source of drinking water in the colony is Damsala Nallah. Same water supply will be followed in future also. Present drinking water analysis does not show any contamination. Though company discharges the treated mine discharge water, following the same procedure at present there are other sources of inflow to the water body. As such, it is suggested that regular drinking water analysis should be carried out and necessary a dose of ferrous sulphate (depending on the periodic analysis results) have to be applied in the treatment system. In the drinking water treatment plant provision of alum dosing, lime dosing, settling, filtering through gravity as well as pressure filters, chlorination are existing to ensure supply of wholesome drinking water to the users. The overhead tanks over the quarters are cleaned at least twice in a year to ensure hygienic condition.

iv. Control Impact on Water Regime:

As stated earlier, there is no perennial source of water crossing the mining lease area. The closest water body is Damsala nalla, which flows 0.8km away from the lease boundary and 1.6-2 Km from the quarry. As per the Central Ground Water Board classifications, the area falls under green (safe) zone. The hydrogeological modelling for underground mining also suggests there would be no such significant impact on water regime due to underground mining. However company would take adequate steps for roof top rain water harvesting as well try to recycle and reuse the mining water to maximum possible extent.

v. Noise Pollution Control:

Noise level in the work zone (near blast hole drilling spots) is slightly above 90 dB (A) (for 8 hr. exposures). Noise level in the COB Plant control room (manned 24 hours) is well within the limit. As general precautions against noise pollution to reduce the effect of high noise level, the following ameliorating measures are proposed to be taken.

- Continuing the provision of ear protective devices like acoustic wool, earplugs, and ear muffs for workers exposed to noise level of more than 90 dB(A).
- Continuing the provision of sound proof air conditioned cabins for the workers deployed at machines like shovels, haulpak dumpers, dozer, drills and graders etc.
- Proper maintenance of noise generating machinery including transport vehicles would be ensured.
- A 5 meter thick tree belt shall developed around the periphery of the mine, COB Plant other industrial units and the colony to screen the noise.
- Reducing the exposure time of the workers, wherever required.
- Acoustic enclose are been provided for DG sets.

Even through noise level arising due to blasting operating is for very short period, Controlled blasting techniques along with general precautions have to be adopted as given under blasting hazards.

In order to control the impact of noise due to operation of the machineries in COB Plant, it is necessary to optimize the impact distance and to cover, either or both the impact surfaces by a resilient material like rubber, vibration springs, pads etc, whenever practicable. Toward this end the chutes of the crusher have been rubber lined, the iron screen panels have been replaced with polyurethane/ rubber screens and stricter maintenance practices etc. have been adhered to.

vi. Control of vibration, noise and fly rock generation due to blasting :

Measures to be taken while blasting in both opencast and underground mines are as follows:

- Blasting shall be carried out only during day time as per schedule.
- Blasting will not be done during stormy weather and lightening.
- Maximum permissible charge per delay would depend on the distance of structure to be protected from the blasting
- Detonation cords, cord relays, milli-second delay detonators will continue to be used for connection and initiation of holes.
- Suggested number of rows in a blast should not be more than four to reduce fly rock and ground vibrations.
- Length of blast will depend on the total number of holes which can be blasted in a round while observing the above guidelines/ restrictions. However, it should be as large as design would permit. Attempts should be made to have one large blast with less frequency than to have several small blasts.
- Drill cuttings may be used as stemming material.
- Muffling of holes including the area to be blasted must be done
- Multi delay in a hole by using nonel is in practice to control vibration, noise and fly rock.

- Further to above various controlled besting techniques like pre-split blasting, use of NONEL and SME (Site Mixed Emulsion) shall continue to be practiced.

vii. Afforestation :

In the Sukinda Chromite Mine, green belt will also be developed on external over-burden dump, vacant areas, around office buildings, garage, stores etc. and along the side of roads.

The green belt will serve the following purposes:

- Compensate the loss of vegetation due to setting up of the mine
- Prevent the spread of fugitive dust generated due to mining and allied activities
- Attenuate noise generated by the mine.
- Reduce soil erosion
- Help to stabilize the slope of external soil and over-burden dumps
- Increase green cover and improve aesthetics.
- Once the mine is abandoned the green belt will accelerate recolonisation of the area by animals scared away by the project.

Afforestation on the slope:

The stabilization and rehabilitation activities shall commence soon after the stacks reach the boundaries of the allotted areas. The stacks of associated minerals shall be developed in the form of benches of 10- 30 m in height. After completion of one bench, a berm of 10-15 m width will be left from the sides and then second lift will be made in form of another bench and so on. Afforestation on the slopes will commence as soon as the first bench reaches the extremities of the lease area. Terraces on the slopes will be slope inward and 05m to 0.75m deep pits will be dug at an interval of 1m X 1.5m. Saplings will be planted in the pits with a mixture of topsoil and manure/fertilizer at the beginning of monsoon months. This will bind the associated mineral stack slopes. After completion of one bench a passage of about 10-15m width will be left out from the sides and second lift will be taken in the form of another bench and similar biological reclamation will be done.

Post Plantation Care:

Immediately after planting the saplings, watering will be done. Further watering will depend on the rainfall. In the dry seasons watering will be regularly done especially during April to June. Water of younger saplings will be more frequent. Manuring will be done using organic manure (animal dung, agricultural waste, kitchen waste etc.). Younger saplings will be surrounded with tree guards. Diseased and dead plants will be uprooted and destroyed and replaced by fresh saplings. Growth / health and survival rate of saplings will be regularly monitored and remedial actions will be undertaken as required.

The year-wise proposal for plantation and expenditure to be incurred in the additional area of 100 Ha, with in Mining Leasehold area of 406ha. and in applied area of 73.685 ha along the slopes of associated mineral stacks and in vacant land is as shown in drawing no. SCM/MP/10/12 and is given below in Table 11.6.6

Table 11.6.6
Year wise proposed Afforestation and Expenditure

Year	Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total Plantation		
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)
2013-14	2.5	6250	8.00	20000	3.00	7500	13.50	33750	27.0
2014-15	1.6	4000	7.95	19875	3.15	7875	12.70	31750	25.4
2015-16	3.0	7500	8.05	20125	1.25	3125	12.30	30750	24.6
2016-17	3.0	7500	7.85	19625	1.05	2625	11.90	29750	23.8
2017-18	3.4	8500	8.05	20125	0	0	11.45	28625	22.9
TOTAL	13.5	33750	39.9	99750	8.45	21125	61.85	154625	123.7

The coir matting shall be done in the Ni- limonite overburden dump of OB-X quarry where the chances of wash off is more but may not be possible in overburden dump of OB-II quarry due to its very hard nature and sharp edges of overburden material.

viii. Land Use Planning :

There is no vacant space available with the mining leasehold area for overburden storage. The overburden generated during the planned period shall continue to be stored in additional allotted area of 100ha. After these areas are exhausted the dumps shall be stabilized and rehabilitated in a time bound manner. However after the underground mining achieves its rated capacity the overburden generation shall be minimum.

Although associated mineral stack height will be high in the proposed stacks but forests will be grown over it, and the afforestated associated mineral stacks will match well with the existing hilly forests, in vicinity. The terraced formation of these stacks will form better landscaping and hence the aesthetics.

Apart from the associated mineral stacks, there will be other areas like peripheral plantation zone around the quarries, industrial units, and avenue plantation within township and also selected areas, which will be put under vegetative cover.

The quarry voids will not remain as land scars. Rather, they will also aid in better landscape formation by transforming naturally into lagoons, as described above.

Necessary socio-economic schemes and development activities will continue to be taken up through Tata Steel Rural Development Society (TSRDS) so as to ensure that the present land use pattern is not disturbed in the buffer zone.

ix. Occupational Health and Safety:

Occupational health and safety is very closely related to productivity and good employer-employee relationship. The mine has developed and certified to OHSAS-18001 (Occupational Health and Safety Assessment Series), an international specification. In the process, each and every activity has been evaluated for its implications on OHS related risks, suitable objectives and targets are set and implementation in progress to reduce the concerned high risk prone areas.

Documented procedures exist at all relevant levels and functions to ensure proper OHS management system. The main factors of occupational health in open cast mines are dust and noise. Safety of employee during blasting operation and maintenance of mining equipment and handling of explosive materials are being taken care of as per mines rules and regulations. To avoid any adverse effects on the health of workers due to dust, noise and vibration etc. sufficient measures have already been addressed in this chapter. Some other measures mainly relating to safety and health include.

- Identification and assessment of the risk of health hazards at work places.
- Training of employees for use of safety appliances and first aid.
- Extensive publicity and propaganda related to safety.
- All safety measures e.g. use of safety appliances, safety training, safety awards, posters, slogans related to safety etc.
- Organization of First Aid in mines including training and re-training of First Aiders.
- Provision of rest shelters for mine workers with amenities for drinking water, fans etc.
- Provision of high capacity pumps for pumping out water from mining pits.
- Working of mine as per approved mining and environmental management plans.
- Regular maintenance and testing of all mining equipment as per manufacturers guidelines.
- Training of workers on sanitation, cleanliness, hygiene and health care.
- Periodical medical examination of all workers by medical specialist (are being conducted as per Mining Rules) so that any adverse effect may be detected in its early state. As per the test conducted till the end of last year no occupational disease has been reported.
- Monitoring the values of different environmental parameters, which may lead to occupational health hazards to the workmen and specifying various control measures.

x. Socio-economic measures:

The socio economic measures suggested for the region, in general are :

- Special care for irrigation should be provided to the locality to maximize the agricultural production and assumed output from the land. This in turn, improves the bargaining capacity of the labour. TSRDS, at present helps farmers by way of giving pump sets etc.
- Alternative domestic fuels (gobar gas/ night soil gas) would minimize fuel wood consumption and would be a very important step to check deforestation without safe guarding the forests, the soil fertility is difficult to maintain. TSRDS has already done some work in this line and is in process.
- Proper road communication to all the surrounding villages is a necessary precondition for hiring labour.
- Different types of contractual labour disorganize the labour unity. Better co ordination and improved life style of the workers is possible if all the workers would be kept under the same umbrella.

In addition to these, the following measures, which TSRDS is presently practicing, are suggested to be taken up on a continuous basis to improve the socio-economic level of the villagers around.

- Generation of employment for the local people. Apart from this, giving constant encouragement to the local villagers to carry out several business like mining, contractor ship, supply of mining materials and also small scale rural business developments like brass material making, sapling production etc.

- Provision of adequate medical and health facilities to the employee and also in the local villages in a systematic manner.
- Regular grant to neighboring schools and colleges and also constant encouragement for cultural activities and sports in local villages.
- Provision of various types of training for the local villages e.g. training for improved agro-techniques, first aid and safety, adult literacy programme etc. to make them self sustaining.
- Assisting the local villages by giving agricultural aids pumps sets etc.

Prepared by:

(G. K. Guin)

RQP

Regd. No.: RQP/BBS/044/2003/A
Dated: 9th May' 2012

PART – C

(PROGRESSIVE MINE CLOSURE PLAN)

12. INTRODUCTION:

a) Name of Lessee: SUKINDA CHROMITE MINE, TATA STEEL LIMITED

b) Location:

Sukinda Chromite Mine lies in the district of Jajpur, Orissa and is connected with Jajpur Keonjhar Road Railway Station on the Howrah-Chennai trunk line of East Coast Railways by 52 KM all weather road. J K Road Railway Station is 336 KM away from Tatanagar Railway Station via Kharagpur and is 337 KM from Howrah, 100 KM from Bhubaneswar-state capital and 156 KM from Paradip, the nearest major port on the Eastern Coast.

The all weather road connecting Sukinda Chromite Mine with JK Road Railway station meets NH-200 connecting JK Road with Bhuban at Mangalpur at a distance of 21 KM and Express Highway no 1 connecting Daitari with Paradip at Duburi at a distance of 32 KM from the mine.

The Key Plan of the lease area of 406 ha, the allotted area of 100ha contiguous with the southern lease boundary of the existing lease and the applied area of 73.685ha is shown in Drawing No. SCM/MP/01/12. The Surface Plan of the leasehold area is shown in drawing no. SCM/MP/02/12 updated as on 01.08.2011. The extent of Latitude and Longitude of the above area is mention below in Table. 12.1.1

Table:12.1.1
Details of Latitude & Longitude

Sl No.	Area Type	Area (ha)	Latitude From	Latitude To	Longitude From	Longitude To	Remarks
1	ML Area	406.00	$21^0$00’39.60”	$21^0$02’5.81”	$85^0$44’27.10”	$85^0$46’22.37”	Mining Lease Area
2	Additional Area	100.00	$21^0$00’27.41”	$21^0$01’1.68”	$85^0$45’22.09”	$85^0$46’51.42”	Allotted for dumping of Overburden
3	Applied Area	73.685	$20^0$59’34.88”	$21^0$00’51.00”	$85^0$46’46.84”	$85^0$47’32.69”	Applied for disposal of Tailing

c) Lease Area and Break up:

The breakup of the lease area of 406 ha, the allotted area of 100ha contiguous with the southern lease boundary of the existing lease and the applied area of 73.685ha is given below in Table 12.1.2.

Table 12.1.2
Breakup of Lease & Other Area

Location	Total Area	Non-Forest Area (ha)	Forest Area (ha)	KF / Gramya Jungle (ha)	DPF/Forest Block No-27 (ha)	Remarks
ML Area	406.000	332.303	73.697	0.085	73.612	Mining Lease Area
Additional Area	100.000	0.000	100.000	4.550	95.455	Allotted for dumping of Overburden
Applied Area	73.685	65.315	8.370	6.428	1.942	Applied for disposal of Tailing

The leasehold area of 406 ha. comprises of 73.697 ha of forestland, which was already broken up prior to 25.10.1980. The diversion proposal for the same was duly submitted and second stage clearance has been obtained for diversion of the entire 73.697 ha. Forest land vide approval letter no. 8-78/96 FC dated 27.01.98 (Annexure-10). Similarly the Forest Clearance for 100ha of additional area for overburden dumping was obtained vide letter no. no. 8-28/2004 FC dated 11.05.2005 (Annexure-21). Similarly the forest clearance for 8.370ha. of corridor involving forest land to the applied area is under active consideration by forest department.

d) Validity of Lease:

The mining lease of Sukinda Chromite Mine comprises of 406 hectares of land in villages Kalarangiatta, Kaliapani, Mahulkhal & Forest Block No. 27 in the Revenue district of Jajpur, Sub-division Jajpur and Thana Tomka. Date of expiry of the mining lease is on 11.01.2013. This Mining Plan is being prepared and submitted under Rule 22 & 24A of the Mineral Concession Rules, 1960 for an area of 406 ha as a part-fulfillment of lease renewal application. The proposals in this Mining Plan have been made for 20 years period form 12^{th} Jan' 2013 onwards.

e) Present Land Use pattern:

The location and extent of lease area, the type of lease area (forest, non-forest etc) and the present land use pattern is given in the in plan SCM/MP/12/12. The activity wise breakup of the land is given below in Table 12.1.3.

Table 12.1.3
Present Land Use

Sl No	Activity	Present Land use (Ha.)
1	Area to be excavated	199.77
2	Storage of Top Soil	0.00
3	Over Burden Dump	37.47
4	Mineral Storage	37.08
5	Infrastructure (Workshop, magazine etc)	3.27
6	Roads (Present L= 7812m, W= 25m)	19.05
7	Railways	0.00
8	Greenbelt	38.53
9	Tailing Pond	15.95**
10	Effluent Treatment Plant	0.32
11	Mineral Separation Plant	18.38
12	Township Area*	36.18
13	Others (to be specified)	0.00
14	Area which will remain untouched	0.00
	Total	406.00

* The township area includes the roads with in the township
** included area of pyroxenite section for temporary storage of tailings

f) Method of Mining and Mineral Processing:

Same as Chapter 4 (a) and 10

12.1 REASON OF CLOSURE:

The final closure of the mine or part thereof and cessation of mining and processing operations is not envisaged within the next five years and even within the validity of the lease either due to lack of demand, uneconomic operations, or exhaustion of the reserves. As such since this is an existing mining lease and operating mine, a Progressive Mine Closure Plan is being submitted along with this document as per provisions of Rule 23B of the MCD (Amendments) Rules, 2003 and amendments made thereof.

12.2 STATUTORY OBLIGATION:

The mine has been working for over the last five decades. All the statutory obligations pertaining to DGMS, IBM, and State Pollution Control Boards are complied with time to time. The legal obligations to be implemented are as follows:

a. No special conditions have been imposed in the lease deed that has been executed in 18.05.1998 for the second renewal of the lease over an area of 406 ha. for a period of 20 years with effect from 12.01.1993 (Annexure-31).
b. The Mining Plan was approved by IBM vide letter no. 314(3)/97-MCCM(C)/MP-7 dated 12.12.1997 enclosed as Annexure-2. Subsequently, the Scheme of Mining has been approved by IBM vide letter no. 314(30/2003-MCCM(C)/MP/S-6 dated 22.10.2003, Progressive Mine Closure Plan has been approved by IBM vide letter no. 314(3) / 2003 – MCMM (C) /MP – S-6 dated 08.08.2005, Modification of Mining has approved by IBM vide letter no. 314(3) / 2007 – MCMM (C) /S-9 dated 05.07.2007, The Scheme of Mining for the period 2008-13 has been approved vide letter no. 314(3)/2008-MCCM(CZ)/MS-17/97, Dated: 02.02.09, subsequent Modification to the Scheme of Mining has been approved vide letter no. 314(3)/2011-MCCM(CZ)/MS-08, Dated: 22.07.11are enclosed as Annexure-3, 4, 5, 6 & 7 respectively.
c. Approval for the diversion of 73.697 ha. of forestland within the lease area of 406 ha. was obtained prior to the second renewal of the lease vide letter no. 8-78/96-FC dated 27.01.1998 which is enclosed as Annexure-10.
d. Approval for diversion of 100ha of Additional forest land beyond Mining Lease area was obtained vide letter no. 8-28/2004 FC dated 11.05.2005 enclosed as Annexure-21.
e. The consent order to operate by the Orissa State Pollution Control Board under Sec. 21 of the Air (Prevention & Control of Pollution) Act, 1981 and under Sec. 25/26 of Water (Prevention & Control of Pollution) Act, 1974 vide letter no. 7552/IND-I-CON-226, dt 03.05.2011, Consent Order No.-1223 is valid upto dt 31.03.2016 is enclosed as Annexure-34. All the conditions imposed in the above letter have been complied.
f. Conditions for adopting a system of deep hole blasting and / or working opencast mines with the help of heavy machinery under MMR 106 (2) (b) from the Director of Mines Safety, Bhubaneswar has already been enclosed as Annexure-18.
g. Approval for modifications in the slope angles vide letter no. BCU/CH-7/P-106/2005/1396 dated 03.07.2006 has already been furnished as Annexure-18.
h. Compliance to the conditions imposed in Environment Clearance is submitted to Ministry of Environment & Forest on six monthly basis. Copy of the same for the period March’11 to Sept’11 is attached as Annexure-35.

12.3 CLOSURE PLAN PREPARATION:

a) Name & address of the applicant

The applicant and the present nominated owner of the mine is Mr. H. M Nerurkar. The copy of photo ID of nominated owner along with passport details towards address proof is enclosed as Annexure-14. The details of address are as follows.

Mr. H. M Nerurkar
Managing Director
Tata Steel Ltd.
At. / P.O. – Jamshedpur
Dist – East Singhbhum
Jharkhand – 831001

Phone: 0657- 2423298
Fax : 0657- 2431818
Email: mdoffice@tatasteel.com

b) Status of the applicant

The Tata Steel Limited is a public limited company having its Steel Works at Jamshedpur in the state of Jharkhand. The company also operates a Ferro Alloys Plant at Joda in Keonjhar district of Orissa state. Besides this, the company operates number of metal and coal mines in many parts of the country to cater its works requirements in respect of various raw materials. The applicant is the Nominated Owner of the company. The resolution of Board of Directors for nominated owner, Memorandum of Association & Article of Association issued under Companies Act etc. is enclosed as Annexure-15 & Annexure- 12 respectively.

The present members of the Board of Directors of TATA STEEL are:

Name	Pan number	Voter ID number	Citizenship	Passport number / DL number
Ratan N. Tata	AAAPT0002F	MT 04 019 060249	Indian	Z 2177343
B. Muthuraman	AAHPB7489R	JVN 1545873	Indian	H 5574492
Nusli N. Wadia	AAAPW0990M	Not allotted	Indian	F 6352000
S.M. Palia	AABPP2138D	Not allotted	Indian	A 6129072
Suresh Krishna	AABPK3154E	Not allotted	Indian	Z 1759392
Ishaat Hussain	AAVPH 6348F	MT 04 019 057708	Indian	Z 2026139
Jamshed J. Irani	AAFPI1888M	BR 50292 333139	Indian	H 6855092
Subodh Bhargava	AAIPB9290R	MT 08 038 363390	Indian	Z 2007543
Jacobus Schraven	----	Not Applicable	Dutch	NT07K92F5
Andrew M. Robb	----	Not Applicable	British	800240979
Karl-Ulrich Kohler	----	Not Applicable	German	507726111
H.M. Nerurkar	ABGPN0776M	JVN 4233342	Indian	G 9227924

c) **Name, Address And Registration Number Of The Recognized Person:**

Mr. G.K Guin
Head (MPP)
TATA STEEL Ltd. Sukinda

Dist	:Jajpur
State	:Orissa
Pin Code	:755028
Phone	: 09238100835
Fax	: 06726268734

Registration No.: RQP/BBS/044/2003/A, valid till 03.02.2013 (The copy of the certificate is enclosed as Annexure-16).

The RQP, Sri. G. K. Guin has been duly authorized by the applicant. The letter of consent is enclosed as Annexure -24.

The Key Person has been assisted by the officers and staff of Planning Cell. The underground part has been prepared by Sri. Haridrumat Behera, RQP, having Registration No. RQP/BBS/093/2010-A valid up to 19.04.2020 (Copy of RQP certificate enclosed as Annexure-17). Services from the departments of Geological Services, Land & Lease, Safety, Environment and the field personnel of Sukinda Chromite Mine were also availed in relevant areas of their expertise.

d) **Name of executing agency:**

The following persons are responsible for implementing the various proposals as mentioned in this document:

- Agent, Mines Manager and Chief (Mining), Mr. Sushanta Kumar Mishra
- Environment Officer, Mr. Dipak Behera.

However, in case of the absence of any of these persons either on account of leave or transfer, the next incumbent holding the above statutory positions shall continue to be responsible.

13. MINE DESCRIPTION:

13.1 GEOLOGY:

Same as Chapter 3 Para 3 (a) & (b) of Mining Plan.

13.2 RESERVES:

Same as Chapter 3 & Table 3.5 to 3.8 of Mining Plan.

13.3 MINING METHOD:

Same as Chapter 4 Para 4(a) & 4 (f) of Mining Plan.

13.4 MINERAL BENEFICIATION:

Same as Chapter 10 of Mining Plan

14. REVIEW OF IMPLEMENTATION OF MINING PLAN/ SCHEME OF MINING INCLUDING FIVE YEARS PROGRESSIVE CLOSURE PLAN UP TO THE FINAL CLOSURE OF MINE:

14.1 Review of Mining Plan/ Scheme of Mining:

The review of the implementation of the approved Mining Plan has already been dealt in great detail in the scheme of mining under para 1.4, 9.0 of Part-I and Part-II respectively.

14.2 Review of Progressive Closure Plan

The salient features of the approved Progressive Closure Plan were reviewed and the deviations are enumerated below:

CHAPTER 10.3 (a): Name & address of the applicant

Same enumerated under para 1.4 (b), Chapter1.1 of Part-I

CHAPTER 10.3 (b): Status of the applicant

Same enumerated under para 1.4 (b), Chapter1.2 of Part-I

CHAPTER 12.3 (f): Production level

Same as enumerated under para 5.3 of Part-I

CHAPTER 4.4: Waste management

Same enumerated under 1.4, chapter 6 of Part-I

Afforestation:

The year wise proposal of afforestation for last three years and compliance thereof is as given below in Table 12.1.1.

Table 12.1.1
Year wise Afforestation

Year	Plan		Actual	
	Nos	Area (ha)	Nos	Area (ha)
2008-09	11750	4.70	122000	4.70
2009-10	21750	8.70	84000	9.50
2010-11	20000	8.00	85000	8.50
Total	53500	21.4	291000	22.7

Total 2150 m of retaining wall using earth, overburden material and boulders were made against the plan of 1700m in the year 2008-09.

Air & Water Quality Management:

As planned, ambient air quality is being monitored at a frequency of twice a week by conducting 24 hour sampling and all the parameters are found to be within prescribed limits. Similarly mine water is being treated in effluent treatment plants before discharging out of the lease hold area to ensure that all the water quality parameters are within the permissible limits Water quality at lease end will continue to be monitored at a frequency of once a month for all water quality parameters as prescribed by the Orissa Pollution Control Board. The year wise ambient air and water quality recorded during the period 2008-11 is enclosed as Annexure-9.

15. CLOSURE PLAN:

15.1 MINED OUT LAND:

The present land use patterns of the entire lease hold area of 406 ha. is as given in the para 1.d. Presently around 4.03 ha of mined out land in the extreme western part of OB II Quarry has been back filled since the ore body is not existent at greater depth. As mentioned in conceptual plan, back filling of quarries and its subsequent rehabilitation by way of plantation shall be taken up after the economic pit bottom depths are reached along the entire strike of the deposit. The conceptual Mining Plan showing the areas that shall be so taken up for back filling have been furnished as shown in Drawing Nos. SCM/MP/05/12. The land use pattern as envisaged till 2017-18 and end of the life of mine over the entire lease hold area of 406 ha. is given in Table 15.1.1.

Table. 15.1.1

Land use Pattern of the Mining Lease Area

Activity	As at the beginning of plan period (2012-13) in ha.	As at the end of Plan period (2017-18) in ha.
Area to be excavated	201.50	175.49**
Storage of Top Soil	0.00	0.00
Over Burden Dump	37.47	76.11
Mineral Storage	30.70	32.86
Infrastructure (Workshop, magazine etc)	7.92	7.92
Roads (Present L= 7812m, W= 25m)	19.05	19.05
Railways	0.00	0.00
Greenbelt	38.53	29.69
Tailing Pond	15.95***	10.00****
Effluent Treatment Plant	0.32	0.32
Mineral Separation Plant	18.38	18.38
Township Area*	36.18	36.18
Others (to be specified)	0.00	0.00
Area which will remain untouched	0.00	0.00
Total	406.00	406.00

Remarks:

- * - includes roads within the colony area
- ** - includes pyroxenite section & excludes OB-II area as backfilled
- *** - includes area of pyroxenite section for temporary storage of tailings
- **** - excludes area of pyroxenite section

As stated earlier the chromite ore bodies in the lease area are all dipping vertically downwards. As mentioned under para 4.0 of the conceptual mining plan, the benches will be pushed back as the quarries are deepened further for extracting the ore. Associated mineral/ overburden generated in 2012-13 till part of 2014-15 shall be stacked in the additional area of 100ha till the space is exhausted. Subsequently the overburden generated during part of 2014-15 till 2017-18 shall be backfilled in OB II quarry in a manner as shown in the drawing no SCM/MP/08/12. However if problem is faced to accommodate more overburden in the backfilled area of OB-II beyond planned period, company may seek for additional area out of applied area of 289 Ha for disposal of associated mineral/ overburden generated during 2018-19 onwards. The rehabilitation of the dumps will be done as follows:

1. Along the periphery of lease boundary where the topography permits wall of at least 0.5m made by stacking boulders will be constructed to prevent the siltation and spreading of solids by water erosion.
2. In order to further prevent the solids (particularly the finer particles from the nickeliferous limonite stacks) from flowing into the adjoining area it is proposed to provide garland drains at the toe of all such stacks wherever the topography so permits. These garland drains shall be provided with stone partitions at intervals for arresting the flow of the solid particles. The drains shall ultimately lead to settling pits followed by an effluent treatment plant before the surface run off from these stacks is ultimately discharged out from the leasehold.
3. In order to prevent the formation of gullies along the slopes of these stacks it is proposed to cover these slopes with coir matting. However the dumps having Ni-limonite Associated mineral/ overburden shall be stabilized by coir matting only as coir matting is not possible in quartzite dumps due to the sharp edges of Associated mineral/ overburden material.

It is further proposed that the slopes of all the associated mineral stacks shall be afforested by planting different varieties of grass and bushy plants along these slopes to further arrest the solids from wind erosion.

As the area for stacking the finished product will be reduced due to the increase in area of mining operations, the top of the overburden dumps will be used for stacking of finished products if required. The details of toe wall construction and rehabilitation of associated minerals stacks is been described in chapter 9 and 15.4.

15.2 WATER QUALITY MANAGEMENT:

There is no perennial source of water like river, stream or nallah running through the lease area. Damasala nallah is flowing from east towards the south and is at a distance of 0.8KM due north from the mining lease. Brahmani river is running at a distance about 15 KM from the lease area. The Sukinda valley experiences annually about 110 cm to 180 cm total rainfall. Eighty percent of total rainfall is during monsoon i.e. June to September. Because of such uneven distribution of rainfall, the major portion is going as surface runoff. Garland drains have been made all around the quarries and dumps. The entire surface water is coursed along such garland drain along the western lease boundary to an ETP located near lease boundary. Settling tanks have been provided along the garland drains to arrest the suspended solids.

Monitoring of Surface runoff water, mine discharge water and ground water are carried out. Water quality analysis is carried out by engaging competent and approved agency by Orissa Pollution Control Board by drawing samples from strategic points. Ground water, mine discharge water before and after treatment (for Cr+6) and Damasala nalah water at the upstream and downstream of the lease is monitored more frequently. Samples are fully analyzed for the parameters specified by Orissa Pollution Control Board. Analysis of Water quality data is enclosed as Annexure-9.

15.3 AIR QUALITY MANAGEMENT:

The air borne particulate matter is the main air pollutant contributed by mining and mineral dressing processes. Apart from the fine sized solid particles resulting during the operational phase of mine, namely drilling, blasting, excavation, transport and milling, particulate matter is carried in the atmosphere due to wind action over mineral and waste dumps as well as tailing disposal areas, which continues even after closure of the mine.

Existing Air Quality Status

During the operational stage the air quality monitoring is been done at different work zone locations like in the open pit faces, processing plants, waste tailing and reject dumps, transport roads, in mining colonies and adjacent villages for three dry seasons, as per IBM/MoEF norms; to draw on climatic data coupled with emission data. The pollutants to be monitored are respirable particulate matter & suspended particulate matter containing significant silica/ silicates, trace & heavy metals, toxic elements and other gases like CO, SOx & NOx. The monitoring stations are as shown in drawing no. SCM/MP/09/12.

To monitor the effectiveness of dust prevention and control actions, it is necessary to compare background levels of airborne dust with conditions down wind, and back this up with adequate meteorological equipments. At present air quality monitoring is carried out at six locations within the core zone by engaging competent and approved agency by Orissa Pollution Control Board. Four separate dust samplers complete with particle size partitioning and independent power generators provide sufficient data. The equipment selected is capable of collection of Sulphur dioxide and Nitrogen oxides samples. The method of deployment of Respirable dust samples are down wind condition at the mine site and in nearby villages where there is chance of dust nuisance. Air quality analysis is carried out twice in a week all around the year. Air Quality Data reveals no adverse impact on the air quality due to the mining activity. Analysis of Air quality data is enclosed as Annexure-9.

Corrective Measures for Prevention of Air Pollution

The corrective measures for prevention of air pollution are as follows:

- Wet drilling is practiced during drilling operations. Drill operators are provided with dust masks.

- Over charging is avoided during blasting to prevent generation of dust.
- Water is sprinkled before loading and unloading to prevent the dust becoming air borne.
- In the haul roads water is sprayed in many runs to suppress the dust generation.
- The dust raised from the dump areas by action of wind will be prevented from become air borne by vegetation of the dumps.
- The entire process in the chrome ore beneficiation plant is wet. Water is added in sufficient quantity in the form of fine spray in the feed hoppers and screens to allay the dust in all other processing plants.
- The exhaust from the generators is let out in to the atmosphere at sufficient height.
- The exhaust from all mobile equipments is monitored and corrective actions are taken to ensure that the emissions are within the norms.

15.4 WASTE MANAGEMENT:

Type, Quality and Quantity of Overburden & Mineral Rejects:

As mentioned under para 6.0, the different types of overburden viz. nickeliferous limonite, serpentinite and quartzite have been defined as associated minerals. The quality of overburden is as follows:

Nickeliferous Limonite		**Quartzite**	
Constituents	Weight %	Constituents	Weight %
Magnetite	6.9	Silica (SiO_2)	90.5 to 94.0
Gadolinite	12.4	Alumina (Al_2O_3)	1.66 to 4.75
Haematite	8.3	Phosphate	0.004 to 0.068
Serpentinite	1.6	**Serpentinite**	
Geothite	10.9	MgO	38.01
Pyrolusite	0.3	SiO_2	34.5
Limonite	22.1	Cr_2O_3	3.54
Nickel Oxide	0.77 to 0.61	CaO	0.35
Chrome	6.3	Al_2O_3	1.61
Co-Mn Oxide	0.07	Fe	8.6
Quartz	30	LoI	15

The yearly generation of overburden / associated mineral from the respective quarries proposed through both opencast and underground means is given in Table 15.4.1. The yearly generation may vary due to changes in production requirements, which is largely governed by fluctuations in the market demand. However, efforts shall be made to maintain the total quantity for the 5 - year period in order to ensure proper development. The quantum of associated minerals to be generated is reproduced as follows:

Table 15.4.1

Associated Minerals / Overburden generation (Lakh Cum)
Opencast & Underground Mining

Year	Generation of Associated Minerals / OB (Lakh CuM)		Total OB (Lakh CuM)
	OB X Quarry (Ni- Limonite)	OB II Quarry (Serpentinite & Quartzite)	
2012-13*	13.00	1.90	14.90
2013-14	54.12	8.63	62.75
2014-15	59.49	8.95	68.43
2015-16	66.36	0.91	67.28
2016-17	66.94	0.96	67.89
2017-18	67.67	1.04	68.71
Total	327.58	22.39	349.96

* OB generation figure for 2012-13 is from 11.01.2013 till 31.03.2013.

Build Up Of Dumps and protective measures:

The yearly build up of dumps from year to year has been shown in the form of plans and sections in drawing no. SCM /MP/08/12. The method of dumping has already been mentioned in para 7.3 of the mining plan.

It is proposed that rehabilitation of the associated mineral stacks shall be taken up. As mentioned all the so-called overburden/ wastes of the past are essentially associated minerals as defined in the mining plan and there is every likely hood that these stacks of associated minerals shall be utilized in the not too distant future.

These stacks have already attained heights of almost 80 m at places due to dearth of space for accommodating associated mineral in the past several years after the reduction of the leasehold area. As may be seen from the surface plan, these stacks have been made in lifts of 10- 30 m each, leaving a berm of 10-15 m between the two lifts. These stacks are quite stable as per studies conducted by CMRI, Dhanbad. The possibility of flattening the overall slopes of these dumps further with a view to increase their stability was explored but the same cannot be flattened any further since these stacks are located just beyond the ultimate pit limits of the quarries and any further widening of the berms would mean that the stacks would encroach into the mineralized area which shall be taken up for mining in the near future. Under the circumstances it is proposed that the following special measures shall be taken up for the rehabilitation of these associated mineral stacks:

5. In order to prevent the siltation and spreading of the solids by water erosion it is proposed that toe walls of at least 0.5 m height made by stacking of boulders shall be constructed at locations where the topography and drainage pattern so permits along the lease boundary. The year wise toe wall construction proposed is as shown in the table 15.4.2 and is shown in the drawing no. SCM/MP/10/12.

Table 11.3.2
Construction of Retaining Wall

Year	Additional Area of 100 ha.	Applied Area of 73.685ha.	Total	Cost	Breadth (m)	Height (m)
	Length (m)	Length (m)	Length (m)	(Rs Lakh)		
2011-12	1000	0	1000	15	1.00	1.00
2014-15	0	3500	3500	53		
Total	1000	3500	4500	68		

6. In order to further prevent the solids (particularly the finer particles from the nickeliferous limonite stacks) from flowing into the adjoining area it is proposed to provide garland drains at the toe of all such stacks wherever the topography so permits. As the garland drains exist to prevent the flow of solid waste to the adjoining leases, the construction of garland drains close to the lease boundary is not envisaged in the next five years. These garland drains shall be provided with stone partitions at intervals for arresting the flow of the solid particles. The drains shall ultimately lead to settling pits followed by an effluent treatment plant before the surface run off from these stacks is ultimately discharged out from the leasehold.

7. In order to prevent the formation of gullies along the slopes of these stacks it is proposed to cover these slopes with coir matting. Coir matting shall be done in the Ni-Limonite dumps were the chances of wash off is more. Coir matting over the OB-II quarry overburden material is not possible due to the very hard nature and sharp edges of overburden material.

8. It is further proposed that the slopes of all the associated mineral stacks shall be afforested by planting different varieties of grass and bushy plants along these slopes to further arrest the solids from wind erosion. The year wise stacks with the lease and additional area of 100 Ha. which have saturated shall be covered for bush plantation. The year-wise proposal for plantation and expenditure to be incurred in the additional area of 100 Ha, with in Mining Leasehold area of 406ha. and in applied area of 73.685 ha along the slopes of associated mineral stacks and in vacant land is as shown in drawing no. SCM/MP/10/12 and is given below in Table 15.4.2.

Table 15.4.2
Year wise proposed Afforestation and Expenditure

Year	Mining Lease Area of 406 ha.		Additional Area of 100 ha.		Applied Area of 73.685ha.		Total Plantation		
	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Area (Ha)	Plantation (Nos.)	Cost (Rs Lakh)
2013-14	2.5	6250	8.00	20000	3.00	7500	13.50	33750	27.0
2014-15	1.6	4000	7.95	19875	3.15	7875	12.70	31750	25.4
2015-16	3.0	7500	8.05	20125	1.25	3125	12.30	30750	24.6
2016-17	3.0	7500	7.85	19625	1.05	2625	11.90	29750	23.8
2017-18	3.4	8500	8.05	20125			11.45	28625	22.9
TOTAL	13.5	33750	39.9	99750	8.45	21125	61.85	154625	123.7

The coir matting shall be done in the Ni- limonite overburden dump of OB-X quarry where the chances of wash off is more but may not be possible in overburden dump of OB-II quarry due to its very hard nature and sharp edges of overburden material.

15.5 TOP SOIL MANAGEMENT:

Quantity of generation of the topsoil in Sukinda Chromite Mine is insignificant as the area to be broken for mining operation is almost saturated. However the small quantity of topsoil, which may be generated, will be used for plantations.

15.6 TAILING DAM MANAGEMENT:

A tailing dam exists within the lease area to store the tailings generated from the Chrome Ore Beneficiation Plant. There is a dirth of space for disposal of tailings in the existing tailing pond. With the reduction of the mining lease area in 1996 during its second renewal, the tailing pond went outside the granted area of 406 ha. A new tailing pond was established within the reduced lease covering an area of 10 ha. The tailing dam is being heightened every year to accommodate the tailing generated from the COB plant.

IIT, Kharagpur was engaged to assess the holding capacity of the tailing pond and the stability of the dam. In their report, IIT, Kharagpur has recommended for the tailing dam heightening to be done to a maximum height of 215 mRL. Presently the dam has already reached a height of 212 mRL. Another 3 meter heightening can be done within the next one month.

The present tailing dam is not having sufficient capacity to cater for the requirement till the end of planned period. Company has therefore has identified and applied for an additional area of 65.315 ha of non-forest Government land for future storage of the tailing which is under process. It may take a further period of atleast two years to acquire this additional land.

Further we also have a proposal to install parallel plate filters in the Beneficiation Plant in order to separate out the water from the tailings slurry/ slimes within the plant premises. With the introduction of these parallel plate filters within a year, the tails shall be in cake/ solid form which can be transported in dumpers to the proposed additional area to be acquired for the purpose. Pilot scale test for these press filters have already been successfully conducted. It is expected that this new method of solid disposal shall substantially reduce the threats associated with the stability of the conventional tailing ponds/ embankments.

During the planned period the dry tailing shall be stored in the applied area of 73.685ha. Presently one study is going on detailing the future tailing generation, its management, storage at the applied area, runoff management, confinement and mitigation measures to be followed while transportation of dry tailings. Details are also being working out about management of tailing after the closure of plant with method of stabilization, rehabilitation. R&D study is also going on exploring the possibility of reuse of tailings.

Physical Stability:

No change is proposed in the management of existing tailing dam which is going to be exhausted very soon. The tailing which shall be stored the pyroxenite section of OB-II Quarry shall be stored within the earthen embankment all around the pit. Water shall be regularly pumped to minimize any hydrostatic pressure. The dry tailing shall be stored in the applied area. Retaining wall with garland drain shall be provided all around the storage area for runoff management.

Chemical Stability:

The existing tailing dam shall be left to dry naturally. However backup pumping arrangements shall be kept to dewater the monsoon water. Similarly water from the tailing stored in the temporarily discontinued pyroxenite section of OB-II quarry shall be tried to keep devoid of any excess water by continuous pumping. All the water pumped shall be reused in the Chrome Ore Beneficiation Plant. Thus, no water from the pit shall be allowed to be discharged out of the lease.

15.7 INFRASTRUCTURE:

Decommissioning of any of the existing infrastructural facilities available such as public roads, plants, power lines, buildings and structures, water treatment facilities, water supply sources in the area etc. is not envisaged within the next 5 years. Thus, measures to be taken for their disposal, physical stability, maintenance and restoration of the land do not come within the scope of this Progressive Mine Closure Plan.

15.8 DISPOSAL OF MINING MACHINERY:

Decommissioning or disposal of existing mining machinery is not envisaged during the next 5 years. All the mining equipments being used within the mine are of mobile type and in case some of these equipments are decommissioned due to obsolescence or otherwise damaged beyond repair the company has a system to dispose off such old equipment in its entirety or as scrap by auctioning the same to outside parties.

15.9 SAFETY & SECURITY:

Drop gates have been provided at the entry of all access roads leading to the mine. These gates are manned round the clock by the security staff to prevent the un-authorized entry of persons and outside vehicles. No part of the mine is proposed to be abandoned during the next 5 years.

15.10 DISASTER MANAGEMENT AND RISK ASSESSMENT:

Potential Emergency Conditions:

Emergency condition may arise in Sukinda Chromite Mine by virtue of the following:

- Land Slide
- Breach of Slime dam
- Fire/ Explosion
- Accidents related to man and machine

Steps to prevent emergency situation:

The following steps are taken to prevent occurrence of the emergency situation:

a) Land Slide:

a. Regular slope monitoring is being done as per the conditions stipulated by Director General of Mine Safety (once in a quarter in the dry season and once in a fortnight in the rainy season.)
b. Catch water drain are provided around the quarries so as to divert the rainwater.
c. Whenever any local bench failure occurs, the area around it shall be visually inspected to ascertain the nature of failure and to take timely preventive action.
d. The over burden dumps are maintained at a slope same as natural angle of repose to prevent slope failure as per the guidance of approved byelaws.

b) Breaching of Slime Dam:

An officer from COB plant inspects slime dam once in a week and the findings are recorded in a register. During the inspection the officer concerned checks for erosion to the walls, seepage/ leakage of water, free board (to be maintained not less than 2m above the water level). If the officer notices any problem, it is brought to the notice of Head (Plants) who in turn arrange to do away with the problem.

c) Fire/ Explosion:

i) Lightening arresters are fixed in the magazine.
ii) No naked fire is permitted in the explosive magazine and diesel storage tank premises.
iii) Flammable materials like grease, cotton waste etc are not allowed to accumulate these premises, which may cause fire hazard.
iv) Explosive magazine is guarded round the clock.
v) The construction of underground diesel tank has been done as per the approved design.
vi) The exhaust silencer of the diesel tanker and explosive van has been fitted with flame arrester.
vii) Fire extinguishers have been provided in the explosive magazines, diesel tankers and diesel storage tanks.
viii) Necessary training has been imparted to the drivers of the tankers.
ix) The approved code of fire fighting procedure for Sukinda Chromite Mine is followed.
x) No naked fire is permitted and smoking has been prohibited inside the canteen, plant and office premises.
xi) The LPG storage shed is adequately ventilated near the ground level and near roof. Un authorized entry in to the storage shed is prohibited.
xii) The LPG cylinders are not stored more than 3 m high when stacked vertically. When stored horizontally, the cylinders are not stacked more than 5 m high and 7 m high in case of full and empty cylinders respectively.
xiii) At the time LPG storage the surroundings are being kept clean and free from all flammable materials, waste vegetation etc.
xiv) Manual operated emergency trip switch is provided in the COB plant switchyard.
xv) All the protective relays are tested once in a two-year for the protection of the equipment. All the earthling systems of COB plant HT & LT installations are tested once in a year.

d) Accidents to Man & Machine:

All the safety practices in relation to man and machine are followed under the guideline of safety department as per the statutory norms.

Emergency Plan:

Land Slide:

Monitoring of the pit slope is done once in 3 months as per the prescribed guideline. If the movement of the strata is found to be more than 5mm/ meter, the movement is treated excess. In such eventualities, repeated measurements shall be taken. On getting repeated similar excess readings, the area to be declared unsafe and the men and machinery shall be removed from the area till the reading stabilizes.

Breaching of Slime dam:

In case of breaching of slime dam:

- The input to the dam shall be stopped forthwith.
- Immediate actions shall be taken for withdrawal of men and machine from the downstream areas to safe places.
- Simultaneously the bridging of the breach shall be done by putting extra earth with the help of mining machinery to arrest the flow of the slime.
- The affected area shall be fenced off to prevent the inadequate entry of any person. There has been no nearby nallah, stream or human habitat adjacent to the dam. Precautions shall be taken to prevent the flow of the slime to the nearby land.

Director of Mine Safety, Bhubaneswar region shall be informed for further guidance.

Explosion:

In case of explosion there will be one or more of the following manifestations:

- Fire & Missile of Fire: the mine is having full-fledged fire fighting cell with trained staff and requisite fire fighting appliances. The water sprinkler deployed in the mine is also kept as one of the resources for firefighting at the time of need. An exhaustive procedure and work instructions are in place as per the guideline of ISO: 9001:2000, ISO 14001 and ISO: 18001.
- Missiles of Broken/ damaged materials: the unsafe material shall be removed and the injured persons shall be taken care of by rendering them first aid and / or sending them to medical treatment.

Mock drill is done once in six months except for fire, which is done once in a quarter. The emergency preparedness planning is updated for any deficiency. The concerned departmental heads are responsible for conducting the mock drills. After every mock drill necessary training shall be imparted to the concerned employees on the emergency preparedness and response. The concerned procedure for emergency preparedness undergoes amendments for any deficiency notice during mock drill or occurrence of such situations.

15.11 CARE AND MAINTENANCE DURING TEMPORARY DISCONTINUANCE:

In the event of any temporary discontinuance of mining activity either due to any court order or statutory requirements or other unforeseen circumstances all efforts shall be made to ensure the following minimum services:

- Uninterrupted power supply,
- Uninterrupted drinking water supply and operation of drinking water treatment facilities,
- Uninterrupted medical and ambulance services,
- Uninterrupted security & fire fighting services,
- Uninterrupted operation of the dewatering pumps within the mine,
- Uninterrupted operation of all effluent treatment plants,
- Laboratory services for checking the quality of drinking water and water that is discharged after treatment,

Necessary staff for ensuring the operation and maintenance of the above services shall continue to remain on duty. The entire mine and plant premises including electrical installations shall be regularly inspected in order to ensure that the workings are safe. In case any unsafe or hazardous conditions are noticed, actions as enumerated above under the emergency plan shall be taken.

16. ECONOMIC REPERCUSSIONS OF CLOSURE OF MINE & MANPOWER RETRENCHMENT:

Not envisaged under the scope of present Progressive Mine Closure Plan and shall be submitted before the final closure of the mine.

17. TIME SCHEDULING FOR ABANDONMENT:

Not envisaged under the scope of Progressive Mine Closure Plan, will be duly submitted during Final Mine Closure Plan. Time scheduling for abandonment is given in Table 17.0.1

Table – 17.0.1
Time Schedule For Abandonment

Month		Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9	Month 10	Month 11	Month 12	2nd Year	3rd Year	4th Year
Activity	**Details**															
Decommissioning/ Demolition	Demolition of Structure															
	Roads/ Railways															
	Electrical Installation															
Removal of equipment & machinery	Shifting of heavy equipment															
Site Safety	Providing fencing															
	Retaining Wall															
	Reclamation of dumps															
Rehabilitation of workings	Plantation															
Maintenance / Monitoring during and after closure operation	Maintenance & monitoring for 3 years after closure															
Retrenchment & Rehabilitation	Rehabilitation															
Research & development	Not envisaged															

18. ABANDONMENT COST:

Not envisaged under the scope of Progressive Mine Closure Plan, shall be duly submitted during the final mine closure plan. The abandonment cost has been estimated to be around Rs.5.40 Crores. and breakup of same is given below in the Table 18.0.1

Table 18.0.1
Calculation for abandonment cost

Sl.No.	Activity	Action	Estimated Cost (Rs)	Remarks
1	Decommissioning	Demolition of Administrative Building, Workshop, Canteen, COB Plant, LOP Plant, Magazine & Weighbridge		Most of the residential quarters have been constructed with partial funding from Welfare Cess, the same are not envisaged to be demolished and has been assumed that the same shall be handed over to the Govt.for their future use
		a) Steel Structure	6120000	
		b) RCC Structure	9075000	
		c) Electrical Installation	724500	
		d) Brick work	1155000	
		e) U/g cables	300000	
		f) Plant Equipments	3284000	
		f) Removal of debris	3375000	
2	Removal of Infrastructure	Transportation		The demolished structures of the Items described in shall be transported for future use / centralised scrap yard
		a) Steel Materials	4800000	
		b) Miscellaneous Material Handling	500000	
3	Removal of Eqpt	Shifting of Eqpt.& HEMM	4800000	It is assumed that, the equipment will be utilised in other mines belongings to the company. The cost of transportation of mobile equipment by trailor has been considered.
4	Site Safety	a) Providing Fencing at Quarry	3430000	Fencing around the proposed lagoons has been considered as per statute (DGMS Circular 11/1959) as @ 9800 mtrs X Rs.350/Mtr.
		b) Providing Security for Item 1 & 2	750000	Security establishment required for the closure operation
5	Remediation & Mitigation measures	a) Retaining Wall	2000000	Retaining wall over 2000 Mtr @ Rs.1000/Mtr.
		b) Reclamation & Rehabilitation of dump	1920000	Plantation cost @ Rs.30000/- per Hect. for 64 Hect.as per Forest dept norms.
6	Reclamation & Rehabilitation of Workings	b) Plantation	8670000	Plantation cost @ Rs.30000/- per Hect. for 289Hect.
7	Administrative / Management for Maintenance & monitoring during and after closure operation	a) Maintenance & Monitoring during closure operation	900000	It is assumed that, during decommissioning of infrastructure, the establishment cost @ Rs.300000/Month X 3 Months
		b) Maintenance & Monitoring after closure for 3 yrs.	2160000	Deployment of 4 Supervisors for monitoring of closure activities for 3 Years.
8	Retrenchment & Rehabilitation	Rehabilitation of employees	0	It is assumed that the employees will be re-deployed within the company.
9	Research & Development	Research & development required for closure activity	0	We don't envisage any R & D jobs to be carried out.
TOTAL			**53963500**	

The year wise reclamation & rehabilitation cost as envisaged during the plan period is given below in Table 18.0.2.

Table 18.0.2

Item	Details	Year	Area (Hect.)	Quantity	Expenditure (Rs. Lakh)	Remarks
			Proposed	Proposed	Proposed	
Reclamation and rehabilitation of mined out land/ area	i) Backfilling	2014-15	22.00	68.43	3220	Cost is part of normal mining job.
		2015-16	7.50	67.28	3220	
		2016-17	7.50	67.89	3220	
		2017-18	5.00	68.71	3240	
		Total	42.00	272.31	12900	
	ii) Afforestation in backfilled area	Nil	Nil	Nil	Nil	To be taken up after backfilling is over.
	iii) Other (Please specify) e.g. afforestation on exhausted benches	Nil	Nil	Nil	Nil	Not envisaged during the Plan period
	iv) Pisciculture	Nil	Nil	Nil	Nil	
	v) Converting into water reservoir	Nil	Nil	Nil	Nil	
	vi) Picnic spot	Nil	Nil	Nil	Nil	
Stabilisation and rehabilitation of dumps	i) Terracing	Nil	Nil	Nil	Nil	
	ii) Pitching	Nil	Nil	Nil	Nil	
	iii) Construction of parapet wall/ retaining wall at toe of dumps	2014-15	0.350	3500 m	53	At Applied area of 73.685 ha
		Total	0.350	3500 m	53	
	iv) Construction of check dams along the slope of vallies etc.	2012-13	0.025	50 m	25	
		Total	0.025	50 m	25	
	v) Construction of settling ponds (Garland drains etc.).	2012-13	0.050	500 m	12.5	
		Total	0.050	500 m	12.5	
	vi) Desiltation of settling ponds channels	2013-14	0.0035	10 No.	1.0	
		2014-15	0.0035	10 No.	1.0	
		2015-16	0.0035	10 No.	1.0	
		2016-17	0.0035	10 No.	1.0	
		2017-18	0.0035	10 No.	1.0	
		Total	0.0175	10 No.	5.0	
	vii) Afforestation on dumps (With in Mining Lease Area of 406 ha)	2013-14	2.50	6250 Nos.	5.0	Refer Table No; 15.4.2
		2014-15	1.60	4000 Nos.	3.2	
		2015-16	3.00	7500 Nos.	6.0	
		2016-17	3.00	7500 Nos.	6.0	
		2017-18	3.40	8500 Nos.	6.8	
		Total	13.50	33750 Nos.	27.0	
	viii) Others (Please specify) Within Additional Area of 100 ha and in Applied Area of 73.685 ha)	2013-14	10.00	25000 Nos.	22.00	Refer Table No; 15.4.2
		2014-15	10.15	25375 Nos.	22.20	
		2015-16	8.75	21875 Nos.	18.60	
		2016-17	8.55	21375 Nos.	17.80	
		2017-18	8.10	20250 Nos.	16.10	
		Total	58.15	145375 Nos.	341.7	
Reclamation of barren area within lease	i) Afforestation (green belt building)	Nil	Nil	Nil	Nil	
	ii) Others (Avenue Plantation)	Nil	Nil	Nil	Nil	

Environmental Monitoring (Core Zone)	i) Ambient air quality	Nil	Nil	Twice a week	50 Lakh per annum	At six locations with in ML area
	ii) Water quality	Nil	Nil	Monthly		Inlet & Outlet of ETP, STP, WTP, Oil & Grease pit, Tube wells
	iii) Noise level Survey	Nil	Nil	Quarterly		At 11 locations in mines
	iv) Ground vibration	Nil	Nil	Yearly		Blast vibration study by CMRI.
	v) Others (Please specify)	Nil	Nil	Seasonal		
Environmental Monitoring (Buffer Zone)	i) Ambient air quality	Nil	Nil	Quarterly		In 10 villages in Buffer Zone
	ii) Water quality	Nil	Nil	Quarterly		Water samples from nearby villages to test Cr+6

19. FINANCIAL ASSURANCE:

The amount of financial assurance to be deposited has been calculated based on the area proposed to be used for mining and allied activities at the rate of Rs. 25000 / hectare. The area to be utilized has been evaluated on the basis of the conceptual mining plan/ land use pattern at the end of the scheme period, which is enclosed as drawing no SCM/MP/13A/12. The change in area at the beginning of the scheme period vrs. That at the end of scheme period has also been shown in the drawing no. SCM/MP/13D/12. For this, the areas put to use for mining and allied activities and the area to be considered for financial assurance has been calculated as in the table 19.1 given below.

Table 19.1
Land use Pattern & Area considered for Financial Assurance

Sl No.	Type of Land Use	Area in Hect.				
		Present Land Use (31.03.2012)	Additional Area requirement during the plan period	As at the End of Plan Period (2017-18)	The area considered as fully reclaimed and rehabilitated	Net area considered for calculation of financial Assurance
		A	B	C=A+B	D	E=C-D
1	Area to be Excavated	199.77	-24.28	175.49**	0.00	175.49
2	Storage of Top Soil	0.00	0.00	0.00	0.00	0.00
3	Over Burden Dump	37.47	38.64	76.11#	0.00	76.11
4	Mineral Storage	37.08	-4.22	32.86	0.00	32.86
5	Infrastructure (Workshop, magazine, U/G infrastructure etc)	3.27	4.65	7.92	0.00	7.92
6	Roads	19.05	0.00	19.05	0.00	19.05
7	Railways	0.00	0.00	0.00	0.00	0.00
8	Greenbelt	38.53	-8.84	29.69	29.69	0.00
9	Tailing Pond	15.95***	-5.95	10.00	0.00	10.00****
10	Effluent Treatment Plant	0.32	0.00	0.32	0.00	0.32
11	Mineral Separation Plant	18.38	0.00	18.38	0.00	18.38
12	Township Area*	36.18	0.00	36.18	0.00	36.18
13	Others (to be specified)	0.00	0.00	0.00	0.00	0.00
14	Area which will remain untouched	0.00	0.00	0.00	0.00	0.00
	Total	406.00	0.00	406.00	29.69	376.31

Remarks:
* - includes roads within the colony area
** - includes pyroxenite section and excludes OB-II backfilled area
*** - includes area of pyroxenite section for temporary storage of tailings
**** - excludes area of pyroxenite section
# - includes area backfilled in OB-II

After subtracting the area already rehabilitated and reclaimed, the area to be considered for financial assurance is calculated to be 376.31 ha. The financial assurance for the above mentioned area @ Rs 25000 per ha. is calculated to be Rs 9407750 (Ninety four lakh seven thousand seven hundred fifty only), already submitted in form of a Bank Guarantee No.

0006BG00009013 dated 19.04.2012 and validity till 31.03.2018 drawn from ICICI Bank in favour of the Regional Controller of Mines, Bhubaneswar is enclosed as Annexure-25.

20. CERTIFICATE:

Certificate duly signed by nominated owner, to the effect that said closure plan complies to all statutory rules, order made by the Central or State Govt., statutory rules, regulations, court etc whatever necessary and whenever any specific permission is required, the concerned authorities will be approached has been attached as Annexure –26.

21. PLANS, SECTIONS ETC.:

The list of plans & sections enclosed along with this Progressive Mine Closure Plan is as given below:

1. SCM/MP/12/12 : Present Land use Plan
2. SCM/MP/13A/12 : Financial Assurance Plan
3. SCM/MP/14/12 : Progressive Mine Closure Plan

Prepared by:

(G. K. Guin)

RQP

Regd. No.: RQP/BBS/044/2003/A
Dated: 9th May' 2012

ANNEXURES

PHOTOGRAPHS