Energy Efficient Flowsheet Development for the Heemskirk Tin Project

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ABSTRACT

This paper details the optimization of the flowsheet developed for the Severn tin deposit, which forms the major ore source for the proposed Heemskirk Tin Project. The Heemskirk tin project is located in Western Tasmania, Australia. It is estimated to contain 6.28 million tonnes of ore, grading 1.14% tin. The optimized flowsheet, which is based on a detailed investigation and assessment of the ore properties and responses to separation techniques, applies the principles of pre-concentration and progressive liberation to minimize comminution energy input while maximizing metallurgical performance.

The outcomes of the laboratory scale investigations resulted in a number of important improvements to metallurgical performance, reductions in comminution energy input and flowsheet simplifications. The most significant flowsheet changes have been (a) the selection of a coarser primary grind in conjunction with heavier regrinds which provide the benefit of reduced tin loses due to fines production and a reduction in comminution energy input and costs, and (b) the elimination of the heavy media separation circuit simplifying the overall flowsheet and minimizing associated tin losses.

Further changes include optimization of the sulfide flotation, gravity, de-slime and tin flotation circuits, which in conjunction with the above changes have led to ~10% improvement in tin recovery.

Based on the outcomes of the optimization program overall tin recovery is now estimated at 79.5% compared with ~70% for the previous flowsheet.

Keywords: Tin, Composition, Mineralogy, Ore Properties, Grind Size, Flowsheet Optimization, Flotation, Concentrate Leaching, Progressive Liberation, Pre-concentration, Comminution

INTRODUCTION

This paper details the optimization of the flowsheet developed for the Severn tin deposit, which forms the major ore source for the proposed Heemskirk Tin Project. The optimized flowsheet, is based on a detailed investigation and assessment of the ore properties and responses to separation techniques. This particular project was chosen as the subject for this paper, as it applies the principles of pre-concentration and progressive liberation with the aims of minimizing comminution energy input, while maximizing metallurgical performance.

Pre-concentration is the upgrading of ore by the selective removal of gangue or low grade material prior to further stages of processing. Whereas progressive liberation, although not a commonly used term, aims to separate valuable minerals either from further downstream processing or an alternate processing route as soon as they are sufficiently liberated, at the coarsest possible particle size.

While the paper showcases the potential benefits, it also highlights the potential trade-offs of pre-concentration and progressive liberation methods.

Project background

The Heemskirk tin project is a greenfields development project, 100% owned by Stellar Resources Limited. The project is located in Western Tasmania, Australia. It is estimated to contain 6.28 million tonnes of ore, grading 1.14% tin.

The Heemskirk Tin deposits comprise three deposits associated with a Devonian granite intrusion within the Proterozoic and Cambrian sedimentary and volcaniclastic rocks of north-west Tasmania. The deposits occur as steeply dipping, stratabound orebodies covering an area of approximately 600 by 500 meters (m). The contained tin occurs as cassiterite within the stock-work and replacement style sulfide mineralization. The primary sulfides present include pyrite and pyrrhotite.

As currently envisaged the project involves the development of an underground mining operation and concentrator to treat 600,000 t/y of ore.

To date the overall project has progressed through to completion of prefeasibility study, and a subsequent flowsheet optimization program for the Severn tin deposit, which forms the major ore source for the project. The project is now poised to enter a definitive feasibility study.

ORE CHARACTERISTICS

The flowsheet optimization program was focused on a global composite produced from the Severn deposit. Head analysis of the Severn composite shown in Table 1 is in close agreement with the average of the overall Severn Mineral Resource.

Table 1. Severn Bulk Composite Head Analysis

Sn	Sn Acid Sol	Fe	As	S	SiO_2	CaO	MgO	Al_2O_3	Mn	Cu	Pb	Zn
%	ppm	%	%	%	%	%	%	%	%	%	%	%
1.00	170	26.1	0.28	14.6	30.1	0.71	2.74	6.29	0.40	0.09	0.63	0.77

The Severn ore is the harder of the Heemskirk ore types, with a Bond Rod Mill Work index of 20.5 kWh/t and Bond Ball Mill Work index of 20.1 kWh/t. Being a very competent ore maximizing comminution efficiency is a key consideration for the project.

Mineralogy

Detailed quantitative optical mineralogy of the Severn ore was completed via MODA (McArthur Ore Deposit Assessments Pty Ltd).

The minerals identified in the ore in approximate descending order of abundance were;

- Gangue ~68.6% (quartz, silicates, carbonates, rutile, carbonaceous matter, fluorite, talc, etc.)
- Pyrite ~ 24.5%
- Pyrrhotite ~ 3.9%
- Magnetite ~3-4%
- Cassiterite ~1.3%

Table 2 summarizes the liberation of cassiterite and the major sulfides present, along with cassiterite associations.

Table 2. Cassiterite, Pyrite and Pyrrhotite Liberation and Cassiterite Associations

Fraction	Free (%)			Cassiterite Associations (%)		
μm	Cassiterite	Pyrite	Pyrrhotite	Binary with sulfides	Binary with gangue	Ternary+
+212	0	46	53	2	71	26
+150	17	49	62	9	41	32
+106	28	67	73	8	45	17
+75	46	74	70	2	41	11
+53	59	70	67	3	36	2
+38	66	85	91	4	25	4
+20	72	87	76	2	21	3
+8	83	97	88	3	14	0

While cassiterite is not well liberated until quite fine sizes (<75 μm), the major sulfides present are adequately liberated at coarse size to allow rejection by flotation. Notably, the majority of cassiterite composites are associated with non-sulfide gangue.

ORIGINAL FLOWSHEET

Figure 1 shows a simplified block flow diagram for the basic flowsheet which was developed during the PFS and further refined and optimized during the subsequent optimization program, based on the outcomes of a detailed metallurgical testing program.

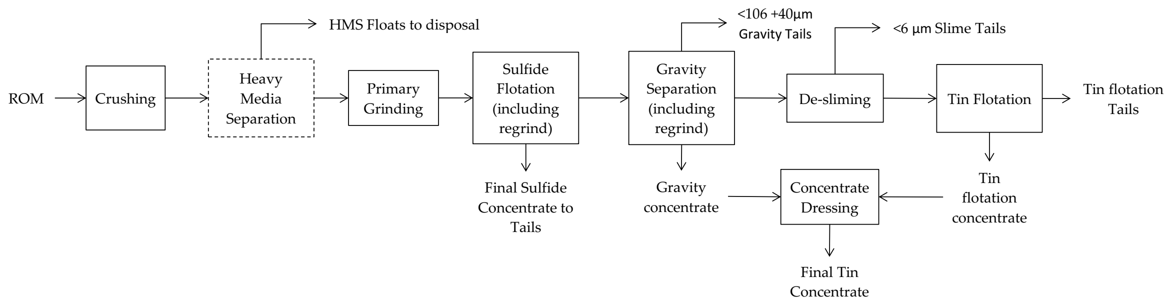

Figure 1. Simplified Block Flow Diagram

Total ore throughput for the project is low at nominally 75 t/h (600 kt/y), meaning only a small concentrator is required. However, the ore is very competent with a BWI of 20.1 kWh/t and RWI of 20.5 kWh/t, and requires a relatively complex flowsheet for effective treatment. The project is reasonably capital constrained and has a complex flowsheet so one aim of the optimization program was to simplify the circuit where sensibly possible.

From a comminution perspective key circuit objectives are to achieve adequate liberation while;

- minimizing tin losses to fines by minimizing over grinding of tin
  - Gravity separation is the major tin recovery method in the circuit, and performance drops off markedly below ~40 µm
  - Cassiterite with an SG of ~6.8-7 is relatively soft and has a strong tendency to over grind.
- minimizing slimes generation
  - As slimes detrimentally impact tin flotation selectivity de-sliming is required ahead of the tin flotation circuit.

The primary comminution circuit consists of two stage crushing followed by a rod mill in open circuit, then ball mill operating in closed circuit.

The PFS originally considered inclusion of heavy media separation prior to the primary grinding circuit as a means of pre-concentrating the ore, and reducing primary grinding circuit energy input. The merits of HMS are discussed further below.

Following the primary grinding circuit the sulfide flotation circuit plays the role of pre-concentrating the feed, rejecting ~27% of the feed mass, ahead of gravity separation, while also reducing sulfur to manageable levels for both gravity separation and subsequent tin flotation.

The sulfide flotation circuit consists of a single roughing train with regrind and two stage open circuit cleaning of the rougher concentrate. A regrind of nominally P80 ~ 25 µm of the rougher concentrate is required to reduce tin losses to low levels. This is achieved via stirred milling.

The gravity separation circuit forms the major tin recovery circuit. Gravity circuit feed is classified by a combination of cyclones and screens to produce feed to separate "coarse gravity" (nominally

>106 μm) and "fine gravity" (nominally <106 μm) circuits, with classification overflow feeding through to the de-sliming circuit.

Both the coarse and fine circuits consist of roughing via spirals and cleaning via tables. Coarse gravity tailings and fine gravity middlings are recirculated to the gravity circuit regrind, which consists of a conventional ball mill. Gravity regrind mill discharge is recirculated to the gravity circuit feed classification stage. With a target regrind P_{80} of ~ 52 μm for the major process stream through the overall circuit, the gravity regrind is a significant grinding duty within the overall flowsheet.

The combine gravity circuits produce a crude concentrate which is upgraded to saleable tin grades in the concentrate dressing circuit by a combination of sulfide flotation, magnetic separation and acid leaching to remove the major diluents; pyrite, pyrrhotite, magnetite and siderite.

A discardable tail, nominally <106 >40 μm, is produced by super fine screening of the fine gravity circuit tails allowing the rejection of an additional 20% of the new feed mass prior to de-sliming and tin flotation.

As slimes detrimentally impact tin flotation selectivity de-sliming is required ahead of the tin flotation circuit. Depending on de-slime cut point the "slime tails" can potentially be the largest tin loss from the circuit. Reducing losses by decreasing de-sliming cut size must be traded off against decreased tin flotation performance in feed to the circuit.

Cassiterite flotation is inherently challenging suffering from poor selectivity against various carbonate and silicate minerals, along with strong interference from sulfide minerals and slime minerals.

The tin flotation circuit consists of a sulfide scavenging stage to reject remaining sulfide mineral followed by tin roughing flotation and two stage open circuit cleaning of the rougher concentrate. The tin flotation circuit produces ~25% grade tin concentrate (post leaching) which is heavily diluted with siderite and various silicates. This is combined with the gravity concentrate (~55% tin post leaching) within the dressing circuit where it is leached with sulfuric acid to remove carbonates.

OPPORTUNITIES TO IMPROVE ORIGINAL FLOWSHEET

The key opportunities for pre-concentration and reduction in comminution energy input, while maximizing metallurgical performance were identified as;

- Heavy media separation (HMS)

  The Heemskirk ores have shown reasonable amenability to HMS as a pre-concentration method to allow the early rejection of gangue. However, more thorough analysis showed higher revenue without HMS despite capital and operation cost reductions as a result of the HMS, this is mainly due to the elimination of tin losses from the proposed HMS circuit and the modest amount of material rejected by HMS.

- Sulfide flotation and coarsening of primary grind size

  Within the Heemskirk flowsheet the sulfide flotation circuit plays the role of pre-concentrating the feed to the gravity separation circuit, while rejecting the bulk of the sulfide minerals which would otherwise interfere with tin flotation. During the course of the flowsheet optimization program it was identified there was a significant difference in the liberation of the sulfides and cassiterite; the major sulfides present are adequately liberated to allow rejection by flotation at coarse size, while cassiterite must be ground to significantly finer size to achieve acceptable recovery to saleable tin concentrate grades. Although coarsening of the primary grind size would require significantly heavier regrind within the gravity circuit, this represented an opportunity for reducing comminution energy input per unit tin, and more importantly reduced production of tin fines and slimes.

HEAVY MEDIA CIRCUIT ANALYSIS

Prior to the optimization program the process flowsheet included heavy media separation (HMS) prior to the primary milling circuit as a means of pre-concentrating the ore, and reducing primary grinding circuit energy input.

Inclusion of a HMS circuit was expected to provide the advantages of rejecting mass early in the circuit, leading to reduced downstream equipment sizes, capital costs, and reduced downstream operating costs. However, this comes at the expense of increased circuit complexity in an already complex circuit, increased tin losses (via the HMS floats product), and increased tin loss opportunities (HMS introduces another tailings stream).

Given these issues and the variability seen in results of the heavy media and heavy liquid testwork, a review of the merits of inclusion of HMS in the flowsheet was completed, along with additional HMS testwork. The review included assessment of the change in CAPEX, OPEX, tin recovery and net revenue with and without HMS included in the flowsheet.

As the impacts of the removal of HMS on CAPEX and OPEX were expected to be sensitive to the mass rejection across the HMS circuit two cases were considered;

1. 23% mass rejection – as per the PFS assumptions
2. 34.5% mass rejection – as per results of the optimization program testwork

The key outcomes of this analysis are summarized in Table 3.

Table 3. Summary: Expected Impact of Removal of Heavy Media Separation Circuit on Capital and Operating Costs

		Based on PFS PDC assumptions	**Based on optimisation testwork results**
Mass rejection in HMS		**23.3%**	**35.4%**
Sn loss in HMS		3.0%	3.9%
CAPEX:			
With HMS	A$ millions	$80.0	$80.0
Without HMS	A$ millions	$79.2	$81.7
Change in CAPEX	**A$ millions**	**$0.9**	**-$1.7**
OPEX:			
With HMS	A$ millions/y	$18.9	$18.9
Without HMS	A$ millions/y	$19.3	$20.2
Change in OPEX	**A$ millions/y**	**-$0.4**	**-$1.4**
Recovery Improvement necessary to offset increased in OPEX without HMS		0.3%	1.0%
Expected recovery improvement without HMS		**2.2%**	**2.8%**
Estimated improvement in net revenue without HMS	**A$ millions/y**	**$2.6**	**$2.6**

At the lower mass rejection considered, 23%, the analysis suggests removal of HMS would give a minor reduction of A$0.9 M in total CAPEX, while at 35.4% mass rejection an increase in total CAPEX of A$1.7 M was estimated. Within the accuracy of the CAPEX estimates (+/- 25%) these changes are not considered significant. Clearly a much larger mass rejection would need to be achieved across the HMS to provide a meaningful reduction in CAPEX.

In both cases considered OPEX is expected to increase with the removal of HMS due to increased throughputs, particularly in the primary grinding and gravity (particularly gravity regrind) circuits. Assuming a mass rejection of 23%, OPEX is expected to increase by of the order of A$0.4 M/y (~A$0.70/t ROM), while at 35.4% mass rejection OPEX is expected to increase by of the order of A$1.4 M/y (~A$2.30/t).

These increases in OPEX would require an improvement in overall tin recovery of 0.3% and 1% respectively to offset these and remain revenue neutral. This compares with an expected increase in recovery due to the elimination of tin loss with the HMS floats of 2.2% and 2.8% respectively.

That is the increase in OPEX is more than offset due to the increase in tin production leading to an increase in overall "net revenue" of approximately A$2.6 M/y for both of the without HMS cases compared to with HMS.

- In this application, HMS proved to be an unattractive means of pre-concentration or improving energy efficiency due to limited tin liberation at the coarse sizes required for HMS, and limited mass rejection.
- Exclusion of HMS from the flowsheet simplifies the overall process flowsheet, reduces tin losses and eliminates an additional loss opportunity (via HMS floats product) for tin.

Other general considerations for application of HMS;

- Would inclusion of HMS allow an increase in overall ore reserves – Would a reduction in cut-off grade and average ROM grade allow reserves to be increased? Can the additional dilution be effectively rejected via HMS?
- Would inclusion of HMS allow mining costs to be reduced by adopting a less selective mining method?
- Could any mineralized waste which must be mined be converted into ore by treatment via HMS?

In this case the answer to these questions was no.

SULFIDE FLOTATION AND COARSER PRIMARY GRIND ANALYSIS

In the PFS the primary grinding circuit consisted of a rod mill in open circuit followed by a ball mill in closed circuit with cyclones, targeting a primary grind size of P_{80} ~ 130 μm.

The optimization test work showed the primary grind could be significantly coarsened from P_{80} ~ 130 μm to P_{80} ~ 250 μm without particularly impacting the sulfide flotation performance.

Coarsening the primary grind size has the following advantages;

- The tin size distribution feeding the gravity circuit is much coarser than previous, which in combination with optimization of the gravity circuit has increased overall gravity recovery by ~10%
- At the coarser grind size more efficient screens can be used for closing the circuit instead of cyclones, thereby reducing the over grinding of cassiterite through eliminating SG effect in classification (Cassiterite SG is ~6.5-7 compared with the average SG of about 3-3.5) and further helping gravity performance. Improved classification efficiency in closing the primary grinding circuit is also expected to improve milling efficiency compared to the use of cyclones.
- Reduced the primary mill size/power by approximately a 30%. However, to balance this both the sulfide regrind duty and gravity regrind duty become much larger growing by 20% and 49% respectively, but only on a portion of the total feed. The net result is it effectively gives a redistribution of grinding energy input providing a tin size distribution which is better suited to the process.

Figure 2 shows the effect of primary grind size on the portion of tin passing 75 μm and 40 μm and on tin P_{80} in the sulfide tailings/gravity circuit feed.

Most significantly, increasing primary grind size from the original P_{80} ~ 130 μm to ~ 250 μm decreased the amount of tin passing 40 μm from 52% to <38%, keeping significantly more tin in the readily recoverable size ranges. Referring to Figure 3 it can be seen gravity circuit performance drops off sharply below ~ 40 μm.

Over the size range investigated coarsening primary grind size had only minimal effect on tin losses to the rougher concentrate for a given sulfur rejection target (~2% variation). However, following regrinding and cleaning any differences in tin losses were eliminated.

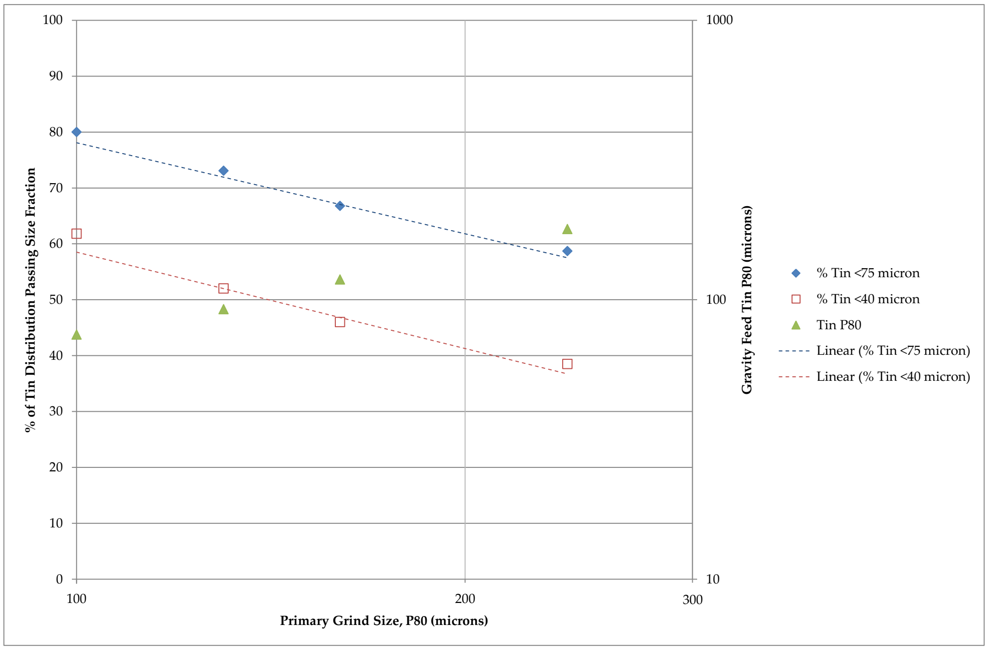

Figure 2 Effect of Primary Grind Size of Tin Size Distribution in Gravity Circuit Feed

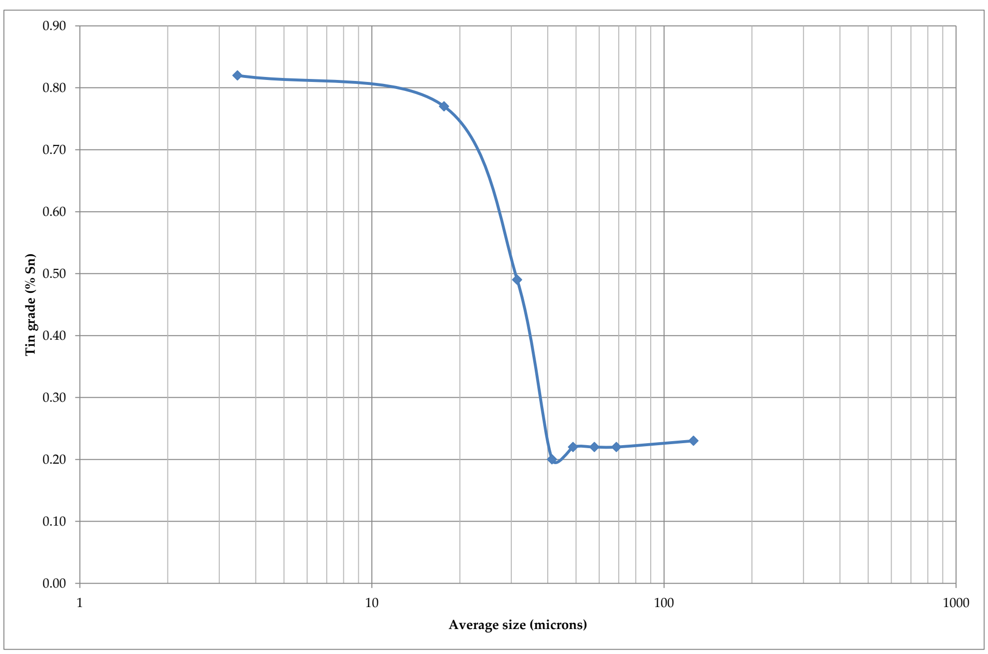

Figure 3 Variation in fine gravity tailings tin grade with particle size

Table 4 compares the overall comminution energy input for two cases;

- Case 1 – the original primary grind size target of P80 ~ 130 μm
- Case 2 – the revised/post optimization primary grind size target; P80 ~ 250 μm

Table 4 Comparison of overall comminution energy inputs

Case		**1**	**2**
ROM Throughput	t/h	75	75
Primary Grinding Duty:			
Primary Grinding Throughput	t/h	75	75
Feed Size (F80)	μm	12,000	12,000
Grind Size (P80)	μm	130	250
Specific Energy Input	kWh/t	16.6	11.5
Gross Power Required	kW	1,370	950
Sulfide Regrind Duty:			
Sulfide Regrind Throughput	t/h	29.3	29.3
Feed Size (F80)	μm	130	240
Regrind Size (P80)	μm	25	25
Specific Energy Input	kWh/t	16.6	20.0
Gross Power Required	kW	540	640
Mass Rejected in Sulfide Flotation	t/h	21	21
Gravity Regrind Duty:			
Gravity Circuit Throughput	t/h	54	54
Feed Size (F80)	μm	131	258
Grind Size (P80)	μm	52	52
Specific Energy Input	kWh/t	10.8	16.1
Gross Power Required	kW	650	960
Total Grinding Energy Required	**kWh/t ROM**	**34.1**	**34.0**
Total Grinding Power Required	kW	2,560	2,550
Sn recovery	**%**	**72.2**	**79.5**
Grinding Energy per unit tin recovered	**kWh/t Sn recovered**	**4,460**	**4,030**
Net Smelter Revenue	**A$/t ROM**	**168**	**185**

On a total energy input per tonne of ore basis (kWh/t ROM) both cases are very comparable despite the quite different specific energy inputs across the three grinding duties. In effect the grinding energy has been redistributed across the circuit to provide a more ideal size distribution to the process.

When the grinding energy input per unit tin recovered is considered, which takes into account the impact of the improved size distribution on overall tin recovery, approximately 10% decrease in total grinding energy per unit metal recovered was achieved with the coarser primary grind size.

Even more significantly, the net smelter revenue is estimated to have improved by A$17/t ROM for the coarser primary grind.

CONCLUSIONS

In this case heavy media separation proved to be an unattractive means of pre-concentration and reducing comminution energy input, fundamentally due to limited liberation of the rejected gangue from the mineral of interest, and modest mass rejections achievable. This highlights the need for detailed assessment on a case by case basis for any pre-concentration methodology. Clearly a one size fits all approach cannot be taken.

The progressive liberation approach taken, achieved reduced overall specific comminution energy input per unit metal recovered, by yielding a steeper size distribution more ideally suited to the size by size recovery response of the overall process flowsheet. This approach reduced losses associated with fines, resulting in a significant gain in overall tin recovery to be achieved.

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